THE ECONOMIC VIABILITY OF PROCESSING TAILINGS TO REDUCE ENVIRONMENTAL LIABILITY

by

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ABSTRACT

Many mining operations produce tailings that contain sulphides. Sulphidic tailings are disposed into a tailings facility and measures are taken to control the release of deleterious seepage resulting from the oxidation of sulphide materials. Often this control needs to be maintained in the long term, which presents a long term environmental risk. This thesis examines the viability of processing tailings to reduce the long term environmental risk of tailings impoundments.

A spreadsheet model was developed to calculate project life costs of two tailings disposal methods – conventional disposal and disposal of autoclaved tailings. A generic mine site in British Columbia and assumed operating parameters were used as a basis for the model. Unit operation designs for the two flowsheets were based on recently designed or constructed equipment or on accepted design methodology. Capital costs and unit operating costs were obtained from recently completed pre-feasibility studies. Monte Carlo simulations were run while varying selected parameters to derive project costs for a range of situations.

The simulation results indicate that for smaller operations where the processing rate is less than 5000 tonnes per day, the mine life is less than 12 years and the sulphur content is less than 12% it may be economically viable to autoclave tailings in order to produce material that would be more geochemically stable in the long term. The combination of parameter values at which autoclaving tailings is economically viable follows a curve as sulphur content decreases in conjunction with processing rate and mine life increases. The economics of autoclaving tailings is sensitive to the amount of solids reporting to the autoclave.

TABLE OF CONTENTS

ABSTRACT ii
TABLE OF CONTENTS iii
LIST OF TABLES iv
LIST OF FIGURES v
1. INTRODUCTION 1
1.1 Objective 1
1.2 Background 1
1.3 Methodology 9
1.4 Assumptions 10
1.5 Thesis Layout 10
2. THE MODEL 11
2.1 Input Parameters 13
2.2 Sulphide Separation 16
2.3 Sulphide Oxidation 22
2.4 Water Treatment 26
2.5 Tailings Facility 33
2.5.1 Tailings Characteristics 33
2.5.2 Facility Design 36
2.6 Tailings Facility Closure 40
2.7 Security and Bonding 42
2.8 Project Cost Summary 46
3. SENSITIVITY ANALYSES 50
3.1 Generic Simulations 50
3.2 Case Studies 62
4. DISCUSSION 64
4.1 Model Results 64
4.2 Model Construction 65
5. CONCLUSIONS 69
6. RECOMMENDATIONS FOR FURTHER STUDY 70
7. REFERENCES 73
APPENDIX A Example Calculation of an HDS Water Treatment Plant Design Using Unpublished Data by Humber (1996) 81
APPENDIX B Example Calculation of a Simple Lime Neutralization Water Treatment Plant Design Using Unpublished Data by Humber (1996) 90
APPENDIX C Dam Height Calculation Macro 98
APPENDIX D Simulation Results 102

LIST OF TABLES

Table 1.1 Security Posted Over Time at Sixteen British Columbia Mines 3
Table 1.2 Base Case Parameters 10
Table 2.1 Summary of Flotation Test Results 17
Table 2.2 Sulphur Content of Desulphurized Tailings in the Model................................ 18
Table 2.3 Assumptions for Flotation 19
Table 2.4 Retention Times for the Autoclave Circuit Components 24
Table 2.5 Look-Up Table for Sizing HDS Water Treatment Plants................................ 28
Table 2.6 Look-Up Table for Sizing Simple Lime Neutralization Water Treatment Plants 32
Table 2.7 Calculated Acid Base Account for Flotation Tailings – Base Case 35
Table 2.8 Assumptions for Tailings Management Facility Design 37
Table 2.9 Sources for Cost Data 47
Table 2.10 Project Cost Summary Sheet – Conventional Case 48
Table 2.11 Project Cost Summary Sheet – Autoclaved Tailings Alternative.................. 49
Table 3.1 Scenario Definitions 62
Table 3.2 Total Cost of the Autoclaved and Conventional Alternatives in Defined Scenarios 62

LIST OF FIGURES

Figure 1.1 Security Values over Time at Selected British Columbia Mines 5
Figure 2.1 Flowsheet for Conventional Tailings Disposal 12
Figure 2.2 Flowsheet for Processing Tailings 14
Figure 2.3 Input Parameters Sheet 15
Figure 2.4 Sulphide Separation Sheet 21
Figure 2.5 Sulphide Oxidation Sheet 25
Figure 2.6 Water Treatment Sheet – Conventional Tailings 29
Figure 2.7 Water Treatment Sheet – Autoclaved Tailings Alternative 31
Figure 2.8 Tailings Facility Sizing Sheet – Conventional Tailings 38
Figure 2.9 Tailings Facility Sizing Sheet – Autoclaved Tailings 39
Figure 2.10 Security and Bond Calculation Sheet – Conventional Tailings 44
Figure 2.11 Security and Bond Calculation Sheet – Autoclaved Tailings 45
Figure 3.1 Capital Cost ($millions) vs. Mill Tonnage 53
Figure 3.2 Total Cost ($millions) vs. Mill Tonnage 54
Figure 3.3 Total Cost ($millions) vs. Mine Life 55
Figure 3.4 Total Cost ($millions) vs. Sulphur Content 56
Figure 3.5 Total Cost ($millions) vs. Mill Tonnage 58
Figure 3.6 Total Cost ($millions) vs. Mine Life 59
Figure 3.7 Total Cost ($millions) vs. Sulphur Content 60
Figure 3.8 Sulphur Content vs. Mill Tonnage for Simulations where the Total Cost for the Autoclaved Alternative is Less Than the Conventional Alternative 61

1. INTRODUCTION

1.1 Objective

The idea of processing tailings as a means of improving the environmental performance of tailings typically elicits the response "it is too expensive!" This thesis calculates and compares the mine life cost of a conventional tailings disposal system and a selected processed tailings system to determine if there are situations in which processing tailings is an economically viable way of reducing the long term environmental liability of a tailings impoundment.

1.2 Background

Many mining operations, particularly base metal mines, produce waste materials containing sulphides. Most of these sulphides will react to produce a low pH, metal enriched drainage that can contaminate watersheds if allowed to enter the receiving environment. This acidic drainage is one of the most significant environmental issues the mining industry must address (Tremblay, 2000). The Intergovernmental Working Group estimated that the liability for acidic drainage at mine sites in Canada is approximately $5.2 billion (MEMPR, 1995). Mine sites that have tailings facilities that are currently producing acidic drainage include Mt. Nansen (Yukon), Duthie (British Columbia), Faro (Yukon), Kam Kotia (Ontario), and Poirier (Quebec). To date, only Faro is treating water. Tailings facilities at numerous mines, such as Heath Steele, could produce acidic drainage if current control strategies were not maintained. Seepage from the tailings dam at the Poirier site contains 38,600 mg/L SO4, 20 mg/L Zn, 1.4 mg/L Cu and 17,300 mg/L Fe at pH 3.2 (Lewis *et. al.*, 2000). It is reported that surface water quality is affected for more than 21 km downstream from the Poirier site. Clearly, acid rock drainage control in tailings facilities is necessary to protect the environment.

The financial consequences of acidic drainage and the growing importance of adequate reclamation funding can be seen in the security bonds levied against British Columbia mines. In a security policy discussion paper (MEMPR, 1995), the Government states that its approach to reclamation is to set broad objectives and then negotiate mine-specific requirements. This approach enables the Government to address a property's unique features. This flexibility is particularly important when acidic drainage is an issue at a mine site. The policy discussion paper also states that the Province will be requiring full security prior to a mine's closure to provide reasonable assurance that all reasonably foreseeable reclamation activities are covered. This policy is reflected in the security bonds posted for sixteen British Columbia mines over time (see Table 1.1). The change in security values over time are presented in Figure 1.1 for selected mines.

Table 1.1
Security Posted Over Time at Sixteen British Columbia Mines
(In $Millions)

Year	Kemess	Huckleberry	Gibraltar	Myra Falls	Sullivan	Island Copper	Highland Valley Copper	Table Mountain
Permit #	M-206	M-203	M-40	M-26	M-74	M-9	M-11	M-127
1970						0.11		
1971					0.1			
1976								
1978			0.335					
1980						0.2		
1983								
1984							0.65	
1989								
1990								
1991								
1992								
1993								0.0035
1994			14					0.11
1995					4.11			
1996					6.11			
1997	12	2		10.8	8.11			
1998					9.11		10.25	
1999			29.5		10.11	4		
2000					11.11			
2001					12.11			
2002					13.11			
Current Value	12	2	29.5	10.8	13.11	4	10.25	0.11

Source: Reclamation permit.

Table 1.1 (cont'd)

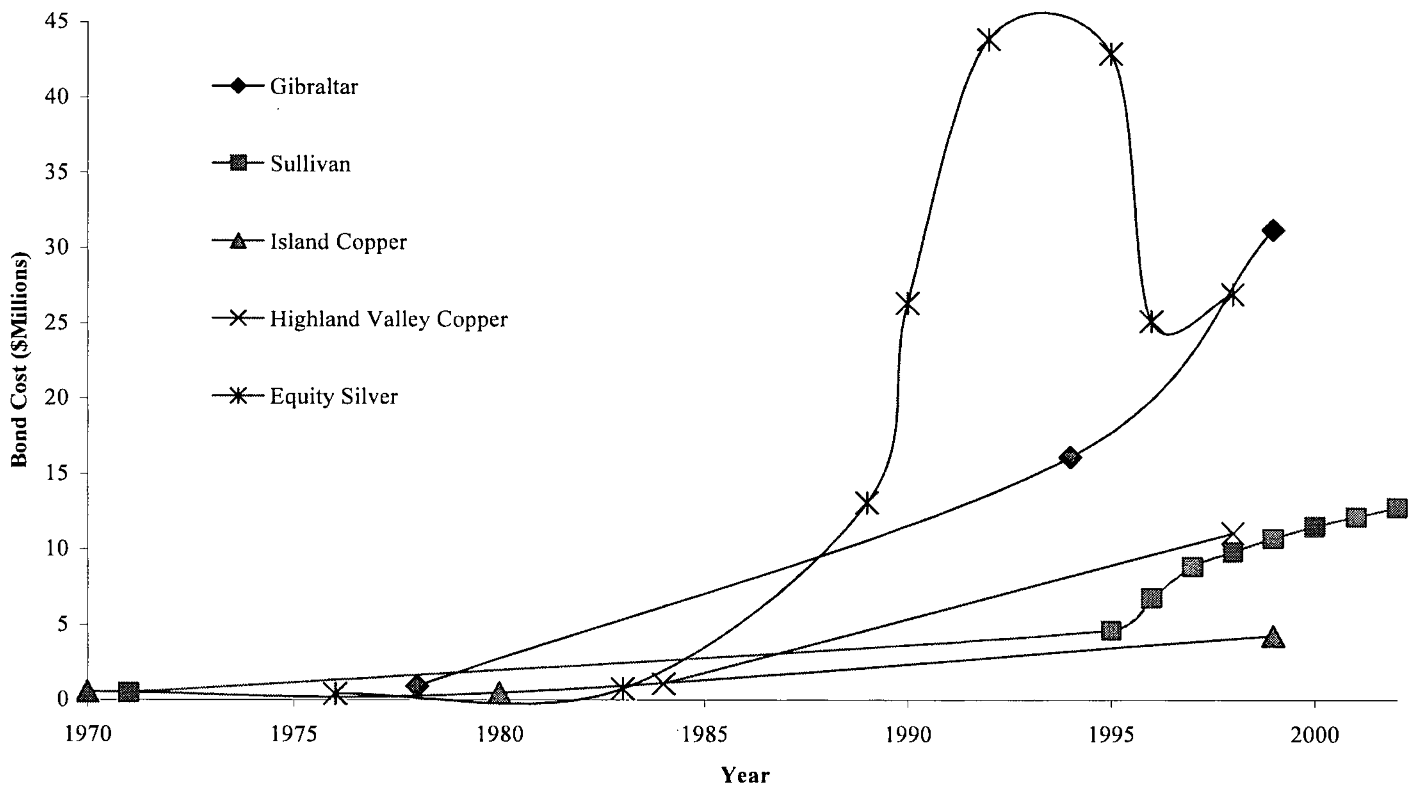
Security Posted Over Time at Sixteen British Columbia Mines

(In $Millions)

Year / Permit #	Golden Bear M-187	Snip M-190	Eskay Creek M-197	Endako M-4	Mt. Polley M-200	Blackdome M-171	Equity Silver M-114	Premier Gold M-179
1970								
1971								
1976							0.125	
1978								
1980								
1983							0.425	
1984								
1989							10	
1990							21	
1991		1						
1992							37.5	
1993								
1994			3.7					
1995					1.15		38.3	
1996							22.7	
1997					1.9	0.1		3
1998							25	
1999				6				
2000	1.545							
2001			3.774					
2002								
Current Value	1.545	1	3.774	6	1.9	0.1	25	3

Source: Reclamation permit.

Figure 1.1
Security Values over Time at Selected British Columbia Mines
(Constant 2001 dollars)



Security is required before the construction of a new mine. The amount of security is based on the reclamation liability created by the mine design. Reclamation typically includes the decommissioning and demolition of all structures, burying foundations, decommissioning roads, resloping waste rock piles, restricting access to underground mine workings or open pits, stabilizing and covering tailings impoundments, and revegetating all disturbed areas where practical (pit walls generally are not required to be revegetated). Cost estimates for these physical works are predictable (the number, size and shape of structures and waste rock dumps are known and contractors will give cost estimates) and generally straightforward.

The challenge in requiring full security is estimating the liability associated with acidic drainage. As a rule, mines do not go into production expecting an acidic drainage problem. Reclamation plans and security bonds are established with the understanding that potential sulphide oxidation will be mitigated and managed such that acidic drainage is not created. Difficulties arise when acidic drainage develops after operations begin. Costs for constructing a water treatment plant are estimated easily enough. The challenge lies in estimating the annual operating costs of the plant, namely predicting the amount of acidity that will be generated, which in turn dictates the amount of lime that will be consumed. It is with this issue that the Equity Silver Technical Committee had to grapple for several years (Equity Silver, 1996). In the end, lime consumption predictions had to be revised after several years of monitoring data was available. The uncertainty in estimating long term water treatment costs at the Equity Silver mine can be seen in the up and down security value shown in Figure 1.1. The initial security value in Figure 1.1 reflects the policy of the day, namely a fixed dollar amount per disturbed hectare. In the mid 1980s acidic drainage developed at the site and the government realized that the posted security would be insufficient to address the issue should the company abandon the site. Negotiations began between the government and the company, with extensive discussion focused on the rate of acidity generation, and consequently lime consumption and long-term water treatment plant operating costs. Water treatment plant data at the time indicated an increasing rate of lime consumption. Therefore, increasing amounts of security were set for the mine, as shown in Figure 1.1. In the mid 1990s, data indicated a leveling off and decrease in lime consumption. The security value was decreased to reflect this change.

The Mine Environment Neutral Drainage (MEND) program was initiated by the federal and provincial governments and the mining industry of Canada in 1988 to co-ordinate research in order to reduce the liabilities associated with acidic rock drainage. Research is in the mechanisms of sulphide oxidation, potential methods for treating existing acidic drainage sites and methods to prevent the formation of acidic drainage. Tremblay (2000) summarizes the results and observations from the MEND program. One of the results is that prevention is the best strategy. Once sulphides start to react and to produce contaminated runoff, the reaction is very difficult to stop.

Prevention methods have focused on the isolation of sulphide minerals from oxygen and/or moisture. This has resulted in the development of engineered covers and subaqueous or underwater disposal. Another approach to prevention has been desulphurization and separate disposal and management of the sulphide concentrate. These methods have shown to be effective when designed correctly; however, the sulphide material remains in the tailings impoundment and poses a risk to the environment if any of the remedial/containment measures fail.

This thesis explores a slightly different interpretation of prevention - that is the prevention of sulphides from entering a tailings impoundment. Remove the primary material that produces acidic drainage and the potential for producing acidic drainage is removed, thereby reducing the long-term liability of the mine. One method of prevention is to sufficiently desulphurize tailings to produce a net neutral product for disposal. The sulphide concentrate can then be oxidized in an autoclave to produce hematite (Fe_2O_3) and sulphuric acid, which is neutralized to produce gypsum and metal hydroxide sludge. The end result is a material that is stable in the physico-chemical environment of a tailings impoundment. With the tailings being inert, the tailings impoundment will not need to be lined and only a simple cover will be required at closure. It is argued that this method of tailings disposal can be comparable in cost to a more conventional tailings disposal method, but have a lower environmental risk, when whole project costs are considered.

There are a number of methods to oxidize or isolate a sulphide concentrate, including roasting, bio-oxidation and encapsulation (in cement, cement derivatives and bitumen). Autoclaving was arbitrarily selected for this thesis based on the reasonably well understood technology.

1.3 Methodology

A spreadsheet model was developed to calculate scoping level designs and the associated costs of two tailings disposal alternatives – conventional tailings disposal and autoclaved tailings disposal. Analysis was made on a generic open pit mine located in British Columbia using assumed operating parameters. The effect of these assumptions on the cost estimate were evaluated by conducting simulations in which input parameters are randomly varied within reasonable lower and upper bound values. In this fashion the situations in which processing tailings are a reasonable alternative are better defined.

