A Feasibility Study on the Use of Desulphurized Tailings to Control Acid Mine Drainage

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\textbf{Key Words:} Desulphurized tailings, acid mine drainage, dry cover, paste backfill, water covers
ABSTRACT

Environmental desulphurisation is an attractive alternative for the management of acid generating tailings. Non selective froth flotation is the most commonly used method to produce desulphurized tailings, as shown from many previous works (McLaughlin and Stuparyk, 1994; Bussière et al., 1995; 1998; Benzaazoua et al., 2000a,b; 2002). This process can reduce the volume of problematic tailings to manage, by making a sulphide fraction concentrate. The desulphurized tailings can then be used as a moisture retention layer in a cover with capillary barrier effects to prevent acid mine drainage.

In this paper, the authors present the results from different laboratory and pilot scale flotation tests which confirm that low sulphide, non-acid generating tailings can be produced by froth flotation. It is also demonstrated that the sulphide concentrates obtained from desulphurisation, when mixed with binders gave good long term results in terms of mechanical behaviour, thus demonstrating that the sulphide fraction can be used for paste backfill material. From previous work (Bussière et al. 1998, 1999) on the evaluation of the operating and capital costs of the desulphurisation process for different ore types, the use of desulphurized tailings to create a dry cover was then studied for several scenarios. The desulphurisation process was compared with other technologies (underwater disposal, engineered soil cover) currently used to prevent acidic drainage (AD) for a new mining operation and the advantages and disadvantages of desulphurisation are presented as well as the costs of this method of AD prevention.

INTRODUCTION

The principal environmental problem of the mining industry today is the generation of acidic drainage (AD) resulting from the natural oxidation of the sulphides in mine tailings. This phenomenon involves the generation of acidity which increase the amounts of toxic metals in the waste water released into the environment by a mine. The environmental impact of the AD has been described in detail in previous work (e.g. Down et Stocks, 1977; Ritcey, 1989; Gray, 1997; Ripley et al., 1996; Aubertin et al., 2002).

Different techniques have been used to limit the release of AD into the environment. Essentially, these techniques try to limit the availability of one or more of the components (water, oxygen, or sulphides) to the others that result in the generation of acidity. To limit the influx of water, one has to manage all elements of the water flux between the surface and the tailings. This can be done either by using low hydraulic conductivity soils or synthetic materials such as geomembranes or bentonite geocomposites. Some Quebec mine sites are currently using this approach for their restoration (Bienvenu and Dufour, 1996; Lewis and Gallinger, 1999).

There are several different techniques available to limit oxygen migration into a mine tailings dump. One can cover the sulphide mine tailings with water (e.g. Fraser and Robertson, 1994; Amyot and Vézina, 1997; Simms et al. 2001), or cover the mine tailings under materials that are oxygen consuming such as wood chips or bark, waste water treatment sludge or the de-inking sludge from a pulp mill waste (e.g. Tremblay, 1994; Tassé et al., 1997; Cabral et al., 2000).
Another alternative is to construct a cover with capillary barrier effects made of either natural materials or non acid generating mine tailings (e.g Nicholson et al., 1989; Aubertin et al., 1995; Ricard et al., 1997; Bussière et al., 2001).

Another approach also proposed to limit the production of AD from mine tailings is environmental desulphurization (McLaughlin and Stuparyk, 1994; Humber, 1995; Bussière et al., 1995, 1998; Stuparyk et al., 1995; Benzaazoua et al., 1999; Benzaazoua et al., 2000a, b, 2002). It consists of separating a sufficient quantity of sulphides from the mine tailings based on its neutralization potential (Figure 1). With this process, there are two different fractions to manage, desulphurized tailings which do not generate AD and a smaller quantity of sulphide enriched tailings which is acid generating. With this approach, it is possible to substantially reduce the volume of acid generating mine tailings.

![Figure 1: Scheme illustrating the environmental desulphurisation principle](image)

This article has two parts. The first part concerns the feasibility of the desulphurisation technique based 1) on a laboratory study to find the first optimization elements of the desulphurization process and 2) on site continuous pilot tests to verify the laboratory results, especially those relevant or similar to actual mill conditions (variations in chemical and mineral composition). The potential to use the sulphide concentrate in a paste backfill is also discussed.