Unit operation designs were based on the design parameters of equipment recently designed and constructed or on accepted design methodology. Estimated costs were obtained from recently completed pre-feasibility studies. For simplicity, capital expenditures were assumed to take place in one year at the beginning of the operation although in reality some of these expenditures may be staggered. Security deposits were also assumed to be made in one year at the beginning. It was assumed that the security deposit is a one time expense covering the cost of closure. In reality, the security deposit is returned to the proponent as reclamation and closure is completed. This assumption was made for the sake of simplicity. Post-closure bonds were assumed to be posted two years prior to closure.

The evaluation of the viability of tailings processing was based on the comparison of the costs between a conventional tailings disposal method and the autoclaved tailings method. It was assumed that the two mine scenarios are identical in every way except for the tailings disposal method. Therefore, only those items associated with tailings disposal are costed in the model. While it can be argued that some cost numbers may be imprecise, all items are costed to the same level of accuracy. The value of the model is not in the absolute numbers but in the comparison of the subtotal cost of the two tailings disposal methods.

1.4 Assumptions

Initial design and costing were based on a number of assumed operating parameters, selected from the range published in Mining Sourcebook (1998). Table 1.2 summarizes these assumptions.

Table 1.2
Base Case Parameters

Parameter	Unit	Value
Milling rate	tonnes/day	2500
Mine life	years	20
Operating days	days/year	344
Sulphide in tailings	%	5
Neutralizing potential of tailings	kg $CaCO_3$ eq./tonne	100

1.5 Thesis Layout

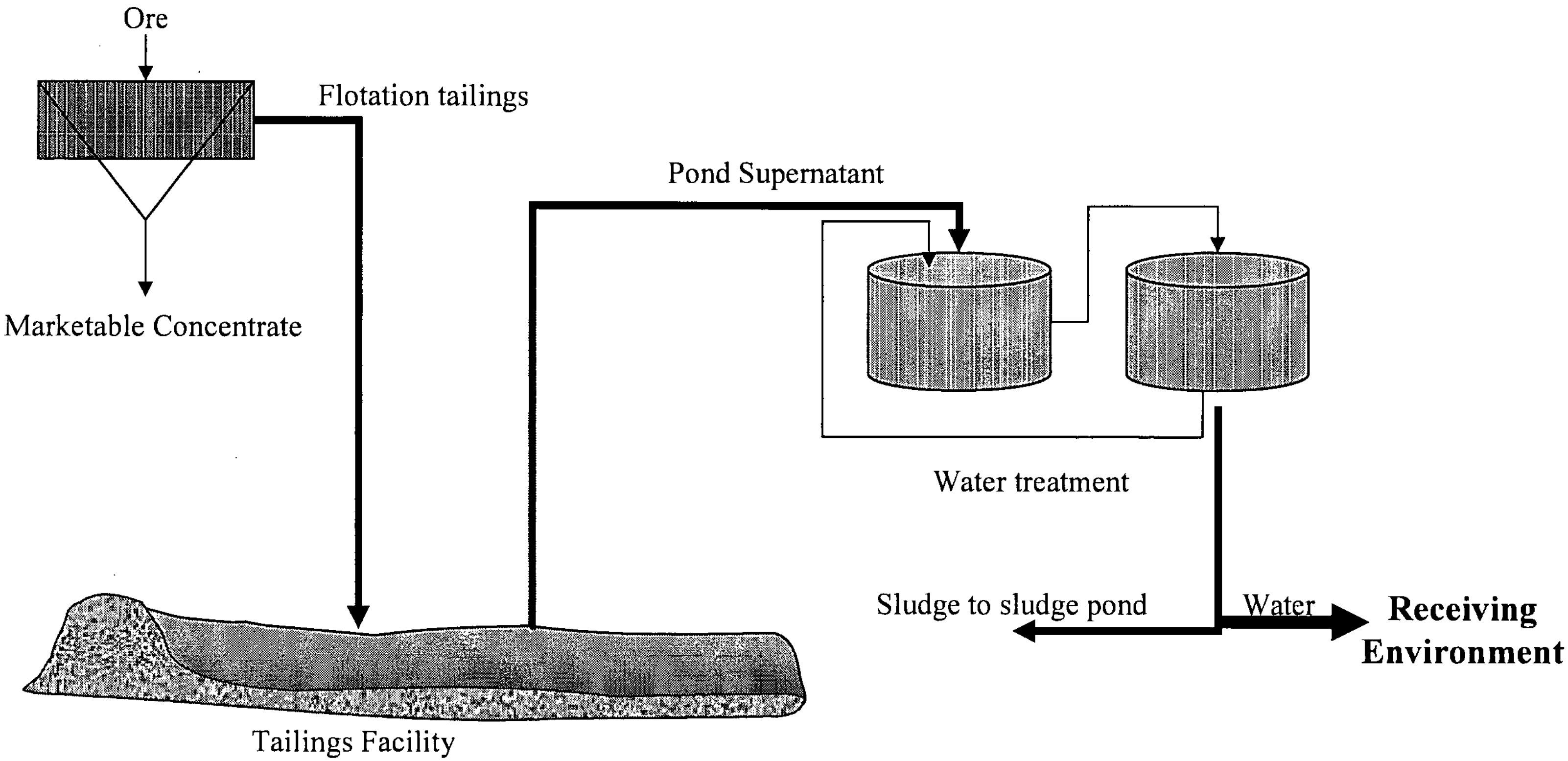
This thesis is organized into five main parts. The model is described in Section 2. Section 3 presents the simulation results. Section 4 discusses the simulation results and the construction of the model. Conclusions and recommendations are given in Sections 5 and 6, respectively.

2. THE MODEL

The model is an Excel workbook containing one worksheet for each unit operation. Calculations for conventional and processed tailings disposal are carried out simultaneously.

Conventional tailings disposal, typical at existing mine sites, is simply pumping the tailings to an impoundment after the marketable minerals have been recovered. Frequently the tailings supernatant is treated, with the water recycled to the mill and/or discharged to the receiving environment. The flowsheet for this process is shown in Figure 2.1. The pertinent issue associated with this disposal alternative is the geochemical behaviour of the tailings material and its influence on the tailings management facility and closure design requirements to safeguard the receiving environment.

Figure 2.1
Flowsheet for Conventional Tailings Disposal



Processed tailings disposal is any combination of unit operations implemented to modify the tailings product. For this thesis, unit operations were selected to achieve a neutral tailings material. Several methods are available for each unit operation. However, specific methods were selected for this thesis based on reliability and industry acceptance. The flowsheet selected is shown in Figure 2.2.

The methods selected for this thesis were flotation for desulphurizing tailings, followed by autoclaving to oxidize the final sulphide concentrate, and direct lime water treatment of the autoclave discharge.

The model comprises seven worksheets: input parameters, sulphide separation, sulphide oxidation, water treatment, tailings management area, security and bonding (includes tailings facility closure), and project cost summary. Each model component is discussed in the following sections.

2.1 Input Parameters

The "Input Parameters" sheet is the base sheet to which all other calculations refer. The user can input the values for parameters relating to flotation cell efficiency, site characteristics of the tailings impoundment area and financial considerations (cost of capital, interest rate and US dollar exchange rate). The parameters mill throughput, mine life, sulphide content in tailings and neutralization potential of tailings are varied randomly in the simulation. The inputs worksheet is shown in Figure 2.3.

Figure 2.2
Flowsheet for Processing Tailings

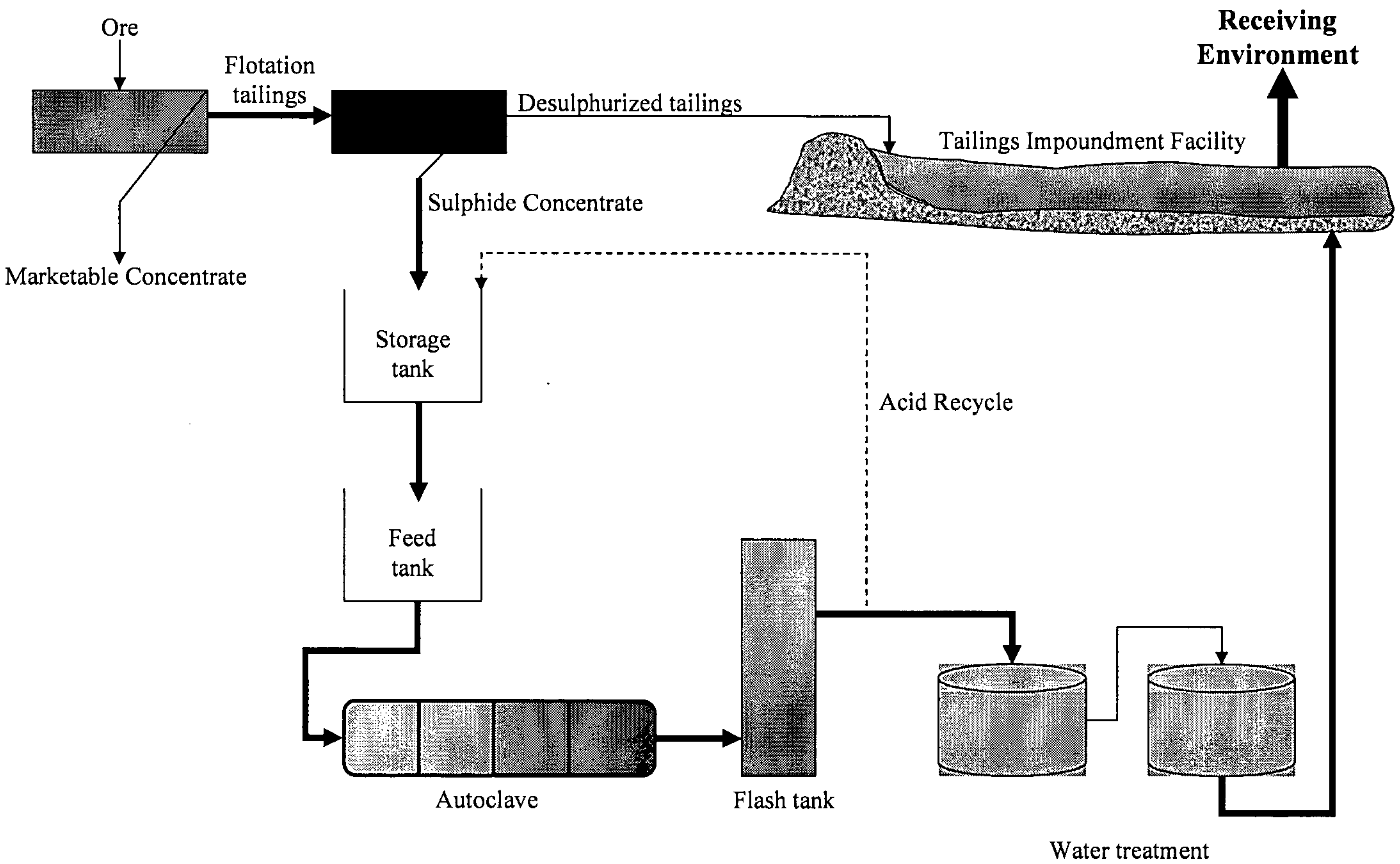


Figure 2.3
Input Parameters Sheet

Project Parameters		**Fiscal Parameters**	
Mill tonnage	2500 tonnes per day	US$ exchange	1.52 $CDN/$US
Sulpide in tailings	5%	%Closure bonded	30%
NP in tailings	100 kg $CaCO_3$ eq./tonne tailings	Bond interest rate	3%
Mine life	20 years		
Operating days/yr	344	**Climate Data**	
Cost of Capital rate	10%	Avg precip	1 m/yr

Flotation Parameters

%S in flotation tails	0.35%
%S in flotation concentrate	48% (assumes 90% pyrite recovery)
Mass pull	10%

Water Treatment

Lime efficiency	80%

Tailings Disposal

Consolidated density	50%
Floor width	150 m
Max dam elevation	100 m
Dam crest width	10 m
Freeboard	5 m
TMA wall slope	30 degrees
Max TMA length	20000 m

Conventional Tailings

Upstream dam slope	20 degrees
Downstream dam slope	20 degrees

Note: upstream dam slope for conventional tailings is shallower to allow the installation of a liner.

Autoclaved Tailings

Upstream dam slope	27 degrees
Downstream dam slope	20 degrees

2.2 Sulphide Separation

Sulphide separation is carried out in the processed tailings alternative only. As discussed in Section 2.0 above, flotation was selected as the separation method for this model. On this worksheet, the overall amount of tailings solids and the amount of sulphides reporting to the concentrate and desulphurized tailings streams are calculated. Equipment sizing is also calculated.

The recovery of sulphides in the flotation circuit was calculated based on a fixed sulphur content in the concentrate and on a calculated mass pull rate (the percentage of the feed rate to the flotation cell that reports to the concentrate), where the sulphur content in the desulphurized tailings is dependent on the sulphur content of the feed. Values for these parameters are discussed below.

A number of researchers have looked at general flotation of sulphides at neutral pH (Humber, 1995; Leppinen *et al*, 1997; Ityokumbul *et al*, 2000; Hodgkinson *et al*, 1994, and Benzaazoua *et al*, 2000). All of the researchers reported success in achieving reasonable (greater than 90%) sulphide recoveries. Test methodologies employed laboratory scale batch flotation cells, xanthates and copper sulphate. Most studies looked at various collectors and collector concentrations. The sulphide content of materials tested ranged from 2.35% to 21.4%. The sulphide content in the desulphurized tailings ranged from 0.06% to 4.15%. When floating cyanidation residues, a wash and repulp with fresh water was found to be necessary to achieve reasonable sulphide recoveries at neutral to alkaline pH (Hodgkinson *et al*, 1994). A summary of reported results are presented in Table 2.1.

Table 2.1
Summary of Flotation Test Results

	Test Number								
Leppinen *et al*, 1997	**9**	**10**	**12**	**13**					
$\%S_{tot}$ starting	4.54	4.54	4.54	4.54					
$\%S_{tot}$ tailings	0.71	0.66	0.48	0.61					
% Recovery	93	88.2	96	93.4					
Humber, 1995	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
*Sample Selbaie 3.2**									
$\%S_{tot}$ starting	23.6	23.6	23.6	23.6	23.6	23.6	23.6	23.6	23.6
$\%S_{tot}$ tailings	21.10	0.70	0.76	1.18	0.97	0.73	1.43	1.55	1.19
% Recovery	14.52	98.31	98.30	97.31	97.71	98.35	96.74	96.27	96.9
Sample Selbaie 3.3	**1**	**2**	**3**	**4**	**5**				
$\%S_{tot}$ starting	3.47	3.47	3.47	3.47	3.47				
$\%S_{tot}$ tailings	0.49	0.32	0.41	0.71	0.39				
% Recovery	87.89	92.31	90.67	81.95	90.10				
Hodgkinson *et al*, 1994	**6**	**7**	**13**	**14**	**16**				
$\%S_{tot}$ starting	-	-	-	-	-				
$\%S_{tot}$ tailings**	0.25	0.24	0.35	0.39	0.44				
% Recovery	95.31	95.37	93.41	92.67	91.56				
Benzaazoua *et al*, 2000	**P**	**M**	**G**						
$\%S_{tot}$ starting	2.9	16.2	24.2						
$\%S_{tot}$ tailings**	<0.3	1.8	1.4						
% Recovery	90	95	96						

$\%S_{tot}$ = percentage of total sulphur in a sample.

* Humber (1995) noted that significant oxidation of the tailings occurred prior to testing and the sample was difficult to treat.

** Values are calculated using published data and assuming 5% S in the test feed material.

The data presented above show that the sulphur content of the desulphurized tailings is distinctly different for feed materials containing low or high sulphur contents. Based on this data, feed dependent sulphur contents were chosen for the desulphurized tailings. These values are presented in Table 2.2.

Table 2.2
Sulphur Content of Desulphurized Tailings in the Model

%S in Feed	%S in Desulphurized Tailings
2-5	0.35
5-10	0.46
10-15	0.57
15-20	0.68
20-25	0.79

The selection of sulphur content categories for feed material is arbitrary. The sulphur contents of desulphurized tailings for feed materials containing 2% to 5% S and 20% to 25% S are an average of data presented in Table 2.1. Intervening values were determined by interpolation.

The total mass of the sulphide concentrate was calculated by using a mass pull rate (the percentage of the feed rate to the flotation cell that reports to the concentrate). The mass pull rate is calculated based on the mass and sulphur balances:

Mass: $\mathrm{F} = \mathrm{C} + \mathrm{T}$ (1)

Sulphur: $f\mathrm{F} = c\mathrm{C} + t\mathrm{T}$ (2)

where F, C and T is the mass of feed, concentrate and tailings, respectively

f, c and t is the %S in feed, concentrate and tailings, respectively

By rearranging equations (1) and (2), the mass pull rate can be calculated by the following equation:

$$\text{MassPullRate} = \frac{\text{C}}{\text{F}} = \frac{f - t}{c - t} \tag{3}$$

In the model, the %S in feed is randomly varied and the %S in tailings is determined according to Table 2.2. It is assumed that flotation will be reasonably efficient and the concentrate will contain 90% pyrite. This results in 48% S in the concentrate.

In calculating equipment size additional assumptions were made regarding the water content of the various streams around the flotation circuit. Values were selected based on flotation design principles discussed by Arbiter (1985). Assumptions for flotation used in this model are summarized in Table 2.3. The mass balance around the flotation circuit is given in Figure 2.4.

Table 2.3
Assumptions for Flotation

Parameter	**Value**	**Source**
Mill tonnage	500 – 20,000 tonnes/day	Randomly varied
Sulphide concentration	2% - 25%	Randomly varied
Pulp density	30%	Arbiter (1985)
Residual sulphide	0.7%	median of Table 2.1
Mass pull rate	50%	Benzaazoua *et al* (2000)
Water recovery in froth	20%	Fig. 11 in Arbiter (1985)

The volume of flotation cells required for a given tonnage and separation was determined using the following equation from Arbiter (1985):

$$NV = \frac{QTEX}{24}$$

where:

NV = total effective cell volume (m^3)

Q = dry ore throughput (tonnes/day)

T = retention time (minutes)

E = pulp expansion factor due to aeration = $\left(\frac{1}{1 - \%\,\text{pulp volume as air}}\right)$ (unitless)

X = pulp flow rate (m^3 pulp/min/tonne dry ore/hr)

and $X = 0.5338\left(\frac{1}{S} + \frac{100}{P\%} - 1\right)$

where: S = specific gravity of ore

P% = pulp density

0.5338 = a constant

It was assumed that retention time is 12 minutes (from Benzaazoua *et al*, 2000), retention time scale up factor is 2 (Arbiter, 1985), percent pulp volume as air is 15% (Arbiter, 1985) and the specific gravity of ore is 2700 kg/m^3 (Arbiter, 1985).

The sulphide separation worksheet in the model is shown in Figure 2.4.

Figure 2.4
Sulphide Separation Sheet

Input Parameters

Mill tonnage	2500 tonnes per day	
	104 tonnes per hour	
Sulphide concentration	5%	
NP	100 kg CaCO3 eq./tonne tailings	
Pulp density	30%	
Specific gravity of ore	2700 kg/m^3	(from example in Flotation (Arbiter, 1985))
% air in pulp	15%	
Collector addition	70 g/tonne	
Frother addition	16 L/tonne	
Flotation time	12 minutes	
Scale factor	2	
Scaled flotation time	24 minutes	
Residual S	0.4%	
Mass pull	10%	
Water recovery in froth	20%	(from Figure 11 in Flotation (Arbiter, 1985))

Sizing Flotation Cells

NV = QTEX/24

Where NV = total effective cell volume
Q = dry ore throughput, tonnes per day
T = circuit retention time
E = pulp expansion factor due to aeration
X = pulp flow rate (m^3 pulp/min/tonne dry ore/hr)

And X = 0.5338[1/S + 100/P% -1]
Where S = specific gravity of dry ore
P% = solid content in pulp by weight

Therefore:

Q = 2500 tonnes per day
T = 24 minutes
E = 1.1765
X = 1.2457 m^3 pulp/min/tonne dry ore/hr

NV = 3664 m^3

Mass Balance

Parameter	Units	Ore	Concentrate	Tails	Totals
Solids mass	tonnes/day	2500	243	2257	2500
Mass of S	tonnes/day	125	117	8	125
Mass of NP	kg $CaCO_3$ eq./day	250000	24344	225656	250000
Water mass	tonnes/day	5833	1167	4667	
Pulp density	%	30	17	33	

Calculated ABA

%S		5	48	0.35
AP	kg/tonne	156	1503	11
NP	kg/tonne	100	100	100
NP:AP		0.64	0.07	9.14

2.3 Sulphide Oxidation

Sulphide oxidation can be achieved by pressure oxidation or by atmospheric oxidation using aeration or bacterial catalyst. Pressure oxidation using an autoclave was selected for this model because of its more rapid process rate which better fits with the overall processing rate of a beneficiation plant.

The solids and water mass in the flotation circuit concentrate is used as the input to the autoclave sizing. The autoclave design employed for this model is based on the design parameters utilized for the pressure oxidation unit at the Miramar Con Mine, a gold mine in Yellowknife, Northwest Territories (Fluor Daniel, 1999). The mine constructed an autoclave to treat pyrite and arsenic trioxide sludge, pyrite being the source of iron to convert arsenic trioxide to iron arsenate. The autoclave product is hematite and iron arsenate. In the model only pyrite is assumed to be treated; therefore, the autoclave product contains only hematite.

The oxidation of pyrite (the predominant sulphide mineral in most tailings) to hematite occurs by the following reaction:

$$4H_2O + 2FeS_2 + 7\frac{1}{2}O_2 \rightarrow Fe_2O_3 + 4H_2SO_4 \qquad (4)$$

The reaction is exothermic, spontaneous and generates heat. This generated heat enables the reaction to be self-sustaining when there is an uninterrupted supply of reactants, namely pyrite and oxygen. Water is added to the autoclave to control the reaction. The oxygen demand by the autoclave is driven by the reaction equation shown above, namely 15 moles of oxygen is required to oxidize 4 moles of sulphur. An oxygen efficiency of 80% was assumed in the model (W.E. Norquist, pers. comm.). The products of oxidation will be hematite and sulphuric acid.

The autoclave circuit includes the following components (see Figure 2.2):

- A storage tank for acidifying the flotation concentrate to remove any carbonates in the feed material,
- A feed tank for decreasing the pulp density and smoothing out the feed rate,
- A four compartment autoclave, for carrying out the oxidation reaction,
- An oxygen plant for supplying oxygen to the autoclave (not shown in Figure 2.2),
- A flash tank for releasing pressure and heat from the autoclave product, and
- A scrubber for removing acid from the steam vented from the flash tank (not shown in Figure 2.2).

Sizing of the components is based on the feed rate, the retention time for each component and the operating parameters of the autoclave. The solids feed rate for each component is calculated by mass balance. Retention times are the same as those used at the Con Mine (W.E. Norquist, pers. comm.) and are summarized in Table 2.4. Operating parameters of the autoclave and pulp densities at various points in the autoclave circuit are from Fluor Daniel (1999). Pulp density of the flotation concentrate entering the storage tank is 52% (see Figure 2.4). Water is added to the feed tank to bring the pulp density down to 26% in preparation for oxidation. The autoclave product exits at 19% solids, due to the dissolution of sulphides. The slurry loses water in the form of steam in the flash tank, thereby increasing the pulp density to 23%. The mechanical efficiency of the autoclave was assumed to be 85% (W.E. Norquist, pers. comm.).

Table 2.4
Retention Times for the Autoclave Circuit Components

Component	Retention Time (hrs)
Storage tank	10
Feed tank	24
Autoclave	1.5

Detailed calculations for sizing equipment are shown in Figure 2.5.

Figure 2.5
Sulphide Oxidation Sheet

Input Parameters

Autoclave feed rate	243.4392	tonnes/day
S feed rate	117	tonnes/day
Water feed rate	1166.667	tonnes/day
Pulp density	17%	
Specific gravity of slurry	1580	kg/m^3 (SG of Con Mine Flotation concentrate at 50% solids for autoclave)
Storage retention time	10	hrs.
Feed retention time	24	hrs.
Autoclave retention time	1.5	hrs.
Oxygen efficiency	80%	
m^3 per cu.ft.	0.02832	

Storage Tank Sizing

Volume of slurry	892	m^3/day
	37	m^3/hr
Retention time	10	hrs
Tank volume	372	m^3
	13131	ft^3

Autoclave Sizing

Total slurry mass	936	tonnes/day
	39013	kg/hr
Specific gravity of slurry	1240	kg/m^3
Volume of slurry	31	m^3/hr
Retention time	1.5	hr
Effective autoclave vol.	47	m^3
Mechanical efficiency	85%	
Total autoclave volume	56	m^3
	1960	ft^3

Autoclave Mass Balance

		In	Out
Total Solids	tonnes/day	243.4392	126
S	tonnes/day	117	0
Water	tonnes/day	693	539
Pulp density	%	26%	19%

Flash Tank Mass Balance

		In	Out
Total Solids	tonnes/day	126	126
S	tonnes/day	0	0
Water	tonnes/day	539	423
Pulp density	%	19%	23%

Feed Tank Sizing

Pulp density for autoclave	26%	
Solid feed rate	243.4392	tonnes/day
water feed rate	1166.667	tonnes/day
added water	-474	tonnes/day
Total slurry mass	936	tonnes/day
Specific gravity of slurry	1240	kg/m^3
Volume of slurry	755	m^3/day
	31	m^3/hr
Retention time	24	hrs
Tank Volume	755	m^3
	26663	ft^3

Autoclave Oxygen Demand

Converting FeS_2 to Fe_2O_3

$$2\ FeS_2 + 7.5\ O_2 + 4\ H_2O \rightarrow Fe_2O_3 + 4\ H_2SO_4$$

Therefore, 15 moles of oxygen is required for 4 moles of sulphide

S feed rate	117	tonnes/day
	4879252	g/hr
S molecular weight	32	g/mol
Moles of S	152477	mol/hr
Oxygen needed	571787	mol/hr
Oxygen efficiency	80%	
Oxygen required	714734	mol/hr
O molecular weight	16	g/mol
O_2 required	11435746	g/hr
	11.4	tonnes/hr
	274	tonnes/day

2.4 Water Treatment

Water treatment is a unit operation in the conventional and processed tailings alternatives. As indicated in Figures 2.1 and 2.2, water treatment would occur at different points in the process. The conditions at these two points would be distinct from each other, requiring different water treatment processes.

Conventional Tailings Disposal

Water treatment is almost always required for excess pond water prior to recirculation to the mill or discharge to the receiving environment. Tailings pond water frequently contains one or all of acidity, sulphate or elevated metal concentrations and need to be removed in order to meet water quality criteria for mill process water or for the environment. Volumes requiring treatment are determined by the amount of water discharged with the tailings solids and the consolidated tailings density, mine water pumped to the tailings facility, precipitation and background run-off entering the facility.

Lime neutralization is applied extensively in the mining industry (Murdoch *et al*, 1995). Field application has shown the ability of lime neutralization to treat very acidic solutions, accommodate a wide range of flows, and with moderate capital and operating costs compared to other treatment technologies. Lime neutralization can achieve near complete precipitation of metals as metal hydroxides and sulphuric acid as gypsum by the following reactions:

$$Me^{2+} + SO_4^{2-} + Ca(OH)_2 + 2H_2O \rightarrow Me(OH)_2 + CaSO_4{\cdot}2H_2O \quad \text{and} \quad (5)$$

$$2Me^{3+} + 3SO_4^{2-} + 3Ca(OH)_2 + 2H_2O \rightarrow 2Me(OH)_3 + 3CaSO_4{\cdot}2H_2O \quad (6)$$

Lime neutralization is typically implemented in one of three ways (Murdoch *et al*, 1995):

1. Lime is added to the effluent stream and mixed with the tailings discharge;
2. The effluent is aerated to oxidize iron and reacted with lime in a separate circuit; or
3. A modification of the second method. After the effluent is aerated and reacted with lime, the slurry is thickened and a portion of the underflow is recycled to the beginning of the water treatment circuit.

The first method produces a voluminous sludge (2.5% solids) with questionable sludge stability. The second method produces a more chemically stable sludge, but the sludge density remains low. The third method, commonly known as the high density sludge (HDS) process, produces a chemically stable sludge with a thickened sludge density of about 20%. The sludge exhibits free draining properties and can rapidly achieve 40% to 50% solids density (Kuit, 1980, Kuyucak *et al*, 1991). Murdoch *et al* (1995) state that the improved sludge characteristics are a result of the formation of precipitates on the surfaces of the recycled particles. In other words, precipitates will preferentially form on existing solid surfaces rather than form new, smaller solid particles. The third method of neutralization was selected for this model.

The HDS water treatment plant was sized based on estimated flow and assumed solution chemistry typical of acidic drainage. Flow was estimated from the amount of process water that would be expressed during consolidation, an assumed precipitation rate of 1000 mm per year and run-off water. Run-off was assumed to be 20% of the process water and precipitation inflows, which is reasonable for a relatively wet climate. Tailings supernatant was assumed to contain sulphate and metals at a pH of 6 during operations. In the post-closure period, water quality was assumed to be 152 mg/L Fe and 3500 mg/L SO_4 at a pH of 3.5. Example calculations are given in Figure 2.6.

HDS water treatment design calculations for determining the size and cost of the plant was based on interpolating unpublished data by Humber (1996). In summary, tailings pond water would be neutralized to pH 9.3 with a residence time of 40 minutes. The size and cost of an HDS water treatment plant was calculated using Humber (1996) for five different scenarios. Results for these scenarios were placed in an Excel look-up table from which all other plant sizing and costing calculations were interpolated. An example calculation using the unpublished data by Humber (1996) is given in Appendix A. The look-up table in the model is given in Table 2.5.

Table 2.5
Look-Up Table for Sizing HDS Water Treatment Plants

Flow (L/min)	Capital Cost (US$)	Operating Unit Cost ($/L/yr)	Sludge Volume (m^3/L/yr)
1,200	1,000,000	235	3.054
4,000	2,300,000	206	3.054
5,000	2,600,000	198	3.054
20,000	6,700,000	169	3.054
35,000	9,400,000	158	3.054

Figure 2.6
Water Treatment Sheet – Conventional Tailings

Input Parameters

Water from processing	5,833	tonnes/day	
Consolidated density	50%		
Expressed water	3,333	tonnes/day	
	3,333,333	L/day	

Infiltration Water (Operations)

Precipitation	1	m/yr	
	0.0027	m/day	
Tailings facility surface area	3,832,210	m^2	(from tailings facility sizing sheet)
Volume from direct precipitation	10,492	m^3/day	
	10,492,018	L/day	
Volume from run-off	2,765,070	L/day	(assume 20% excess for run-on)
Total volume to treat	16,590,422	L/day	
	11,521	L/min	

Infiltration Water (Post Closure)

Infiltration through cover	1.00E-07	cm/s	(typical rate through degraded liners)
	8.64E-05	m/day	
Tailings facility surface area	3,832,210	m^2	(from tailings facility sizing sheet)
Volume of infiltration thru cover	331	m^3/day	
	331,103	L/day	
	230	L/min	

HDS Plant

Design flow	12,700	L/min	
pH (operations)	6	s.u.	
pH (post closure)	3.5	s.u.	
sulphate	3,500	mg/L	
iron	152	mg/L	
Canadian exchange rate	1.52	Cdn$/US$	
capital cost	4,548,556	US$	
	6,913,806	Cdn$	
operating cost (operations)	206,375	US$/yr	(1 order of magnitude less than cost at pH 3.5)
	313,690	Cdn$/yr	
operating cost (post closure)	65,879	US$/yr	
	100,137	Cdn$/yr	

Sludge Pond 773,932 m^3

Autoclaved Tailings Alternative

The oxidized material produced by the autoclave will be comprised of insoluble material, predominately silicates, hematite and sulphuric acid. The calculated sulphuric acid concentration for the base case scenario is 98 g H_2SO_4/L. This is consistent with the acid concentration at the Con Mine, where the flash slurry product contains 95 g H_2SO_4/L (Fluor Daniel, 1999). The autoclave product will also contain dissolved metals, including an estimated 6 g/L of iron (Fluor Daniel, 1999). Clearly, this material needs to be neutralized prior to disposal to avoid affecting the receiving environment.

As discussed above for conventional tailings disposal, the high density sludge water treatment system is the preferred treatment method due to the increased stability of the sludge and the lower sludge volumes. The existing solids in the autoclave discharge will allow the use of a simplified lime neutralization process and still achieve the sludge characteristics typical of an HDS process. In addition, the oxidized nature of the autoclave product makes aeration unnecessary.

The lime neutralization water treatment plant was sized based on the flow and solution chemistry predicted for the autoclave discharge (Section 2.3). Example calculations are given in Figure 2.7.

Figure 2.7
Water Treatment Sheet – Autoclaved Tailings Alternative

Input Parameters (Flash Tank Product)

Pulp density of discharge	23%		
Amount of solids	126	tonnes/day	
Amount of liquid	423	tonnes/day	
S oxidized	117	tonnes/day	
Specific gravity of liquid	1,040	kg/m^3	(SG of flash tank product water at Con Mine (Fluor Daniel, 1999))
Specific gravity of slurry	1,220	kg/m^3	(SG of flash product slurry at Con Mine (Fluor Daniel, 1999)
Residence time	30	minutes	
Lime efficiency	80%		

Sulphuric Acid Concentration in Flash Tank Product

S oxidized	117	tonnes/day
	117,102,037	g/day
S molecular wt	32	g/mol
Moles S	3,659,439	mol/day
1 mole S = 1 mole H_2SO_4		
H_2SO_4 molecular wt	98	g/mol
Amount of H_2SO_4	358,624,989	g/day
Amount of liquid	423	tonnes/day
SG of water	1,040	kg/m^3
Volume of liquid	406,687	L/day
	282	L/min
Concentration of H_2SO_4	882	g/L

Volume of slurry

Amount of slurry	549	tonnes/day
S.G. of slurry	1,220	kg/m3
Volume of slurry	313	L/min

Direct Lime Plant

Design flow	300	L/min	
pH	1.0	s.u.	
sulphate	96	g/L	
iron	6	g/L	
Canadian exchange rate	1.52	Cdn$/US$	
capital cost	1,465,078	US$	(includes 10% markup for 316 stainless steel)
	2,226,919	Cdn$	
operating cost	11,141,655	US$/yr	
	16,935,315	Cdn$	

Sludge Pond 8,332,116 m^3

The lime neutralization water treatment design calculations are interpolated from the unpublished data by Humber (1996), similar to the HDS plant design calculations described above. In the model, autoclave discharge slurry is neutralized to pH 9.3 with a residence time of 30 minutes. An example calculation using Humber (1996) is given in Appendix B. The look-up table in the model used for the lime neutralization water treatment plant sizing and costing is given in Table 2.6. The operating cost was calculated using the following equation:

$$\text{Operating cost(US\$)} = \$40.41 * \text{slurry volume}\left(\tfrac{L}{min}\right) * H_2SO_4 \text{ concentration}\left(\tfrac{g}{L}\right) \quad (7)$$

The unit cost of $40.41 was determined from Humber (1996).