The second part is about the economic factors of the mine tailings management by desulphurisation and comparison to other tailings management and rehabilitation techniques.

**TAILINGS DESULPHURIZATION**

For the laboratory study, the tailings were taken from a slurry sampled at the outlet of the processing plant from a base metal mine. The sulphur, zinc and copper contents were determined by ICP analysis. The chemical composition of the tailings samples and the calculated sulphide composition are presented in Table 1. The primary sulphide mineral in the tailings was pyrite. An important characteristic for desulphurization is the acid generation (or Net Neutralization Potential NNP) of the tailings. From the various methods used to evaluate this parameter, the modified Acid Base Accounting test was chosen because of its consistency from previous testing. The NNP is calculated...
as the difference between the Neutralization Potential (NP) and the Acidity Potential (AP). AP is estimated from the sulphide sulphur content as analyzed by chemical analysis and NP is determined by volumetric titration. The results are summarized in Table 1 and show that all of the tailings studied were acid generating and have a relatively low NP. The tailings grain size was analyzed because this factor is very important for both the flotation processes and sulphide oxidation. One can see from Table 1 the typical tailings grain size distribution as determined by grain size analysis done with a Malvern Mastersizer. Table 1 also shows the specific gravity ($G_s$) of the tailings as determined with a Helium pycnometer; $G_s$ was largely dependant on the sulphide content of the material tested.

Table 1: Mine tailings characteristics

<table>
<thead>
<tr>
<th>ABA static tests for the tailings studied</th>
<th>Main results of the grain size analysis</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ranges</td>
<td>Tailings</td>
</tr>
<tr>
<td>S tot. Wt %</td>
<td>D90 (µm) 76.3</td>
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<tr>
<td>S (sulphide) Wt %</td>
<td>D50 (µm) 17.85</td>
</tr>
<tr>
<td>S (sulphate) Wt %</td>
<td>D10 (µm) 2.09</td>
</tr>
<tr>
<td>AP kg CaCO₃/t</td>
<td>$G_s$ 3.11</td>
</tr>
<tr>
<td>NP kg CaCO₃/t</td>
<td></td>
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<tr>
<td>NNP (total S)</td>
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</table>

Laboratory testing

Flotation testing was done on the tailings samples under laboratory conditions to optimize the desulphurization process. The solids percentage for the sampled pulp was 30%; this is typical of flotation treatment. Conditioning time was 10 minutes, after the addition of the collector and Sasfroth. All of the flotation tests were done in a Denver D-12 lab flotation machine. The cell volume was of 2.5 litres. The speed of the rotor-stator was adjusted to 1500 rpm and airflow was at 2.25 liters per minute. The froths were removed with a spatula by the same operator for all flotation tests to obtain consistent results. For each flotation test, four concentrates and a final desulphurized tailings were recovered separately and analyzed for sulphur grade to study kinetic sulphide recovery.

Optimization test for desulphurisation in Denver cells

The different parameters were optimised to obtain the best total sulphide recovery and to avoid the mechanical trapping of gangue which could cause dilution of the concentrate (increase in volume), except when this dilution was desirable (fraction used in paste backfill). The different optimization steps, for the sampled tailings are listed below and the complete results can be found in the relevant URSTM publications for this subject (Benzaazoua et al. 1998; 1999, 2000, 2002). Here are the main conclusions for each aspect of the optimization tests.

- **Frother type**: alcohol based, Sasfrothe Sc 39 from Prospec Chemicals
In this study, we evaluated several parameters that influence the flotation process of sulphide ores. The parameters include frother concentration, collector type, pH regulator type, conditioning time, pH and redox potential, solid percentage, grain size, and collector concentration and flotation kinetics.

**Frother concentration**: 15 µl/t for ore

**Collector type**: Potassium amyl xanthate, KAX 51 from Prospec Chemicals

**Effect of pH regulator type**: The lime used in the mill process causes pyrite depression, acidification with H2SO4 followed by raising the pH levels with NaOH to raise the alkalinity to allow good output from the pyrite flotation.

**Effect of the conditioning time**: This factor has little influence on the desulphurisation process because the collector (KAX) absorption is essentially instantaneous. The conditioning aids proper pH level regulation.