Table 2.6
Look-Up Table for Sizing Simple Lime Neutralization Water Treatment Plants

Flow (L/min)	Capital Cost (US$)
50	500,000
800	3,000,000
3,000	4,000,000
25,000	5,300,000

SENES (1994) presented HDS and conventional water treatment plant capital costs for various flow rates and acidity values. Comparing Humber (1996) to SENES (1994) for the HDS system at similar flow rates and acidity, the estimated capital costs are similar. For the conventional water treatment system, SENES (1994) presented data for acidity levels that are two orders of magnitude less than the acidity levels predicted in this thesis. However, from the data presented in SENES (1994), capital cost approximately doubles as the acidity increases by two orders of magnitude. If SENES (1994) upper range capital costs are doubled, the Humber (1996) estimates are within the same order of magnitude.

2.5 Tailings Facility

The design of a tailings facility is in part determined by the geochemical behaviour of the material that will be impounded. If the material has the potential to oxidize and produce acidic and metal laden drainage, the tailings facility will need to be lined to prevent the release of contaminated water into ground and surface water. However, lining will be unnecessary if the impounded material produces a neutral drainage. Therefore, an understanding of the tailings characteristics is required.

2.5.1 Tailings Characteristics

Conventional Tailings Disposal

The assumed tailings geochemical characteristics for the base case are 5% sulphide and the neutralization potential (NP) is 100 kg $CaCO_3$ equiv./tonne (see Table 1.2). In acid base account terms, the acidity potential (AP) is 156 kg $CaCO_3$ equiv./tonne for a net neutralization potential of the tailings material of –56 kg $CaCO_3$ equiv./tonne. The neutralization potential to acidity potential (NP:AP) ratio would be 0.64.

In theory, each NP unit will neutralize each acidity potential (AP) unit to produce a net neutral drainage. At NP:AP ratios less than 1 material will eventually produce acidic drainage. At NP:AP ratios greater than 1, only neutral pH drainage should be produced.

However, the empirical nature of the acid base account test does not take into consideration all the factors affecting sulphide oxidation and carbonate dissolution. Data from mine sites indicate that one cannot be certain about the potential of a material to generate acidic drainage when the NP:AP ratio is between 1 and 3. Therefore, the criterion for classifying a material as non-acid generating is an NP:AP ratio greater than or equal to 3 (Price, 1997).

The NP:AP ratio was calculated for 100 simulations where the AP and NP values were randomly varied. In all simulations the ratio was less than 3. Since all scenarios in this thesis result in a material that would be classified as acid generating, the tailings facility in the model for conventional tailings disposal was designed with a liner.

Autoclaved Tailings Alternative

In the processed tailings alternative two types of tailings would be produced – flotation tailings from the final sulphide separation step and neutralized autoclave tailings from the sulphide oxidation step. The flotation tailings will be comprised largely of gangue silicate material with a minor amount of sulphide. The mass balance on the flotation circuit for the base case, as discussed in Section 2.2, is presented in Table 2.7. The NP is expected to remain constant during the flotation tests (Catalan *et al*, 1999). The calculated acid base accounting (ABA) for the flotation tailings is also presented in Table 2.7.

Table 2.7
Calculated Acid Base Account for Flotation Tailings – Base Case

Parameter	Units	Ore	Concentrate	Tails
Mass Balance				
Solids mass	tonnes/day	2500	243	2256
Mass of S	tonnes/day	125	117	8
Mass of NP	kg $CaCO_3$ eq./day	250000	24344	225656
Water mass	tonnes/day	5833	1167	4667
Pulp density	%	30	17	33
Calculated ABA				
S	%	5	48	0.35
AP*	kg/tonne	156	1503	11
NP**	kg/tonne	100	100	100
NP:AP		0.64	0.07	9.14

* Acidity potential where AP = %S * 31.25 (Steffen Robertson and Kirsten, 1989)
** Neutralization potential. 100 kg $CaCO_3$ eq./tonne tailings was assumed for the base case.

Following the classifying criteria discussed in Price (1997) and discussed previously in this section under conventional tailings disposal, the flotation tailings produced under the base case scenario would be classified as non-acid generating. Therefore, a liner would not be required for the tailings facility. The sulphide concentrate from the flotation circuit would be the feed to the autoclave circuit.

The presence of some sulphide in the flotation tailings dictates that a minimum amount of neutralization potential will be necessary in order to build an unlined tailings facility. This requirement means that processing tailings will only be a reasonable consideration if the neutralization potential of the ore meets this critical cut-off. The model does not include this cut-off; however, it must be taken into consideration when evaluating the model results.

The neutralized autoclave tailings, as discussed in Section 2.4, would contain silicate material that remained unaltered through the autoclaving process, hematite, metal hydroxides and gypsum. Silicate and hematite are all well known to be geochemically stable in oxidizing environments with low leaching rates. Studies completed by Zinck and Griffith (2000) demonstrate that high density water treatment sludges have low leachability at neutral to alkaline pH. The study results also show that metal concentrations in the treated effluent are low, indicating that lime precipitation is effective at metal removal from solution.

With both the flotation tailings and the neutralized autoclave tailings likely to produce neutral, low metal drainage, it is expected that the combined tailings will not have a deleterious effect on the receiving environment. Therefore, an unlined tailings facility is used in the model for the autoclaved tailings alternative.

2.5.2 Facility Design

Two basic designs were used in the model – one for acid generating materials and one for materials that would produce neutral, low metal drainage. The fundamental difference between the two designs is the inclusion of a liner in the facility containing acid generating materials to minimize leachate release during operation. The liner assumed for the model comprises of a compacted foundation layer (clay), an impermeable liner and a protective sand layer.

For the purposes of this model it was assumed that the tailings facility would be located in a mountain valley of sufficient length with a tailings dam at one end. Sizing of the tailings facility was based on several assumptions, summarized in Table 2.8.

Table 2.8
Assumptions for Tailings Management Facility Design

Parameter	Units	Conventional Tailings	Autoclaved Tailings
Angle of mountain sides	degrees	30	30
Max dam height	m	100	100
Freeboard	m	5	5
Impoundment floor width	m	150	150
Upstream dam slope	degrees	20	27
Downstream dam slope	degrees	20	20
Dam crest width	m	10	10
Tailings solids density		50%	50%

The dam slope angles were selected to meet the stability requirements at closure (see Section 2.6) and the practicalities of installing a liner in the conventional tailings alternative (C.C. Scott, pers. comm.).

The tailings facility size was determined by extending the length of the facility and increasing the height of the dam until sufficient volume was achieved to store the complete inventory of tailings when the tailings had achieved a consolidated solids density of 50%. The dam height was then increased an additional 5 m to allow for freeboard. The macro written to calculate the dam height and tailings facility length is given in Appendix C. Example calculations determining tailings facility size is given in Figures 2.8 and 2.9.

Figure 2.8
Tailings Facility Sizing Sheet – Conventional Tailings

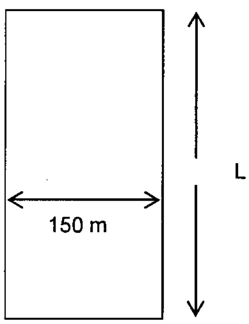
Tailings Discharge to Tailings Facility

Tailings

Solids	2500	tonnes/day
Water	5833	tonnes/day
Pulp density	30%	
Specific gravity of slurry	1233	kg/m^3
Volume inert slurry	6759	m^3/day
Operating days/yr	344	days/yr
Mine life	20	yrs
Water treatment sludge	773932	m^3
Total tailings volume	4.73E+07	m^3
Consolidated density	50%	
Specific gravity	1459	kg/m^3
Consolidated volume	2.36E+07	m^3

Tailings Facility Size

Plan View of tailings floor



Tailings Facility Assumptions

Walls of TMA are	30	degrees
Max elevation is	100	m
Freeboard is	5	m
Floor width is	150	m
U/S dam slope is	20	degrees
D/S dam slope is	20	degrees
Dam crest width	10	m
Max TMA length	20000	m

Floor width	150	m
Minimum dam height	8	m
Freeboard	5	m
Dam Height	13	m
Impoundment Length	19649	m
Impoundment floor surface area	3,975,026	m^2
Dam volume	89,149	m^3
Surface area for reclaim - top	3,832,210	m^2
Surface area for reclaim - dam	5,701	m^2

Figure 2.9
Tailings Facility Sizing Sheet – Autoclaved Tailings

Tailings Discharge to Tailings Facility

Inert Tailings

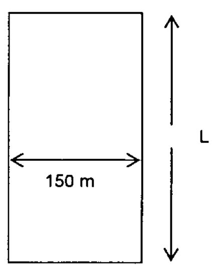
Solids	2256.561	tonnes/day
Water	4667	tonnes/day
Pulp density	33%	
Thicken to	30%	
Solids	2256.561	tonnes/day
Water	5265	tonnes/day
Inert slurry	7522	tonnes/day
Specific gravity of solids	2650	kg/m^3
Specific gravity of slurry	1220	kg/m^3
Volume inert slurry	6165	m^3/day
Operating days/yr	344	days/yr
Mine life	20	yrs
Inert tailings volume	4.24E+07	m^3
Consolidated density	50%	
Specific gravity	1459	kg/m^3
Consolidated volume	2.13E+07	m^3

Combined Tailings

Inert tailings	2.13E+07	m^3
Water treatment sludge	8.33E+06	m^3
Total tailings volume	2.96E+07	m^3

Tailings Facility Size

Plan View of tailings floor



Tailings Facility Assumptions

Walls of TMA are	30	degrees
Max elevation is	100	m
Freeboard is	5	m
Floor width is	150	m
U/S dam slope is	27	degrees
D/S dam slope is	20	degrees
Dam crest width	10	m
Max TMA length	20000	m

Floor width	150	m
Minimum dam height	10	m
Freeboard	5	m
Dam Height	15	m
Impoundment Length	19743	m
Impoundment floor surface area	4,153,230	m^2
Dam volume	101,983	m^3
Surface area for reclaim - top	3,987,326	m^2
Surface area for reclaim - dam	6,579	m^2

2.6 Tailings Facility Closure

The reclamation code in British Columbia (MEMPR, 1992) requires that disturbed land be reclaimed to the level of productivity that existed prior to mining operations, that structures maintain long term stability and that long term water quality is maintained to an acceptable standard. In British Columbia, the most common land use around mine sites is wilderness/forest use. To meet this, reclamation will involve covering the tailings impoundment and revegetating with grass to stabilize the soil surface and planting trees and shrubs to restore a more typical vegetation cover.

The primary structure of concern will be the tailings dam. To maintain stability and allow revegetation, slopes should be in the range of 2.5:1 horizontal to vertical to 3:1 H:V (C.C. Scott, pers. comm. and CDA, 1999) or slope angles of 21.8° to 18.4°, respectively. The assumptions used in the tailings facility design (Section 2.5.2) take into account this stability requirement.

Long term water quality is maintained typically by diverting clean surface runoff around the tailings facility and controlling the seepage from the facility itself. The degree of seepage control required would depend on the predicted quality of the seepage. The poorer the seepage quality, the higher degree of seepage control.

As discussed in Section 2.5.1, tailings disposed in a conventional manner would be expected to oxidize and produce an acidic drainage containing high metal concentrations. Therefore, the closure plan in the model includes seepage control by minimizing infiltration into the tailings facility. In some situations, long term water collection and treatment is required to lower receiving environment impacts (for example, Equity Silver Mine in British Columbia). The model includes a provision for long term water treatment.

The closure plan in the model for the conventional tailings disposal case consists of a spillway in the dam, covering the surface of the impoundment with a low permeability cover, and revegetating the impoundment surface and the dam slope. The cover is comprised of a compacted foundation layer, a 2 mm HDPE liner and topped with growth medium. Post-closure monitoring includes an annual dam inspection and water sample collection. Some minor earthworks will likely be required in the first ten years after closure to repair the diversion ditch channels, spillway and cover until these structures have settled and stabilized. The model includes these provisions.

An additional activity that needs to be considered for the post-closure period is water treatment of tailings impoundment seepages. Most mines are designed and permitted with the expectation that the waste management and the closure measures will be effective in preventing the release of deleterious water to the environment. However, the necessity of long term water treatment often becomes apparent during operations. The cost of water treatment during the post-closure period will depend on the acidity of the drainage(s) to be treated. Predictions of acidity levels (and costs) into the future are usually uncertain. This uncertainty will be the centre of discussions with regulators when negotiating the post-closure bond amount. The approach adopted for the Equity

Silver Mine was to regularly review the monitoring data and adjust the predicted water treatment costs accordingly (Equity Silver, 1996). Given the possibility of water treatment requirements for tailings facilities containing net acid producing tailings, the model includes a provision for post-closure water treatment in the conventional tailings disposal case.

The closure plan in the model for the autoclaved tailings alternative reflects the benign geochemical nature expected of the tailings produced in this alternative. Seepage control and treatment are expected to be unnecessary due to the low metal concentrations predicted for tailings seepage. The closure plan consists of a spillway in the dam, covering the surface of the impoundment with a course granular material and revegetating the impoundment surface and the dam slope. Post-closure monitoring will include an annual dam inspection and water sample collection. Some minor earthworks will likely be required in the first ten years after closure to repair the diversion ditch channels, spillway and cover until these structures have settled and stabilized. These activities are included in the model.

2.7 Security and Bonding

The model addresses the issue of reclamation bonding for the tailings facility only, as it was assumed that all other areas of the mine project would be the same. The model also addresses reclamation bonding in two parts – security and bonding. Security is the amount placed in trust at the beginning of the mine project to fund the necessary physical works at closure. The bond is the amount placed in trust prior to closure to generate sufficient interest income to fund the annual post closure costs. In the model it is assumed that security is posted in the first year and that a bond is posted two years prior to closure. It is also assumed that the bond remains in perpetuity.

The model calculates the amount of security from the estimated closure cost of the tailings management facility. Closure consists of constructing a spillway, cover (impermeable for the conventional case, coarse cover for the processed case) and revegetating the cover and the downstream dam face. The security amount was calculated to be one third of the closure cost estimate to be consistent with the 1997 level of funded liability in British Columbia (Errington, 1997).

The bond cost is estimated from the expected post-closure activities. These activities were assumed to be an annual dam inspection, quarterly water quality monitoring, minor earthworks repairs every three years and maintenance seeding and fertilizing on 20% of the surface area every two years. In the conventional tailings disposal case it is also assumed that a water treatment facility will be operated. Due to the uncertainties associated with predicting post-closure water treatment costs, as discussed in Section 2.6, the water treatment costs estimated for the Equity Silver Mine was assumed for the model. The amount of bond is back-calculated from the estimated annual operating cost assuming the bond earns 3% interest.

An example of the calculations for determining the security and bond amounts for conventional and autoclaved tailings is given in Figures 2.10 and 2.11, respectively.

Figure 2.10
Security and Bond Calculation Sheet – Conventional Tailings

Input Parameters

Surface area - top	3,832,210	m^2
Surface area - dam	5,701	m^2
Bond interest rate	3%	
Cover thickness	1.25	m

Unit Costs

Impermeable barrier	10	$\$/m^2$
Haul & place soil	5	$\$/m^3$
Seeding - ground cover	0.075	$\$/m^2$
Seeding - erosion control	0.12	$\$/m^2$

Closure Cost

Spillway	$50,000
Cover	$62,273,408
Seeding	$288,100
Total closure cost	$62,611,508

Post Closure Cost

	Cost per Yr	Frequency	Avg Cost per Yr
Dam inspection	$15,000	annual	$15,000
Water quality monitoring	$25,000	annual	$25,000
Earthworks repair	$50,000	every 3 yrs	$16,667
Maintenance seeding	$57,620	every 2 yrs	$28,810
Water treatment plant			$1,200,000
Total post closure cost			$1,285,477

Security & Bond Requirements

Security amount	$18,783,452
Bond amount	$42,849,222

Figure 2.11
Security and Bond Calculation Sheet – Autoclaved Tailings

Input Parameters

Surface area - top	3,987,326 m^2
Surface area - dam	6,579 m^2
Bond interest rate	3%

Unit Costs

Simple cover	5 $/m²$
Seeding - ground cover	0.075 $/m²$
Seeding - erosion control	0.12 $/m²$

Closure Cost

Spillway	$50,000
Cover	$19,936,632
Seeding	$299,839
Total closure cost	$20,286,471

Post Closure Cost

	Cost per Yr	Frequency	Avg Cost per Yr
Dam inspection	$15,000	annual	$15,000
Water quality monitoring	$7,500	annual	$7,500
Earthworks repair	$50,000	every 3 yrs	$16,667
Maintenance seeding	$59,968	every 2 yrs	$29,984
Total post closure cost			$69,151

Security & Bond Requirements

Security amount	$6,762,157
Bond amount	$2,305,019

2.8 *Project Cost Summary*

The project cost summary compiles the capital and operating costs for each operating unit and assigns the costs to the year in which the expenditure is made. In the model it is assumed that all capital costs and security payments are incurred in the first year. Operating costs begin in year two and continue to the end of mine life. Bond payments occur two years prior to mine closure. The net present value (NPV) of the project is calculated to enable reasonable comparison between different scenarios. A discount rate of 10% was assumed to reflect the internal rate of return a mine company may use to make a financial decision.

Cost figures for the various capital and operating components were obtained from existing mine operations or from recent feasibility studies. Cost estimates were scaled according to process rate where necessary. The unit cost data and sources for those data are listed in Table 2.9.

The project cost summary for the conventional tailings disposal case (using the base parameters) is given in Table 2.10. The cost summary for the autoclaved tailings alternative is provided in Table 2.11.

Table 2.9
Sources for Cost Data

Item	Unit Rate	Source
Security (closure cost)	calculation	Errington, 1997
spillway	$50,000 lump	SRK Consulting, 2002
impermeable barrier	$\$10/m^2$	SRK Consulting, 2002
place soil cover	$\$5/m^3$	SRK Consulting, 2002
seeding	$\$0.195/m^2$	Confidential Mine (1997)
Bonding (post closure)		
dam inspection	$15,000/yr	P. Healey, pers. comm., 2002
water quality monitoring	$25,000/yr	Confidential Mine (1997)
earthworks repair	$50,000/3 yrs	SRK Consulting, 2002
maintenance seeding	$\$0.195/m^2/2$ yrs	Confidential Mine (1997)
water treatment	up to $1,200,000 /yr	Humber, unpublished data, 1996; Permit M-114
Flotation (capital)	$1,152,000/2300 tonnes	Benzaazoua *et al*, 2000
Flotation (operating)	$0.40/tonne	Mining Sourcebook, 1998
Autoclave (capital)	$23,000,000/120 tonnes	SRK Consulting, 2002
Autoclave (operating)	$4/tonne thru mill	Mining Sourcebook, 1998
Autoclave O_2 plant*	$100/tonne O_2	SRK Consulting, 2002
Water treatment		
HDS (capital)	US$1M – US$9.4M	Humber, unpublished data, 1996
Direct lime (capital)	US$3M – US$5.3M	Humber, unpublished data, 1996
HDS (operating)	US$158 - $235/L/yr	Humber, unpublished data, 1996
Direct lime (operating)	US$40.41/L/g H_2SO_4	Humber, unpublished data, 1996
Tailings Facility		
grubbing	$\$2/m^2$	SRK Consulting, 2002
foundation	$\$1/m^2$	SRK Consulting, 2002
liner	$\$21/m^2$	SRK Consulting, 2002
liner fill	$\$7/m^3$	SRK Consulting, 2002
dam	$\$11/m^3$	SRK Consulting, 2002

*Oxygen is purchased from a third party rather than generated on-site.

Table 2.10
Project Cost Summary Sheet – Conventional Case

Item	Total Cost	Year 1	Year 2 - 17	Year 18	Year 19	Year 20
Capital Expenditures						
Bond & security	$61,632,674	$18,783,452	$0	$42,849,222	$0	$0
Tailings facility	$138,119,031	$138,119,031				
Water treatment	$6,913,806	$6,913,806				
Piping & ancillary	$20,717,855	$20,717,855				
Subtotal Capital	$227,383,366	$184,534,144	$0	$42,849,222	$0	$0
Discount Rate	10%					
NPV Capital	$175,465,122					
Operating Expenditures						
Tailings disposal	$653,600		$34,400	$34,400	$34,400	$34,400
Water treatment	$5,960,111		$313,690	$313,690	$313,690	$313,690
Subtotal Operating	$6,613,711	$0	$348,090	$348,090	$348,090	$348,090
Discount Rate	10%					
NPV Operating	$2,647,041					
NPV Project Cost	**$178,112,163**					

Table 2.11
Project Cost Summary Sheet – Autoclaved Tailings Alternative

Item	Total Cost	Year 1	Year 2 - 17	Year 18	Year 19	Year 20
Capital Expenditures						
Bond & security	$9,067,175	$6,762,157	$0	$2,305,019	$0	$0
Flotation cells	$1,211,099	$1,211,099				
Autoclave & ancillary	$14,484,150	$14,484,150				
Water treatment	$2,226,919	$2,226,919				
Tailings facility	$9,428,270	$9,428,270				
Piping & ancillary	$2,354,287	$2,354,287				
Subtotal Capital	$38,771,901	$36,466,882	$0	$2,305,019	$0	$0
Discount Rate	10%					
NPV Capital	$33,566,289					
Operating Expenditures						
Flotation	$6,536,000		$344,000	$344,000	$344,000	$344,000
Autoclave	$65,360,000		$3,440,000	$3,440,000	$3,440,000	$3,440,000
O_2 Plant	$179,385,683		$9,441,352	$9,441,352	$9,441,352	$9,441,352
Water treatment	$321,770,984		$16,935,315	$16,935,315	$16,935,315	$16,935,315
Inert tailings disposal	$589,955		$31,050	$31,050	$31,050	$31,050
Subtotal Operating	$573,642,622	$0	$30,191,717	$30,191,717	$30,191,717	$30,191,717
Discount Rate	10%					
NPV Operating	$229,592,091					
NPV Project Cost	**$263,158,380**					

3. SENSITIVITY ANALYSES

A number of assumptions were made to derive the costs presented in Section 2.8, specifically mine tonnage, mine life and sulphide content of the ore. The effect of these parameters on project cost was determined by conducting Monte Carlo simulations, varying each parameter simultaneously according to a triangular distribution. The convergence of the simulations was tested by computing the running average of the outputs as the number of trials increased. In each case, the running average exhibited considerable variation which decreased to insignificant levels as the number of trials increased. The simulations are discussed in the sections below.

3.1 Generic Simulations

Mill tonnage and mine life interact to impact several aspects of mine costs. The amount of material processed will have a direct impact on the size of equipment, the size of the tailings facility and the amount of reagents consumed. Mine life influences the size of the tailings facility without influencing the annual operating costs of the mill.

The sulphur content of the tailings, in conjunction with mill tonnage, will influence the size of the autoclave in the autoclaved tailings alternative. The sulphur content will also determine the operating costs of the autoclave. These three parameters, namely mill tonnage, mine life and sulphur content, were varied simultaneously to examine the influence of these parameters on cost.

Mill tonnage was varied from 500 tonnes per day to 20,000 tonnes per day with the most likely value, or peak point of the distribution, at 2,500 tonnes per day. The lower and upper bounds of the distribution were selected based on a review of mineral processing plant data in the Mining Sourcebook (1998), which indicated tonnages in the hundreds of tonnes to tens of thousands of tonnes. Mine life was varied between 10 years and 40 years with the most likely being 20 years. The parameters of the mine life distribution were arbitrarily selected but were reasonable for the industry.

Flotation studies cited in Section 2.2 examined tailings samples containing a wide range of sulphide content. The same range (2% to 25%) was selected for the lower and upper bound for the triangular distribution. A most likely value of 5% was selected for the distribution. Sulphur content of tailings was varied independently from mill tonnage and mine life.

Two hundred simulations were run. The capital costs of the autoclaved and conventional alternatives of the simulations are graphically presented in Figure 3.1. Figures 3.2 to 3.4 present the total costs against each varied parameter resulting from the simulations. Individual simulation results and the distribution of results are presented in Appendix D.

Figure 3.1 shows that the capital cost for the autoclaved tailings alternative is consistently less than the capital cost for the conventional tailings alternative, irrespective of mine life or sulphur content. The main differences in cost are the tailings facility and the security and bond for the conventional alternative. Costs associated with these two elements are significantly higher in the conventional alternative compared to the autoclave alternative. A liner is constructed in the conventional tailings facility, but not in the autoclaved tailings facility. The cost of an autoclave, while expensive, is less than the cost for a liner. Figure 3.1 also shows that the capital cost difference between the two alternatives increases as daily tonnage increases.

However, Figures 3.2 to 3.4 demonstrate the significant cost associated with operating costs, which makes the total cost for the autoclaved tailings alternative higher than the total cost for the conventional tailings alternatives in most simulations, particularly at larger mill tonnages and longer mine lives. It can also be seen that autoclaved tailings total cost increases at a higher rate compared to the conventional tailings total cost.

Figure 3.1
Capital Cost ($millions) vs. Mill Tonnage
All data from first set of simulations are shown

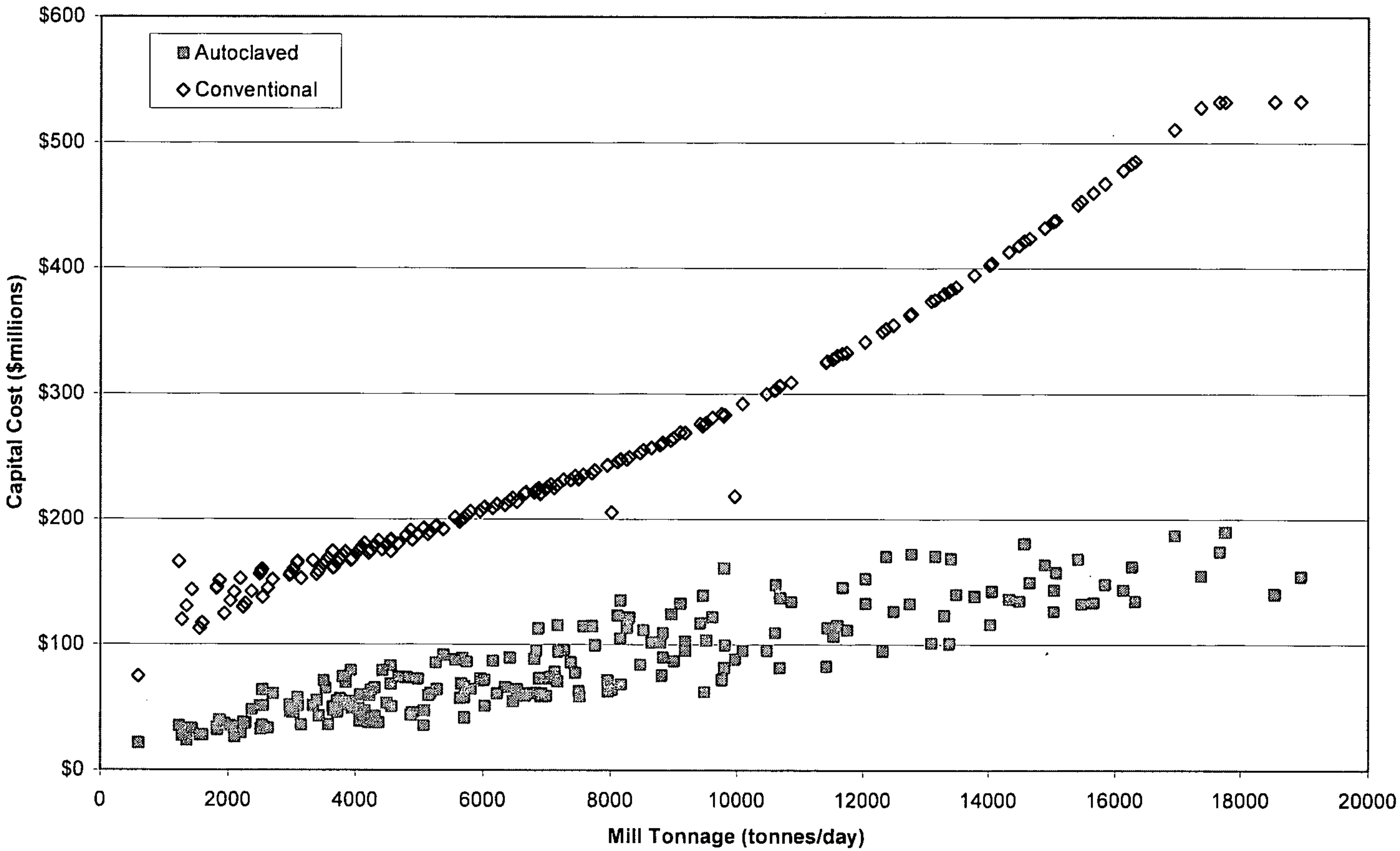


Figure 3.2
Total Cost ($millions) vs. Mill Tonnage
All data from first set of simulations are shown

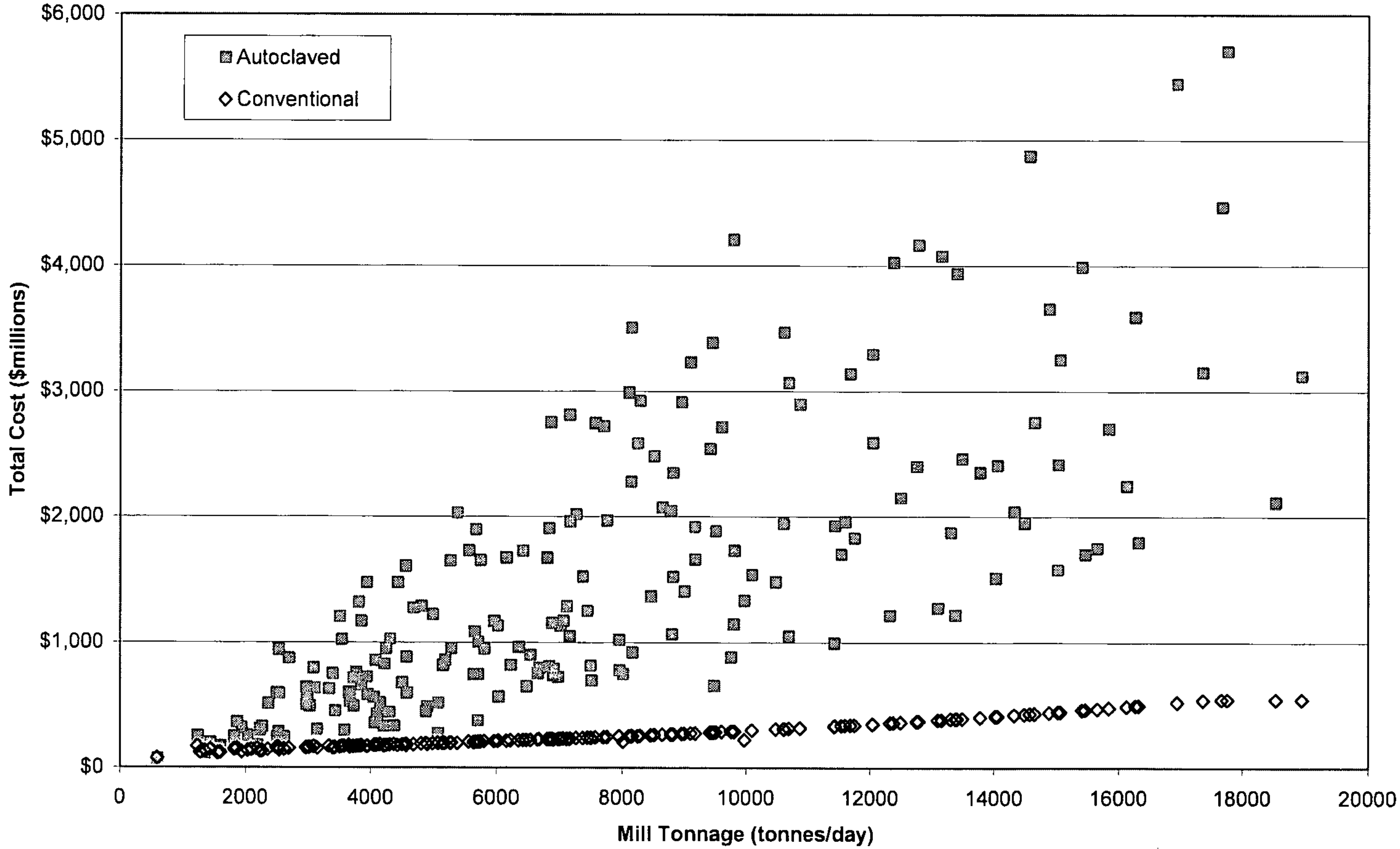


Figure 3.3
Total Cost ($millions) vs. Mine Life
All data from first set of simulations are shown