**pH and redox potential**: The first conditioning step is done under alkaline pH conditions to preserve the neutralization potential of the tailings for desulphurisation. A pH of 11 gives good recovery and suitable dilution for the sulphide concentrate. The pH and the Eh values from a mill circuit used for polymetallic ore (copper, lead and zinc) treatment vary between 10.5 and +50, respectively. However, previous work showed that pyrite doesn’t respond well to flotation at pH levels of between 7 and 8. Therefore we chose a pH level of 6, somewhat aggressive for carbonates, but this pH level does not lower the neutralization potential of the mine tailings to be treated by the desulphurisation process.

**Effect of the solid percentage**: The optimum solids percentage for the best recovery of a specific grade in the concentrate is approximately 45 % solids. However, 25% will do for the flotation tank with frequent rinses and this avoids the need for a supplementary thickening stage.

**Effect of Grain Size**: The fineness of the grain size is one of the most important parameters in sulphide flotation. Our study demonstrated that the collection of grain surfaces is dependant on particle size. We developed models to account for this parameter in the desulphurisation simulation.

**Collector concentration and flotation kinetics**: This study was done in conjunction with the optimization of collector concentrations. In effect, the flotation time and the concentration are the two keys for regulation of the flotation process. Experiments were done to investigate flotation kinetics for the two non-cyanided tailings under the conditions above, using an amyl xanthate collector. The results are shown in Figure 2 where the sulphide recovery vs. time and residual sulphide vs. time are shown for each tailings sample.

The residual sulphide content obtained was approximately 0.4 % and this was obtained with an optimum collector concentration of 100 g/t (Figure 2). Up to a certain level, increasing the KAX dosage didn’t improve the sulphide recovery which stabilized at 97 %. However, a KAX concentration below 40 g/t seems to have no effect on the sulphide flotation. The concentrate weight percentage was approximately 45 %.
Other important parameters were studied to find a solution that was both economic and technically feasible. Among these were grain size, sulphide liberation, surface state, and the conditioning treatment.

Effect of desulphurisation on the neutralization potential

This is a very important factor and should be studied to understand the gradual selectivity of carbonates in a trapping process. Effectively, the selective mechanical trapping process is gradual since the desulphurization of a residue results in reduction of its neutralization potential. The results show that there is no difference in selectivity between the two most significant carbonates, dolomite CaMg(CO$_3$)$_2$ or calcite CaCO$_3$, regardless of their initial grade. The recovery of Ca and Mg, in the concentrates obtained, is directly related to the mechanical trapping process. Since particle entrapment is an integral part of desulphurisation, it is essentially dependant on the flotation time.

In general, the desulphurisation process increases the net neutralization potential (NP-AP) of the treated tailings, compared to their initial state. The removal of the sulphides essentially increases the proportion of neutralizing elements in the treated tailings. Figure 3 shows the difference between the NP of the treated tailings vs. the NP of the non-treated tailings. This diagram accounts for the trapping of gangue in the concentrate.
Figure 3: Relation of the NP of the desulphurized tailings for different values of NP and the sulphide grade of the tailing before desulphurisation. (Dilution of the concentrate fixed at 10% by weight)

Mine site testing

The preliminary laboratory testing allowed the optimization of the desulphurisation process by non-selective flotation. This was done on a reduced scale by using Denver cells. The objective of the pilot tests was to confirm the feasibility of continuous desulphurisation. Figure 4 shows installation of the mobile desulphurisation unit at a polymetallic mill in Canada.

The performance of the desulphurisation circuit was tested for a month. During this period the sulphide content of the mill feed to the mobile unit ranged from 9 to 22 % S. The output of the process was successful in reducing the sulphide levels in the final residue to levels low enough to be non acid generating.

The figure 5 shows the performance when all of the process parameters have been optimised. It shows a single instant in time of the desulphurisation process. 100 g/t of amyl xanthate was used to lower the sulphur grade of the feed from 20 to 0.5 % in the desulphurized tailings. One should note that the concentrate has high sulphur grades which shows that there was very little entrainment of gangue mineral (approximately 19 %).
The on site pilot tests, using the mobile desulphurisation unit, allowed us to improve our knowledge of environmental desulphurisation. The tests were done to verify that the desulphurisation results in the laboratory could be obtained at the pilot scale. The sulphide grade of the desulphurized tailings was very low and the sulphide grade of the concentrates was very high. This confirmed that tailings can be desulphurized using the pilot plant and the tailings feed from an operating mill. The mobile desulphurisation unit is a viable tool to process significant amount of pulp under typical mill conditions.
Concentrate as paste backfill

The potential use of sulphide-rich tailings (flotation concentrates) as paste backfill was studied by Benzaazoua et al. (2003). The amount of sulphides in the tailings has a direct impact on the tailings density and consequently, on the quantity of binder to be added per volume unit (proportions of added binder are always calculated using the dry total mass of tailings). Also, there is an indirect effect that corresponds to the amount of sulphates in the initial mixture that is proportional to the amount of sulphides in the tailings. Therefore, to isolate this parameter, it was necessary to do a series of tests that would both deplete and enrich the sulphides in the mine tailings sample. The end result was tailings mixtures with five different compositions: S1, S2, S3, S4 and S5 (concentrate) with sulphur contents of approximately 2 % (desulphurized tailings), 6 %, 12 %, 18 % and 38.5 % of sulphur respectively. The particle size distributions for the four different tailings were very similar. Two binder types were tested that had the same proportion (4.5 wt %) of binder. The first binder was 50 % Portland cement T10 and 50 % Portland cement T50 and the second was 80 % blast furnace slag and 20 % ordinary Portland cement T10.

The results from the uniaxial compressive strength (UCS) evolution and its relationship to the sulphide content are shown in Figure 6. In addition to the clear effect of binder type, one can note that the amount of sulphides had an influence on the strength (UCS) of the paste fill. However, the influence of the amount of sulphides amount was dependant on the binder type. Samples with Portland cement (T10-T50) generally had less mechanical resistance. However, its positive effect was more noticeable for the binder with slag (T10-Slag) which seemed to generate better resistance after 28 days of curing time. It should be noted that, regardless of the
binder type, the sulphide proportion in the tailings adversely affects the pastefill strength up to a sulphide content of 12%. With higher sulphide contents, there is a beneficial effect, which could be related to one or the two following factors:
- Higher sulphide content implies higher density and this leads to a higher proportion (of volume) of the binder
- Higher sulphide content implies higher sulphate available. Consequently, this leads to precipitation of the sulphates, which aids the cohesion development.

However, to understand the sulphate production behaviour within pastefill systems and to demonstrate the low reactivity of sulphides within the cemented pastefill, we did oxygen consumption tests (Eberling et al., 1994) on the pastefill samples for both of the binders. In addition, free binder samples (wet tailings only) were also studied in terms of their sulphide reactivity and for later comparison with pastefill reactivity. The results of this study are presented in detail in Benzaazoua et al. (2003) and Ouellet et al. (2003). It has been demonstrated that sulphide reactivity is limited in pastefill. This is largely due to high water saturation levels (near 100%) in the backfill. This confirms the assumption that the sulphates affecting mechanical strength are mainly those that were present in the initial mix (pre-oxidized products).

![Figure 6: Effect of sulphur content on the mechanical strength (UCS) of pastefill samples with time for two different types of binder: Cement Portland (T10-T50) and Slag based cement (T10-Slag). (Benzaazoua et al. 2003)](image)

**COST EVALUATION OF THE DESULPHURIZED TAILINGS TECHNOLOGY**

To assess the cost of implementing desulphurized tailings technology, a comparative cost study was done by SNC-Lavalin Environment. The objective of the study was to evaluate and compare the cost of tailings disposal ($/m^3$ of tailings) for a new mine with potentially acid generating tailings, using four different techniques to prevent the onset of acid drainage: 1) a dry cover with capillary barrier effect (CCBE), 2) underwater disposal, 3) complete desulphurisation of all tailings generated and 4) partial desulphurization, i.e. production of depyritized tailings during
the last years of mine operation for a dry cover. Two different site locations were studied, one with a favourable topography (valley site) and one with a less favourable topography (flat site).

**Characteristics of the mine site used for the comparative cost study**

In order to compare the tailings disposal cost of the four tailings disposal techniques (CCBE, underwater disposal, complete and partial desulphurisation), a virtual mine site was made for this study. The base metal mine was an underground operation, located in a wet (positive water balance) climate. The new mine site characteristics used for the comparative cost study are presented on Table 2.