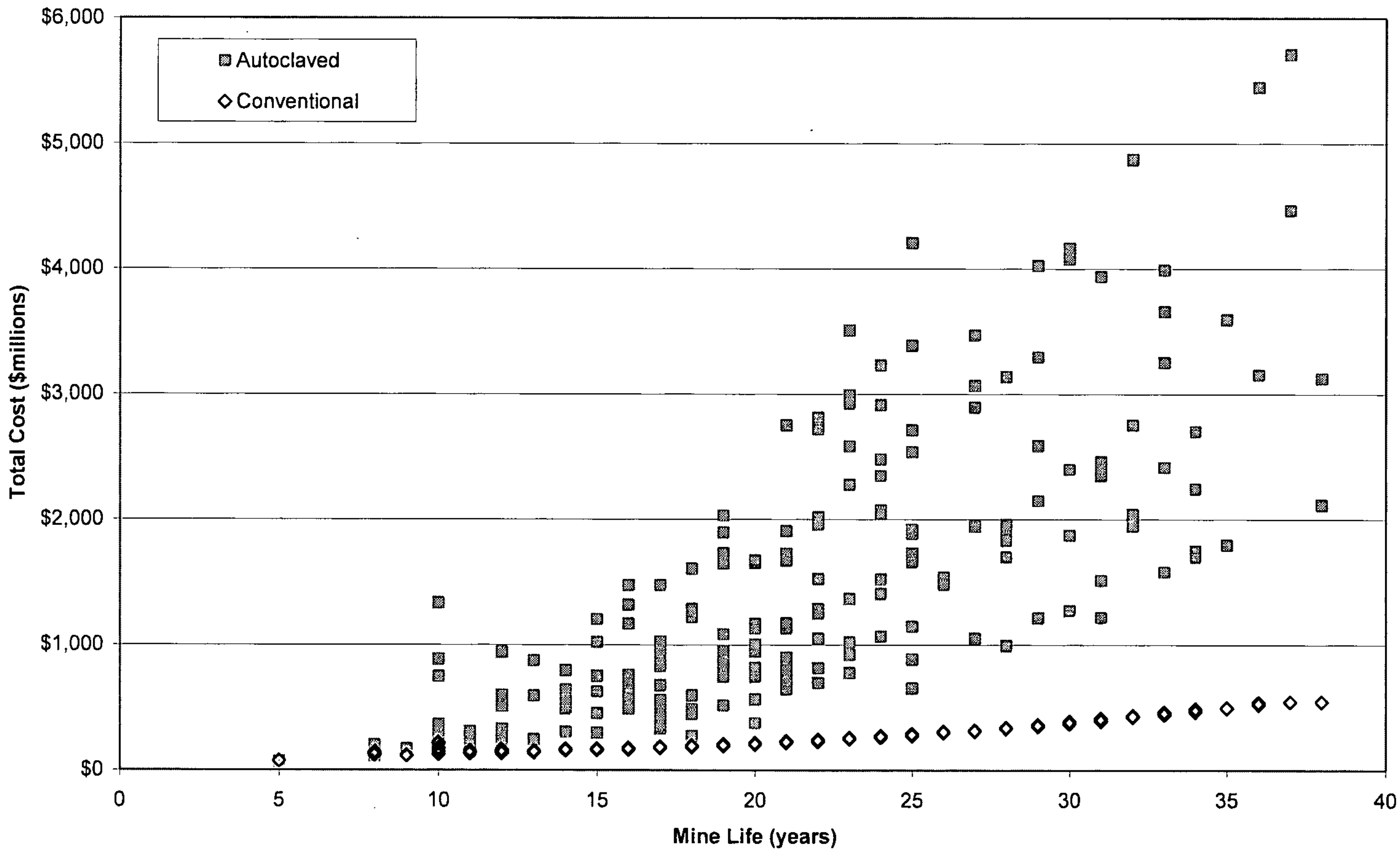
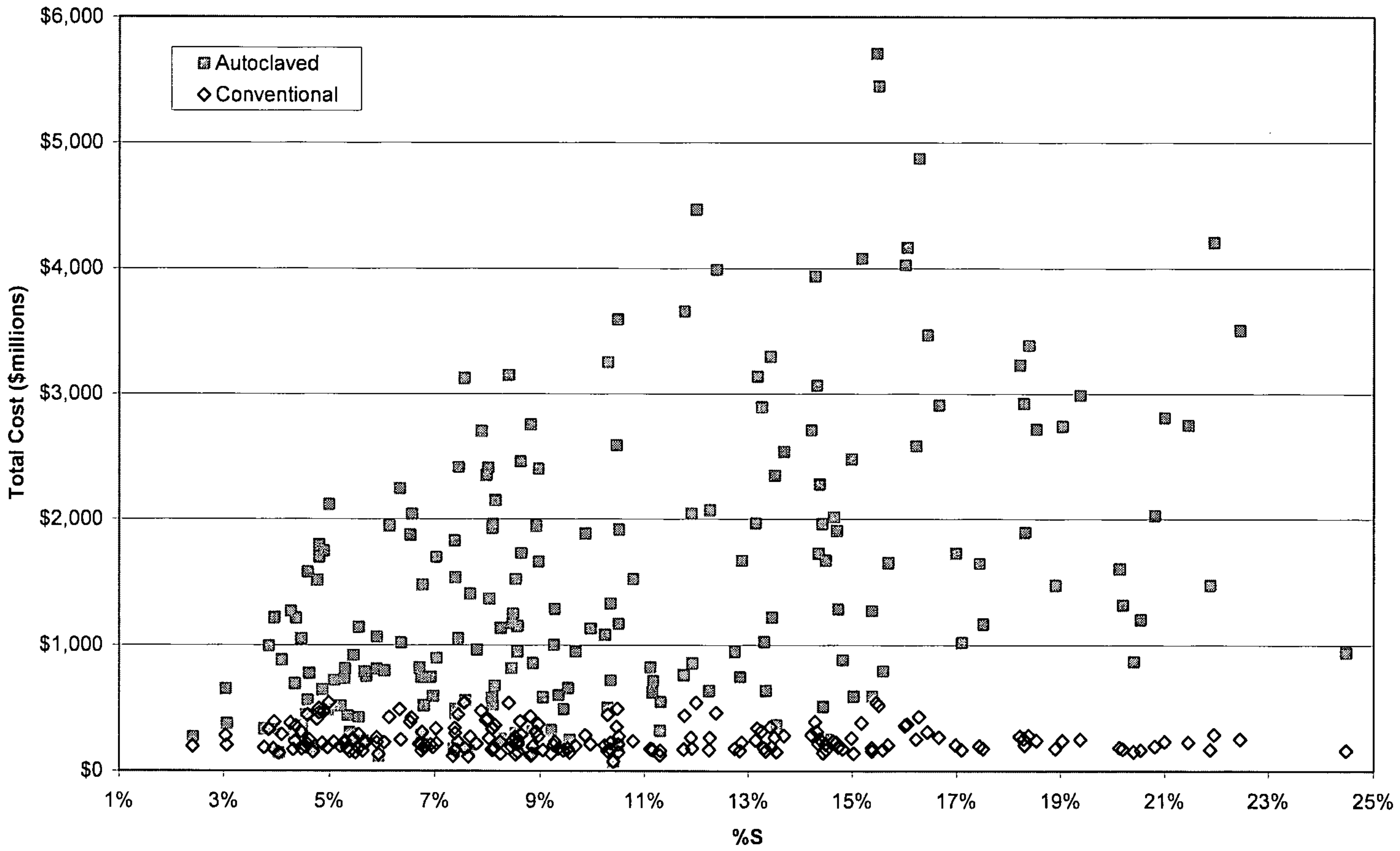


Figure 3.4
Total Cost ($millions) vs. Sulphur Content
All data from first set of simulations are shown



The data presented in Figures 3.2 to 3.4 shows that the total cost for the autoclaved tailings alternative is less than for the conventional alternative in some simulations. The conditions under which this occurred can be broadly described as mill tonnage less than 5,000 tonnes per day, mine life less than 12 years and sulphur content less than 15% S. An additional 200 simulations were run where mill tonnage, mine life and sulphur content were simultaneously varied within this narrower distribution range, namely 500 to 5,000 tonnes per day for mill tonnage, 5 to 12 years for mine life, and 2% to 15% for sulphur content.

The results for this second set of simulations are graphically presented in Figures 3.5 to 3.7. Individual simulation results are tabulated in Appendix D. This second set of data shows trends similar to the first set of data, namely the total cost for the autoclaved tailings alternative increases with increasing mill tonnage and mine life and at a rate greater than for the conventional alternative. However, this second set of simulations clearly shows that there are conditions where the autoclaved tailings alternative costs less than the conventional alternative.

The simulation results were filtered to identify all simulations where the total cost for the autoclaved tailings alternative was less than the total cost for the conventional alternative. This subset of data is shown in Figure 3.8. The data indicate that total costs for the autoclaved alternative are less than the conventional alternative as sulphur content decreases in conjunction with mill tonnage and mine life increases.

Figure 3.5

Total Cost ($millions) vs. Mill Tonnage

All data from second set of simulations are shown

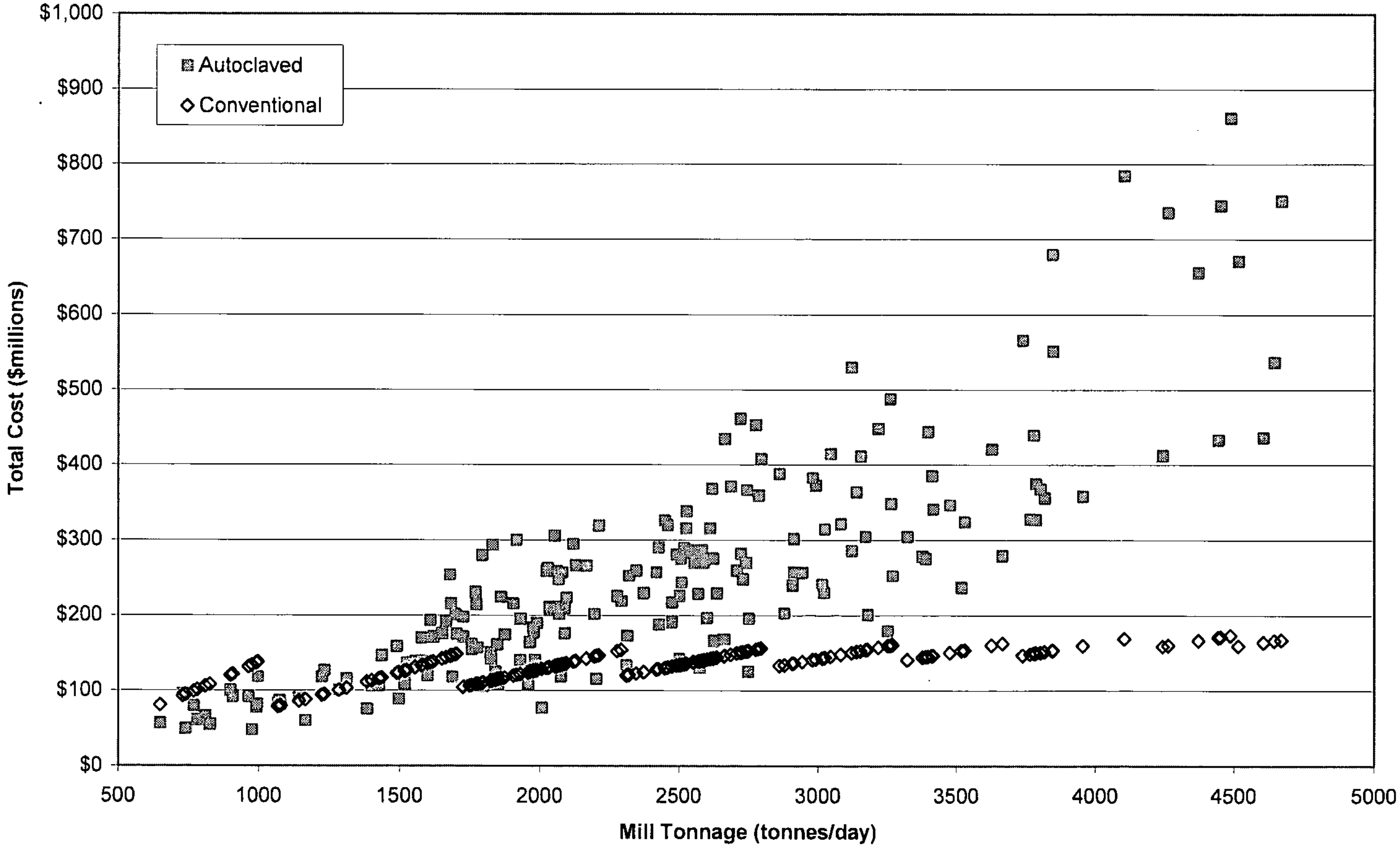


Figure 3.6
Total Cost ($millions) vs. Mine Life
All data from second set of simulations are shown

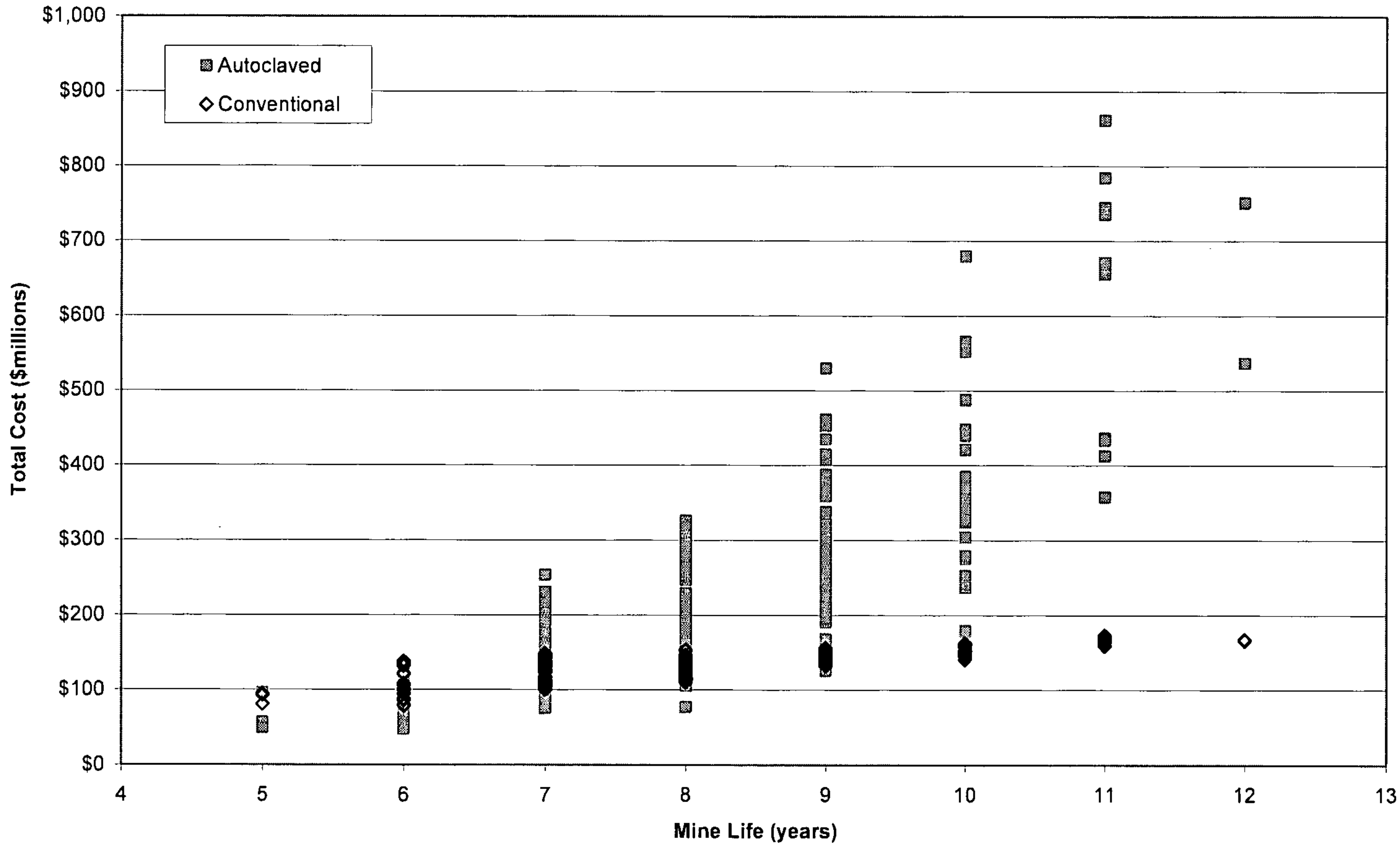


Figure 3.7
Total Cost ($millions) vs. Sulphur Content
All data from second set of simulations are shown