<table>
<thead>
<tr>
<th>Table 2: Mine site characteristics</th>
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<tbody>
<tr>
<td>Ore reserves</td>
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<tr>
<td>Ore Grades</td>
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<td></td>
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<tr>
<td>Tailings Sulphide content</td>
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<tr>
<td>Neutralisation potential</td>
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<tr>
<td>Mill capacity</td>
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<tr>
<td>Mining method</td>
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<td>Backfill</td>
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</table>

**Leachate of tailings and design criteria**

A sampling and testing program done by the Ministry of Environment of Québec (Spiegle, 2002) demonstrated that the majority of non-acid generating tailings tested are leachable under the new *Directive 019 Guidelines* for tailings management for the Province of Quebec, using USEPA 1311 or 1312 leaching test method. So, for this study, it was assumed that the tailings would be leachable, i.e., that the metal concentrations in the leachate of a tailings sample are higher than the provincial limits. This hypothesis was also used for desulphurized tailings. Consequently, for all of the different scenarios studied, the tailings dams were designed with low permeability cores to reduce tailings pore water seepage. This hypothesis is conservative as some desulphurized tailings will not be leachable or because under some of the regulations, the tailings dams could be constructed with tailings or a more permeable material, which would significantly reduce dam construction costs. This would reduce the overall cost for the tailings management and rehabilitation techniques as such as the use of a CCBE or desulphurisation.

**Tailings disposal sites**

To account for the dam construction costs for four different tailings management scenarios, two different tailings locations were studied. The first site was located in a valley (see Figure 7). This site needed the construction of two dams to close the valley and create a tailings impoundment. The stratigraphy was variable and was typical of the soil conditions from Northern Québec, with a resistant silty clay layer at lower elevations and glacial till on the valley slopes. For this site, it
was assumed that a clay layer or a geomembrane would have to be installed on 20% of the area, to reduce seepage and limit groundwater contamination where more permeable material is present.

The second site was located in a flat area and needed the construction of four dams to confine the tailings (see Figure 8). Since the area is flat, it is assumed that the stratigraphy was characterized by 3 to 10 m of soft clay (20 Kpa of shear strength) and that the low permeability clay would meet the regulatory requirement for groundwater protection.

For the two sites, a low permeability dam concept was developed, with a clay core. The abutments were made of sand and gravel or till, with a filter chimney and horizontal finger drains on the downstream side and an erosion protection layer (rip-rap) on the upstream and downstream slopes. Berms were included in the design when necessary to improve stability.

Figure 7: Site #1: Valley type

By using construction costs from recently built tailings dams in Quebec and Ontario and also from previous work (Bussière et al. 1998, 1999), the relationship between the construction cost per linear meter of dam and the height of the dam was established. For a tailings dam 10m high, the construction cost was estimated to be from $2 500 to $2 600 per linear meter for a dam with no berm, depending on the distance to the different construction materials. When berms were required, the cost rises to $3 800 to $4 200 per linear meter.
Tailings management scenarios

As mentioned previously, four different tailings management and rehabilitation scenarios were considered. All of the scenarios are designed to prevent sulphide oxidation and to minimize the seepage of the leachable tailings. The four different tailings management techniques considered for this comparative cost study are 1) a dry cover with capillary barrier effect (CCBE), 2) underwater disposal, 3) complete desulphurisation of all tailings generated and 4) partial tailings desulphurisation.

CCBE

The capillary barrier concept has been used to rehabilitate two acid generating tailings ponds in Quebec, Les Terrains Aurifères and Mine Lorraine sites (e.g. McMullen et al., 1997; Ricard et al., 1997, 1999; Dagenais et al, 2002). The ability of a CCBE cover to function is due the fact that the cover acts as a moisture retention structure, which shields the tailings from atmospheric oxygen as well as allowing construction of thinner layers, which reduces the cost of construction. The moisture retention capability exists because the composite cover system ensures that the low permeability (fine) material placed between two coarse material layers remains saturated almost indefinitely. Since the coarse material layer underneath does not easily retain water, its unsaturated hydraulic conductivity is low. Consequently, the fine material layer in the middle remains saturated, since water is attracted to this layer by greater capillary forces, and because the downward flow is greatly reduced due to the low unsaturated hydraulic conductivity of the coarse layer. It is assumed that the sulphides will not oxidize during the deposition of the tailings and will remain saturated. More details on CCBE can be found in the literature (e.g. Nicholson et al. 1989; Aubertin et al. 1995; Bussière et al. 2003).
For this scenario, different cover designs were considered, with total cover thickness ranging from 1.1 m to 1.5 m. The sand and gravel layers (top and bottom of the cover) ranged from 0.3 to 0.5 m and the silt layer (middle layer) ranged from 0.3 to 0.4 m. It was assumed that silt could be found within 1.5 km of the tailings disposal site and that a sand and gravel pit was located within 1.5 to 7.0 km of the site for purpose of feasibility analysis. The cost of $1500 per hectare for lime addition to the tailings, $5000 per hectare for monitoring and $7000 for revegetation was also included.

This concept becomes less feasible when the construction materials are not located nearby. The construction of the cover is also easier during the winter, when machinery can move on the frozen tailings (Ricard et al. 1997). In warmer climates, the construction of a CCBE becomes more expensive. However, this concept requires lower dams than the under water disposal technique, since the cover can follow the natural deposition slope contours of the tailings.

Water covers

This tailings management technique refers to depositing the tailings under a water cover. Research has demonstrated that mine tailings oxidation is reduced to low values (usually between 1 to 5 mole of O₂/m²/an (Li et al. 2000)) by placing them under a water cover (MEND, 2001). Fresh tailings are disposed of within the confines of an engineered tailings management facility, also referred to as a man-made tailings impoundment, and kept submerged. For the cost evaluation study, the depth of the water cover ranged from 0.8 to 1.2 m for the different scenarios. As shown by some operators (Talbot and Cayouette, 2003) the cost of tailings placement is significant and had been considered in the model. The rehabilitation costs of a water cover were considered to be minimal and an amount of $150 per linear meter of dam had was used.

It appears from the simulation that water covers represent one of the less expensive methods of tailings management. However, a water cover can present an increased risk level from the possibility of geotechnical containment structures failure.

Desulphurisation of all tailings (complete desulphurisation)

This technology consists of separating the sulphide fraction of the tailings using froth flotation. Through this process, a high sulphur fraction (acid generating) is produced, which can be managed more easily due to their reduced volume. The tailings impoundment will be the depository for all low-sulphur tailings in excess of that used for backfill. Since the desulphurized tailings are not potentially acid generating, there is no requirement to keep them submerged or saturated. At mine closure, the tailings will be revegetated for permanent reclamation. For this study, it was assumed that the entire sulphide fraction was used in the paste backfill. Depending of the topography of the site and the tailings management technique, the sulphide rich material could also be disposed in the tailings pond, as long as it remains saturated. It has been shown (Benzaazoua et Bussière 1999; Benzaazoua et al. 2000) that the cost of depyritization varies with the neutralisation potential (NP) of the tailings, the higher the NP the lower the desulphurisation
cost. Different scenarios were studied for NP ranging from 40 kg CaCO$_3$/t to 140 kg CaCO$_3$/t.

As previously shown in this paper, the sulphide rich tailings can be successfully used in paste backfill preparation. The production of low sulphur tailings has been performed in the laboratory by the researchers at the University of Quebec in Abitibi Temiscamingue (UQAT-URSTM) (Bussière et al. 1998; Benzaazoua et al. 1999, 2000), at INCO’s Clarabelle Mill (McLaughlin, Robertson, 1994) and also at different Falconbridge sites, including the “New Tailings Area” in Sudbury.

The main advantage of this technique comes from the fact that non-acid generating tailings pond does not represent a long term liability. In some locations, the use of this technology can help in obtaining the mine permits.