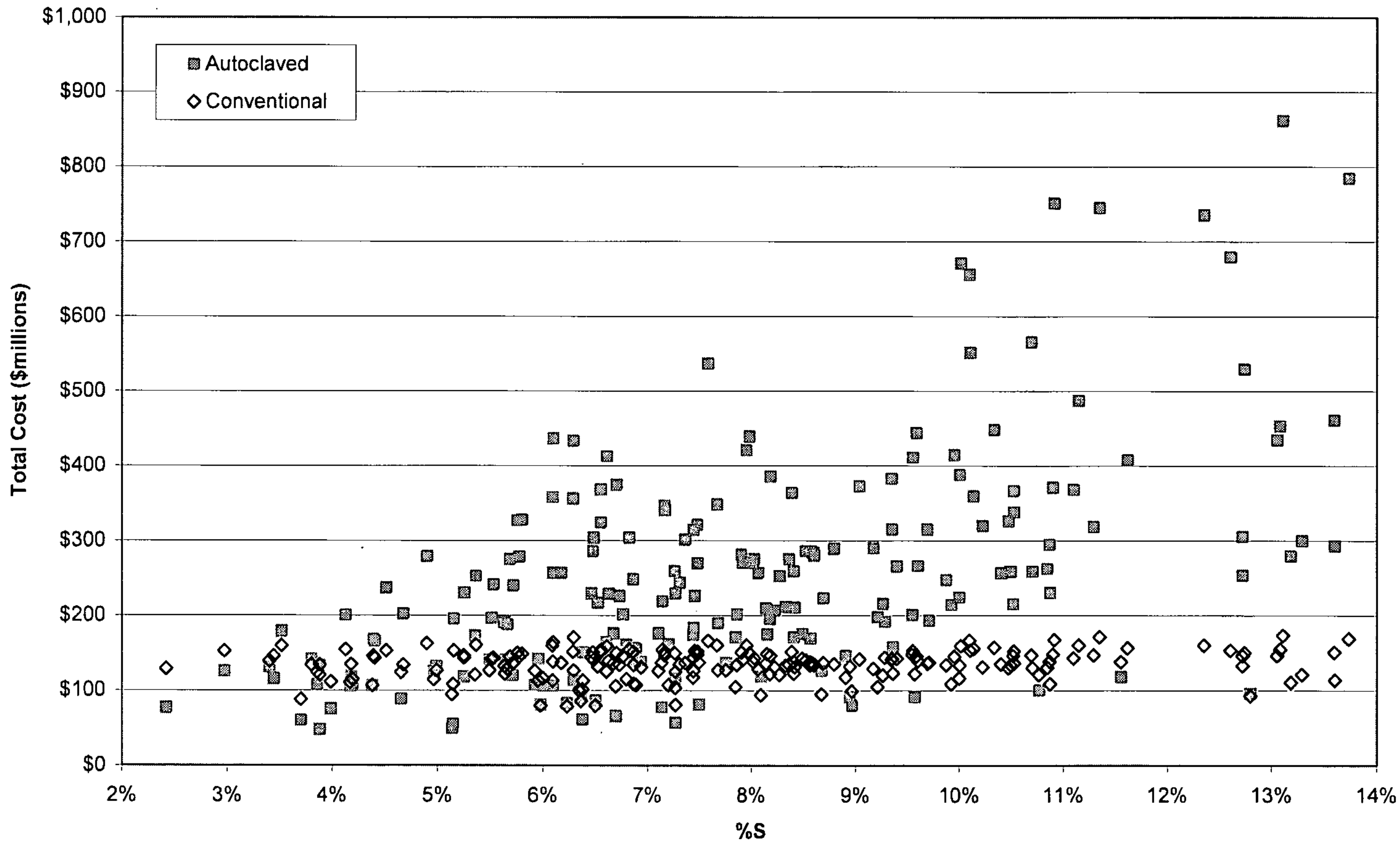
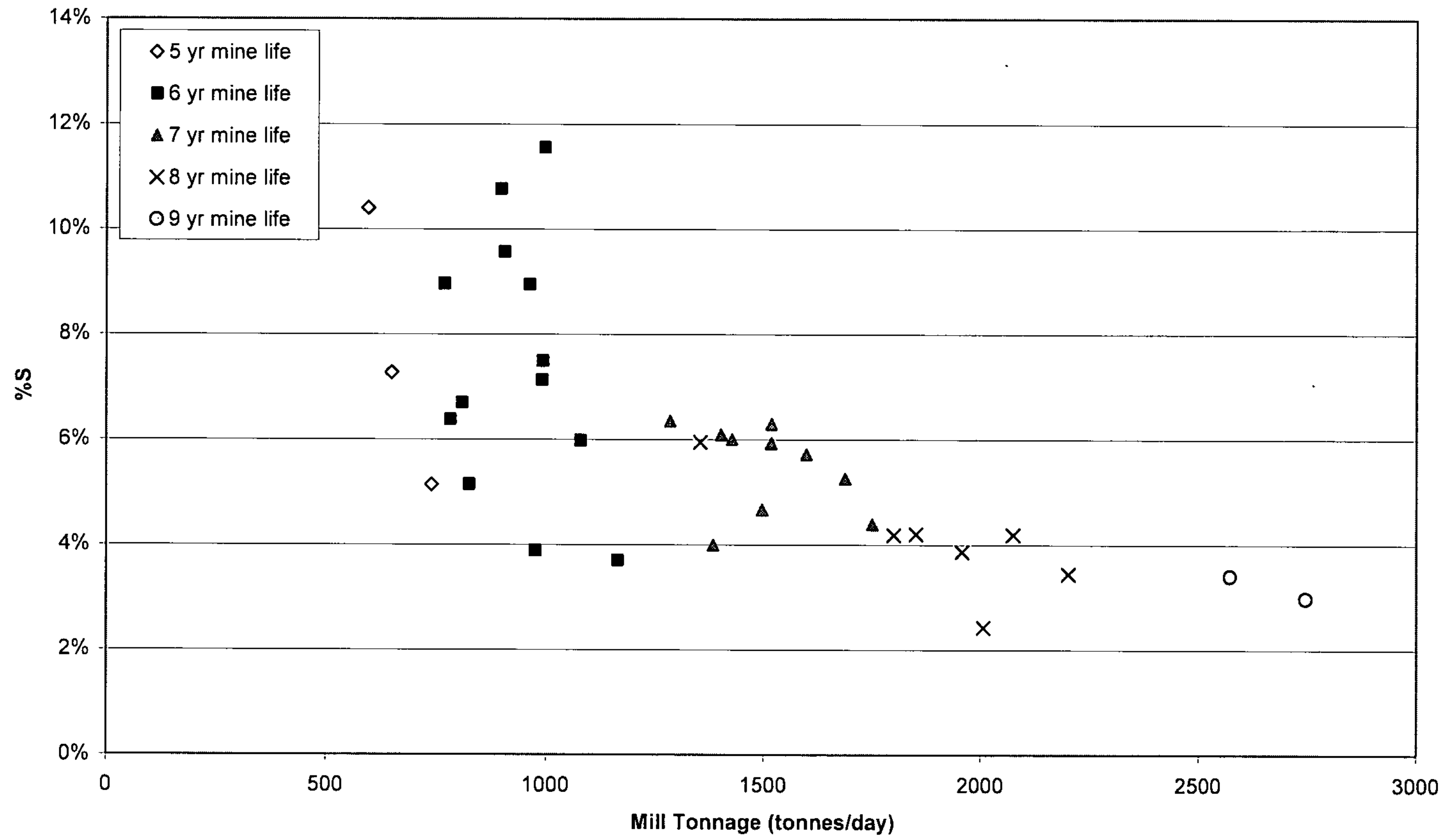


Figure 3.8

Sulphur Content vs. Mill Tonnage for Simulations where the Total Cost for the Autoclaved Alternative is Less Than the Conventional Alternative



3.2 Case Studies

Model runs were completed on three scenarios representing typical mining operations. One scenario was a small tonnage operation mining a deposit containing a relatively high percentage of sulphur. The second scenario was an underground operation mining a massive sulphide deposit. The third scenario was an open pit operation mining a relatively low sulphide deposit. The scenario definitions are given in Table 3.1. Scenario results are summarized in Table 3.2.

Table 3.1
Scenario Definitions

Parameter	**Scenario 1**	**Scenario 2**	**Scenario 3**
Mill tonnage	1,000 tonnes/day	4,000 tonnes/day	20,000 tonnes/day
Sulphur content	12% S	20% S	2.5% S
Mine life	5, 10 and 20 years	5, 10 and 20 years	5, 10 and 20 years

Table 3.2
Total Cost of the Autoclaved and Conventional Alternatives in Defined Scenarios
Costs in $millions

	Scenario 1		Scenario 2		Scenario 3	
Mill tonnage %S	1,000 t/day 12%		4,000 t/day 20%		20,000 t/day 2.5%	
Mine Life	Autoclaved	Conventional	Autoclaved	Conventional	Autoclaved	Conventional
5 yr	$107	$148	$596	$158	$432	$237
10 yr	$174	$139	$1,027	$173	$742	$283
20 yr	$239	$133	$1,463	$193	$1,069	$396

The scenario results suggest that autoclave alternative costs for a high tonnage, low sulphur operation (Scenario 3) can be comparable with an intermediate tonnage, high sulphur operation (Scenario 2) for a given mine life. This is largely a consequence of the different mass pull rates at the sulphide separation step resulting in similar amounts of concentrate flowing to the autoclave. Following equation (3) in Section 2.2, the mass pull rates for Scenarios 2 and 3 are 40.6% and 4.5%, respectively. This translates to 1624 tonnes per day and 900 tonnes per day of material to the autoclave, respectively. However, the autoclaved alternative total cost remained higher than the total cost for the conventional alternative, which is consistent with the simulation results presented in Section 3.1.

4. DISCUSSION

4.1 Model Results

The model results indicate that for smaller operations under certain conditions it may be less costly to autoclave sulphidic tailings rather than dispose a reactive tailings material. This result was observed in the simulations where the mill tonnage was less than 3,000 tonnes per day, the mine life was less than 10 years and the sulphur content of the tailings was less than 12%. The autoclaving alternative is cost effective as sulphur content decreases in conjunction with mill tonnage and mine life increases.

For all simulations, the capital cost associated with autoclaving tailings was less than the capital cost associated with conventional tailings disposal. This in itself may be very appealing to mine developers for the lower interest payments on the funds borrowed prior to production. However, autoclaving does have significantly higher operating costs. For all simulations, autoclaving operating costs were one to two orders of magnitude higher than the operating costs of conventional tailings disposal. The autoclaved tailings operating cost is dominated by the oxygen requirements of the autoclave and by the lime requirements to neutralize the autoclave discharge.

Lime consumption was expected to be high due to the high acid concentration in the autoclave discharge. However, if a very concentrated acid solution was to be produced, such as is anticipated in the simulations containing very high (~20%) sulphide content, it may be more practical to separate the sulphuric acid from the autoclave discharge and sell it. This could potentially reduce the overall operating cost of the tailings processing circuit by significantly reducing the lime consumption rate in the water treatment plant and by generating revenues to offset the cost of oxygen for the autoclave. In addition, the autoclave discharge would also likely contain a significant amount of dissolved metals that could possibly be recovered.

Operating costs for a processed tailings alternative may also be reduced by using an alternate oxidation method. Oxidation methods that could be considered include bio-oxidation and atmospheric pressure leaching. Reagents associated with these processes are likely less expensive than oxygen. On the other hand, the oxidation rates achievable with these alternate methods are likely to be significantly less than the rate achieved in an autoclave. Although this could require a processing time that extends beyond the other mining activities, the operating costs may be covered by a bond – similar to the bonding of long term water treatment operations of today but with more predictable annual costs.

The potential benefits of using an alternate oxidation method include reduced financial burden and good containment of problematic material prior to treatment while still achieving an inert tailings material for disposal. The main technical uncertainties that will need to be addressed are the efficiency of the oxidation process and the geochemical stability of the oxidized solids.

4.2 Model Construction

Many of the design calculations in the model are extrapolations from existing designs. The limitation of this approach is that at the very low and very high ends of the distribution range, the accuracy of the design calculations decrease. The areas of the model where this is an issue are discussed below.

Autoclave Design

The sulphide content of the material directly affects the size of the autoclave, where the amount of water required to maintain the heat balance is dictated by the sulphide content. The autoclave sizing in the model is based on the Con Mine autoclave (see Section 2.3) regardless of sulphide content in the autoclave feed. However, the Con autoclave was designed for a sulphide concentration of 12% in the autoclave feed, equivalent to about 6.5% sulphide in tailings in the model. The autoclave size (and cost) estimated for the simulations where pyrite content is greater than about 10% is likely an optimistic estimate. In other words, the autoclave size would likely be larger to accommodate the increased amount of water required to control the heat generated from the higher sulphide content. As a result the cost would be higher; however, this does not alter the comments in Section 4.1.

Water Treatment Plant Design

There are two limitations to the water treatment cost estimates in the model. One, the range of flows in the model exceed the range of data available. Second, there are no operational data on water treatment plant efficiency in neutralizing autoclave discharges in the manner proposed in the model.

As discussed in Section 2.4, cost estimates for water treatment were calculated by interpolating from a look-up table generated from Humber (1996). The lower and upper bounds of mill tonnage produce autoclave discharge flows that are outside the range of flows where operational data are available. At the low end, water treatment costs may be underestimated, due to a minimum cost that is required to build a plant regardless of treatment volume. Costs may be overestimated at the upper end because the proportion of incremental cost against treatment volume may change at the higher treatment volumes.

Operational data for dilute acidic solutions was used as a basis for estimating the cost of neutralizing autoclave residue. However, field or test data were not found to verify the approach used in the model.

Uncertainty Associated with Tailings with Uncertain Acidic Drainage Potential

The model is designed for situations where there is a high sulphide content in the tailings. The lower bound for sulphide content assumed for this thesis was 2%. Even with sufficient neutralization to prevent acidic drainage, the oxidation of this much sulphide can be sufficient to produce unacceptably high metal concentrations in neutral drainage. In addition, the heterogeneous nature of tailings facilities allows for the possibility of localized acidic drainage although the overall ratio of neutralization potential to acidity production potential indicates neutral drainage. To accommodate these potential scenarios, tailings facility design in the model is done to the same standard as if the entire tailings inventory would produce acidic drainage. The model does not calculate the probability of a geochemical characterization being incorrect and the cost of the remedial measures required to compensate.

5. CONCLUSIONS

A spreadsheet model was developed to compare the whole project costs of an autoclaved tailings disposal alternative to a conventional tailings disposal alternative. The purpose of autoclaving (or processing) tailings is to reduce the long term environmental risk of tailings facilities. Autoclaving was arbitrarily selected as the tailings processing method for its demonstrated ability to efficiently oxidize sulphides.

This thesis demonstrates that processing tailings may be an economically viable way of reducing the environmental liability of mine sites under certain conditions. The conditions under which this holds true are an interplay between mill tonnage, mine life and sulphur content in the conventional tailings. In general terms, the economics of autoclaving tailings become attractive at lower sulphur contents and smaller tonnages. The major findings of this thesis are summarized below.

1. Autoclaved tailings disposal should be considered as a disposal option when the sulphur content of mill tailings is 14% or less, the mill tonnage is less than 5,000 tonnes per day and the mine life is less than 12 years. Cost competitiveness of autoclaved tailings follows a curve as sulphur content decreases in conjunction with mill tonnage and mine life increases.
2. Autoclaving tailings has a lower capital cost requirement compared to conventional tailings disposal.
3. The cost of autoclaving tailings is most sensitive to flotation efficiency.
4. The model would be improved by the addition of sub-models for water treatment and autoclave design.

6. RECOMMENDATIONS FOR FURTHER STUDY

The results of running the model under various simulation scenarios suggest that under certain situations processing tailings may be an economically viable option to reduce the long term environmental risk of tailings facilities. The following studies are recommended to improve the model and to verify the observations discussed herein.

1. Develop a simplified version of MetSim®[1]

   The model uses a single autoclave design as the basis for estimating the autoclave size for a range of scenarios. However, there are limits to the applicability of a single design. An autoclave design sub-model that utilized a more detailed mass balance and a heat balance would improve the accuracy of the model at higher sulphur contents.

2. Include a water treatment model

   Water treatment design and cost depends on the anticipated flowrates and the estimated chemistry of the flows. The model currently uses an assumed water quality for all scenarios and interpolates costs from a limited amount of operational data. The model would have wider applicability and be more accurate if a water treatment sub-model was included. The water treatment model should allow the user to input the predicted water quality of the flows to

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[1] MetSim® is a computer program that calculates the mass and energy balance of metallurgical processes. Recent versions of the program also include process control, equipment sizing, cost estimating and process analysis. MetSim® is sold by Proware. Additional information can be obtained from http://members.ozemail.com/au/~ozmetsim/met1/index.html.

be treated and contain a wider range of operational data from which to interpolate.

3. Verify lime neutralization of autoclave residue

   Autoclave residue neutralization in the model assumes direct neutralization of the residue, as opposed to the more typical approach in a processing plant where the solids and liquor of the residue are separated in a counter current decant circuit and neutralized separately. A bench scale test is recommended to evaluate the practicality of direct neutralization of autoclave residue that has not been separated. Parameters to study should include lime demand and sludge characteristics.

4. Add additional processing alternatives

   The model currently contains one alternative to conventional tailings disposal. Other alternatives could be equally effective in achieving a geochemically stable tailings material. Bio-oxidation, atmospheric pressure leaching, and autoclaving in multiple smaller vessels are some alternatives that could be included.

5. Expand the disposal and closure alternatives

   The model assumes a fixed tailings disposal and closure method for the processed tailings alternative. Other methods may also be suitable. For example, dry stacked tailings and direct vegetation may be a viable disposal and closure method when the run-off quality from tailings material is not a concern.

6. Cost benefit analysis of lower capital costs

   The capital cost of a project, and ways of reducing the amount of capital required, is an important issue due to the fact that capital is almost always borrowed money. In some cases, the alternative with a higher overall project cost will be selected for the lower capital cost. The model currently does not include the cost of capital and other financial considerations, such as tax issues. Inclusion of these items into the overall project cost summary may redefine the conditions under which processing tailings would be financially attractive.

7. REFERENCES

Arbiter, N (ed), 1985. Flotation. In: SME Mineral Processing Handbook, N.L. Weiss (ed.). Society of Mining Engineers, New York, NY, 1985.

Benzaazoua M., Bussiere B., Kongolo M., McLaughlin J., Marion P., 2000. Environmental Desulphurization of Four Canadian Mine Tailings Using Froth Flotation. *Int. J. Miner. Process.* 60 (2000) 57-74.

British Columbia. Ministry of Energy, Mines and Petroleum Resources (MEMPR). Health, Safety and Reclamation Code for Mines in British Columbia. 1992.

British Columbia. Ministry of Energy, Mines and Petroleum Resources (MEMPR). "Mine Reclamation Security Policy in British Columbia. A Paper for Discussion." February, 1995.

Canadian Dam Association (CDA), 1999. Dam Safety Guidelines.

Catalan, L.J.J., Li, M.G., McLaughlin, J., Nesset, J., St-Arnaud, L., 1999. Évolution de Résidus Dépyritisés comme Matériau de Recouvrement pour Limiter le Drainage Minier Acide. Proceedings of Congrès APGGQ, Séminaires "Mines écologiques", Rouyn-Noranda, Québec.

Diaz M.A. and Gochin R.J., 1995. Flotation of Pyrite and Arsenopyrite at Alkaline pH. *Trans. Instn Min. Metall. (Sect. C: Mineral Process. Extr. Metall.)*, **104**, January-April 1995.

Errington, John, 1997. Short Course for Bonding and Security for Mines with Acid Rock Drainage. Short Course Presentation at the Fourth International Conference on Acid Rock Drainage. Vancouver, BC. June 1, 1997.

Equity Silver Mines Ltd. Report of the 1995 Technical Committee. February 21, 1996.

Fluor Daniel, 1999. Miramar Con Mine Recommissioning Project. Project report prepared for Miramar Con Mine Ltd.

Healey, P.M., 2002. SRK Consulting. Personal Communication.

Hodgkinson, G., Sandenbergh, R.F., Hunter, C.J and De Wet, J.R., 1994. Pyrite Flotation from Gold Leach Residues. *Minerals Engineering*, Vol. 7, Nos 5/6, pp. 691-698.

Humber, A.J., 1995. Separation of Sulphide Minerals from Mill Tailings. In: Proceedings of Sudbury '95, Conference on Mining and the Environment. Sudbury, ON. Vol. 1, pp. 149-158.

Humber, A.J., 1996. AES Consulting. Lime Treatment Pre-feasibility Design, unpublished data.

Ityokumbul, M.T., de Aquino, J.A., O'Connor, C.T. and Harris, M.C., 2000. Fine Pyrite Flotation in an Agitated Column Cell. *Int. J. Miner. Process*. 58 (2000) 167-178.

Kuit, W.J., 1980. Mine and Tailings Effluent Treatment at the Kimberley, BC Operations of Cominco Ltd. CIM Bulletin, December 1980.

Kuyucak, N., Mikula, R.J., and Wheeland, K., 1991. Evaluation of Improved Lime Neutralization Processes. Part III Interpretation of Properties of Lime Sludge Generated by Different Processes. In: Proceedings of the Second International Conference on the Abatement of Acidic Drainage, Montreal, PQ, September 16-18, 1991.

Kuyucak, N., Sheremata, T. and Wheeland, K., 1991. Evaluation of Improved Lime Neutralization Processes. Part I Lime Sludge Generation and Stability. In: Proceedings of the Second International Conference on the Abatement of Acidic Drainage, Montreal, PQ, September 16-18, 1991.

Leppinen J.O., Salonsaari P. and Palosaari V., 1997. Flotation in Acid Mine Drainage Control: Beneficiation of Concentrate. *Canadian Metallurgical Quarterly*, vol 36, no. 4, pp. 225-230.

Lewis B.A., Gallinger R.D., and Wiber M., 2000. Poirier Site Reclamation Program. In: Proceedings from the Fifth International Conference on Acid Rock Drainage, Denver, CO, 2000.

Mining Sourcebook, 1998. Mineral Processing Plants – Costs. Can. Min. J., pp. 65-83.

Ministry of Energy, Mines and Petroleum Resources (MEMPR), 1995. Mine Reclamation Security Policy in British Columbia. A Paper for Discussion prepared for the Province of British Columbia. February, 1995. 25pp.

Murdoch, D.J., Fox, J.R.W. and Bensley, J.G., 1995. Treatment of Acid Mine Drainage by the High Density Sludge Process. In: Proceedings of Sudbury '95, Conference on Mining and the Environment. Sudbury, ON. Vol. 2, pp. 431-439.

Norquist, W.E., 2002. Fluor Daniel. Personal communication.

Permit M-4, Reclamation Permit for the Endako Mine. Ministry of Energy and Mines (British Columbia).

Permit M-9, Reclamation Permit for the Island Copper Mine. Ministry of Energy and Mines (British Columbia).

Permit M-11, Reclamation Permit for the Highland Valley Copper Mine. Ministry of Energy and Mines (British Columbia).

Permit M-26, Reclamation Permit for the Myra Falls Mine. Ministry of Energy and Mines (British Columbia).

Permit M-29, Reclamation Permit for the Similco Mine. Ministry of Energy and Mines (British Columbia).

Permit M-40, Reclamation Permit for the Gibraltar Mine. Ministry of Energy and Mines (British Columbia).

Permit M-74, Reclamation Permit for the Sullivan Mine. Ministry of Energy and Mines (British Columbia).

Permit M-112, Reclamation Permit for the Afton Mine. Ministry of Energy and Mines (British Columbia).

Permit M-114, Reclamation Permit for the Equity Silver Mine. Ministry of Energy and Mines (British Columbia).

Permit M-127, Reclamation Permit for the Table Mountain Mine. Ministry of Energy and Mines (British Columbia).

Permit M-171, Reclamation Permit for the Blackdome Mine. Ministry of Energy and Mines (British Columbia).

Permit M-179, Reclamation Permit for the Premier Gold Mine. Ministry of Energy and Mines (British Columbia).

Permit M-187, Reclamation Permit for the Golden Bear Mine. Ministry of Energy and Mines (British Columbia).

Permit M-190, Reclamation Permit for the Snip Mine. Ministry of Energy and Mines (British Columbia).

Permit M-197, Reclamation Permit for the Eskay Creek Mine. Ministry of Energy and Mines (British Columbia).

Permit M-200, Reclamation Permit for the Mt. Polley Mine. Ministry of Energy and Mines (British Columbia).

Permit M-203, Reclamation Permit for the Huckleberry Mine. Ministry of Energy and Mines (British Columbia).

Permit M-206, Reclamation Permit for the Kemess South Mine. Ministry of Energy and Mines (British Columbia).

Price, W.A., 1997. Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia (Draft). Province of British Columbia, Reclamation Section, Energy and Minerals Division, Ministry of Employment and Investment, April 1997.

Ross, A.H. (ed), 1985. Hydrometallurgy. In: SME Mineral Processing Handbook, N.L. Weiss (ed.). Society of Mining Engineers, New York, NY, 1985.

Runge, I.C., 1998. Mining Economics and Strategy. Society for Mining, Metallurgy, and Exploration, Inc., Littleton, CO, USA, 295 pp.

Scott, C.C., 2002. SRK Consulting. Personal communication.

SENES Consultants Ltd., 1994. Acid Mine Drainage – Status of Chemical Treatment and Sludge Management Practices. Prepared for The Mine Environment Neutral Drainage (MEND) Program, June 1994.

Steffen Robertson and Kirsten (SRK), 1989. Draft Acid Rock Drainage Technical Guide. Prepared for the BC Acid Mine Drainage Task Force, Volume I.

SRK Consulting, 2002. Study of Management Alternatives – Giant Mine Arsenic Trioxide Dust. Report 1CI001.10 prepared for Department of Indian Affairs and Northern Development, December 2002.

Tremblay, G.A., 2000. The Canadian Mine Environment Neutral Drainage 2000 (MEND 2000) Program. In: Proceedings from the Fifth International Conference on Acid Rock Drainage. Denver, CO. Vol. 1, pp. 33-40.

Zinck, J.M. and Griffith, W.F., 2000. An Assessment of HDS-Type Lime Treatment Processes – Efficiency and Environmental Impact. In: Proceedings from the Fifth International Conference on Acid Rock Drainage. Denver, CO. Vol. II, pp. 1027-1034.

APPENDIX A
EXAMPLE CALCULATION OF AN HDS WATER TREATMENT PLANT DESIGN USING UNPUBLISHED DATA BY HUMBER (1996)

HDS Process Design

Use:
1. Complete solids generation w/ recycle estimate
2. Complete operating parameters
3. Complete process design
4. Set pond requirements
5. Complete Process Flowsheet and Equipment List

Base Conventional
4000 L/min

11 August, 2002
Pre-Feasibility
Rev. #1

Enter the Process Data on this sheet:

General Design Information

Design Flowrate:	4,000 L/min
Solids Generation	5.9 g/L plant feed
Recycle Ratio	11 (?:1)
Solids SG	2.8
Feed pH	3.5
Rapid Mix Tank pH	13.5 pH Units
Reactor pH	9.3 pH Units
Clarifier U/F Density	25 %
Clarifier Overflow Solids	0 mg/L

Aeration Requirements

Feed Iron Content	152 mg/L
Percentage Ferrous Iron	100 %
Average Density of Air	1.201 kg/m^3
Oxygen Transfer Efficiency	20 %

Vessel Residence Times:

Reactor Residence Time	40 minutes
Lime Sludge Mix Tank	4 minutes
Rapid Mix Tank	3 minutes
Flocculant Tank	2 minutes
Clarifier Upflow Ratio	1.200 $(m^3/hr)/m^2$
Recycle Water Tank	0.5 minutes

Flocculant Dosing System

Flocculant Dose Rate	100 mg floc/kg solids (range from 50 to 200)
Flocculant Addition Rate	7.0 mg floc/L plant feed (range from 1 to 10)
Undiluted Floc Concentration	0.5 %

Lime Dosing System

Lime Addition Rate (as $Ca(OH)_2$)	3.4 g lime/L plant feed
Lime Slurry Concentration	15 %
Slurry pH	14 pH Units
Solids SG	2.4
Storage Requirements	6 hours
Available CaO	90.0 %
Lime use	2.9 g lime (CaO)/L plant feed

Ferrous/Ferric Iron Dosing System

Fe^{2+}/Fe^{3+} Dosage (15%)	0 mg Fe^{2+}/Fe^{3+}/L Feed
Available Iron	23 %

Operating Costs

Lime Cost	100 US$/tonne		
Flocculant Cost	3600 US$/tonne		
Ferric/Ferrous Sulphate Cost	140 US$/tonne		
Power Cost	200 hp	0.05	US$/kw-hour
Manpower Cost	8 man-hours/day	24	US$/man-hour
O&M Capital	3 % of capital cost	2,300,000	US$ total capital

Water Quality and Sludge Generation Prediction

HDS Process Design
Base Conventional
4000 L/min
11 August, 2002

Ion	Ion Wt. (g/mol)	Hydroxide Formula	Hydroxide Weight (g/mol)	Mass of Ion Present (mg/L)	Mass of OH^- (mg/L)	Mass of Precip. (mg/L)
Al	26.98	$Al(OH)_3$	78.01	10.00	18.91	28.91
Ag	107.87	AgOH	124.88	0.10	0.02	0.12
As	74.92	$As(OH)_3$	125.95	0.57	0.39	0.96
Bi	208.98	$Bi(OH)_3$	260.01	0.00	0.00	0.00
Ca	40.08	$Ca(OH)_2$	74.1	55.00	0.00	0.00
Cd	112.41	$Cd(OH)_2$	146.43	0.15	0.05	0.20
Cu	63.55	$Cu(OH)_2$	97.57	35.00	18.74	53.74
Fe	55.85	$Fe(OH)_3$	106.88	152.00	138.88	290.88
Pb	207.2	$Pb(OH)_2$	241.22	26.00	4.27	30.27
Mg	24.31	$Mg(OH)_2$	58.33	14.00	19.59	33.59
Mn	54.94	MnO_2	86.94	25.00	0.00	39.56
Ni	58.71	$Ni(OH)_2$	92.73	4.20	2.43	6.63
S*	32.06	$CaSO_4.2H_2O$	172.18	0.00	0.00	0.00
Sb	121.75	$Sb(OH)_3$	172.78	0.00	0.00	0.00
Se	78.96	$Se(OH)_4$	147	0.00	0.00	0.00
Si	28.09	$Si(OH)_2$	62.11	0.00	0.00	0.00
Zn	65.38	$Zn(OH)_2$	99.4	54.00	28.10	82.10
SO_4^{2-}*	96.06	$CaSO_4.2H_2O$	172.18	3500.00	0.00	3853.71
CO_3^{2-}	59.98	$CaCO_3$	100.06	123.00	0.00	205.19
TSS	n/a	n/a	n/a	n/a	n/a	263.00
Total					**231.37**	**4888.85**

Residual SO_4^{2-} concentration 1350 mg/L (pure solubility range from 1240 - 1435 mg/L)

*** Use either (S) or (SO4).**

Solids Generation = 5.9 g/L
(includes 10.0 % lime enerts)
(includes 20.0 % unreacted lime solids)

Lime Requirements

Based on calcium requirements	2.75	g Ca(OH)2/L effluent OR	(SO_4^{2-} based)
	0.00	g Ca(OH)2/L effluent	(S based)
Based on hydroxide requirements	0.50	g Ca(OH)2/L effluent	

Lime Utilization = 80.0 %
Available CaO = 90.0 %

Lime use = 3.4 g $Ca(OH)_2$/L effluent
Lime use = 2.9 g lime (CaO)/L effluent

Sludge Quality Prediction

HDS Process Design
Base Conventional
4000 L/min
11 August, 2002

Ion	Mass of Ion Present (mg/L)	Mass of OH^- (mg/L)	Mass of Precip. (mg/L)	Mass of Metal (mg/L)	Sludge Composition (%)
Al	10.00	18.91	28.91	10.00	0.17
Ag	0.10	0.02	0.12	0.10	0.00
As	0.57	0.39	0.96	0.57	0.01
Bi	0.00	0.00	0.00	0.00	0.00
Ca	55.00	0.00	0.00	0.00	0.00
Cd	0.15	0.05	0.20	0.15	0.00
Cu	35.00	18.74	53.74	35.00	0.60
Fe	152.00	138.88	290.88	152.00	2.59
Pb	26.00	4.27	30.27	26.00	0.44
Mg	14.00	19.59	33.59	14.00	0.24
Mn	25.00	0.00	39.56	25.00	0.43
Ni	4.20	2.43	6.63	4.20	0.07
$CaSO_4.2H_2O$	0.00	0.00	0.00	n/a	0.00
Sb	0.00	0.00	0.00	0.00	0.00
Se	0.00	0.00	0.00	0.00	0.00
Si	0.00	0.00	0.00	0.00	0.00
Zn	54.00	28.10	82.10	54.00	0.92
$CaSO_4.2H_2O$	3500.00	0.00	3853.71	n/a	65.70
$CaCO_3$	123.00	0.00	205.19	n/a	3.50
TSS	n/a	n/a	263.00	n/a	4.48
Lime Inerts	n/a	n/a	976.61	n/a	16.65
Total		**231.37**	**5865.47**	**321.02**	**95.81**

Balance Check: 100.00 %

Solids generation = 5.9 g/L
Ultimate drained percent solids = 55 %
Sludge pond lifetime = 20 years

Annual Average Data:

Plant feed rate =	3,750 L/minute	
Total dry solids production =	31.7 tonnes/day	11560.8 tonnes/year
Sludge volume purged =	106.3 m^3/day	38811.4 m^3/year
Volume at ultimate density =	37.2 m^3/day	13587.7 m^3/year
Pond volume required =	272,000 m^3	

Vessel Sizes

			Tank Dimensions (no freeboard included)								aspect ratio
Lime Sludge Mix Tank:	4 m^3 =	1014 USgal	D =	2.0 m	or	6.6 ft	H =	1.2 m	or	4.0 ft	1.64
Rapid Mix Tank:	15 m^3 =	3932 USgal	D =	3.0 m	or	9.8 ft	H =	2.1 m	or	6.9 ft	1.42
Reactor Vessel:	198 m^3 =	52427 USgal	D =	7.0 m	or	23.0 ft	H =	5.2 m	or	16.9 ft	1.36
Flocculation Tank:	10 m^3 =	2658 USgal	D =	2.5 m	or	8.2 ft	H =	2.0 m	or	6.7 ft	1.22
Clarifier Diameter:	16 m =	53 ft									
Lime Storage Tank:	33 m^3 =	8839 USgal	H =	4.0 m	or	13.1 ft	W=L =	2.9 m	or	9.5 ft	1.38
Recycled Water Tank:	2 m^3 =	540 USgal	D =	1.5 m	or	4.9 ft	H =	1.2 m	or	3.8 ft	1.30

Aeration Requirements

Total Iron Content =	152 mg/L
Percent Ferrous Iron =	100 %
Oxygen Transfer Efficiency =	20 %
Total Flow In =	4000 L/min
Total Ferrous Iron =	0.6 kg/min
=	36.5 kg/hr
Aeration required =	**104** m^3/hour
=	**61** SCFM

Sludge and Reagent Flowrates

Sludge Purge and Recycle

Sludge Purge Data			Sludge Recycle Data		
Sludge Purge = Solids Generation =	23 kg/min =	52 lbs/min	Solids Recycled =	258 kg/min =	569 lbs/min
Solids Volume =	8 L/min =	2 USgpm	Solids Volume =	92 L/min =	24 USgpm
Water Flow =	70 L/min =	19 USgpm	Water Flow =	774 L/min =	205 USgpm
Total Flow =	79 L/min =	21 USgpm	Total Flow =	866 L/min =	229 USgpm
SG Slurry =	1.19		SG Slury =	1.19	
pH Slurry =	9.3 pH Units		pH Slurry =	9.3 pH Units	
SG Solids =	2.8		SG Solids =	2.8	
Slurry % Solids =	25.00 %		Slurry % Solids =	25.00 %	

Lime Circuit

Lime Dosing			Lime Loop Out Of Storage Tank			Lime Loop Return To Storage Tank		
Solids Mass =	15 kg/min =	34 lbs/min	Solids Mass =	61 kg/min =	135 lbs/min	Solids Mass =	46 kg/min =	101 lbs/min
Solids Volume =	6 L/min =	2 USgpm	Solids Volume =	25 L/min =	7 USgpm	Solids Volume =	19 L/min =	5 USgpm
Water Flow =	87 L/min =	23 USgpm	Water Flow =	346 L/min =	91 USgpm	Water Flow =	260 L/min =	69 USgpm
Total Slurry Flow =	93 L/min =	25 USgpm	Total Slurry Flow =	372 L/min =	98 USgpm	Total Slurry Flow =	279 L/min =	74 USgpm
Slurry SG =	1.10		Slurry SG =	1.10		Slurry SG =	1.10	
pH Slurry =	14 pH Units		pH Slurry =	14 pH Units		pH Slurry =	14 pH Units	
SG Solids =	2.4		SG Solids =	2.4		SG Solids =	2.4	
Slurry % Solids =	15.00 %		Slurry % Solids =	15.00 %		Slurry % Solids =	15.00 %	

Flocculant Dosing			**Iron