Desulphurisation of a fraction of the tailings (partial desulphurization)

Under this scenario, the tailings are desulphurized only during the last years of mine operation. The desulphurized tailings are used to cover the potentially acid generating (PAG) tailings with an inert dry cover. The cover made of depyritized tailings is designed to maintain an elevated water table, which will keep the PAG tailings deposited during the early years of operation in a saturated condition which will prevent sulphide oxidation. Since the foundation and the tailings dams are characterized by very low permeability, an elevated water table can be maintained in the PAG tailings with a layer of 1 to 2 meters of desulphurized tailings. The PAG tailings will act as an oxygen barrier by limiting oxygen diffusion by this high degree of saturation in the tailings. This approach is a variation of the previous scenario and requires the same capital investment in a flotation circuit at the end of the milling circuit, but has lower operating costs, since only a fraction of the tailings need to be desulphurized.

Results from the cost study analysis

Table 3 (a-b-c) and Figure 9 show the tailings disposal cost, expressed as $/mt of tailings, for the four different tailings disposal techniques that were considered.
Table 3 (a-b-c): Results of the cost study analysis (in $/m.t. of tailings)

- A -

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<thead>
<tr>
<th>Valley site</th>
<th>Minimum</th>
<th>Maximum</th>
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<tbody>
<tr>
<td>Underwater disposal</td>
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<td>0.89 $</td>
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<tr>
<td>Partial desulphurisation</td>
<td>0.86 $</td>
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<td>Complete desulphurisation</td>
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<tr>
<td>CCBE</td>
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- B -

<table>
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<td>CCBE</td>
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<td>1.54 $</td>
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</tbody>
</table>

Figure 9: Tailings disposal cost comparison

Considering that the models and design hypothesis used for this study are representative of a majority of existing sites and can be applied to future mining projects, the results from this study show that:
• Partial desulphurisation of tailings presents saving opportunities especially when the tailings pond location is characterized by flat terrain and a soft foundation, which increases the dam construction cost.

• Under water disposal is the most economical option, when topography of the site and foundation conditions are favourable. However, the cost difference between this option and the partial desulphurisation option is very small, especially for tailings with low neutralisation potentials.

• The construction cost of a CCBE can vary significantly from one site to another, depending on the proximity of the construction materials.

• Complete desulphurisation of tailings remains an viable economic option when it becomes expensive to build low permeability tailings dams.

• There is a wide range of field conditions where the four different scenarios are comparable from an economic point of view.

We should remember that those results are valid for new operations only. Rehabilitation of existing tailings impoundment by water cover may not have the same economic appeal.

We should also remember than this study does not compare the risk levels associated with each scenario. Different mine operators may not have the same risk tolerance. Some may prefer the CCBE approach to avoid the risks associated with the management and maintenance of a large water dam. Others may not be ready to use a new approach such as desulphurisation or may prefer complete desulphurisation to get rid of any tailings sulphide contents on the surface.

**Conclusions**

This project shows, through laboratory tests, that the desulphurisation process is technically feasible. Continuous in-situ testing on a pilot scale demonstrated that good sulphide recoveries can be obtained without affecting the initial neutralization potential of the tailings. The concentrate, even if sulphide-rich, can be used to produce high quality backfill.

A cost study was performed to evaluate the cost of different tailings management techniques for potentially acid generating tailings. While underwater disposal is an attractive method when the site topography is suitable, desulphurisation (total or partial) can also be economically viable under different conditions. For a new mine, the construction of a CCBE at the end of mine-life is more expensive unless suitable construction materials can be found close to the mine site.

The desulphurization techniques, particularly when combined with paste backfill, should be considered as an integrated management technique throughout the mine life, facilitating the rehabilitation at the end of mine life. Desulphurization may facilitate permitting of new mine projects, particularly in regions with restrictive environmental regulations.
The comparison of tailings management methods presented in this work is based solely on economic considerations. Risk aspects and long term liabilities were not taken into account. This study open the door for further work on Risk analysis, Life cycle analysis and development of better integration between the upstream metallurgical process for value recovery and the downstream desulphurisation process for environmental purposes.

REFERENCES

BUSSIÈRE, B., LELIÈVRE, J., OUELLET, J. and BOIS, D. “Utilisation de résidus miniers désulfurés
comme recouvrement pour prévenir le DMA : analyse technico-économique sur deux cas réels”, *Sudbury '95, Conference on Mining and the Environment*, vol. 1 : 59-68. 1995


RITCEY, G.M. *Tailings Management, Problems and Solutions in the Mining Industries*. Elsevier. 1989

SPIEGLE, T., 2002, personal conversation.


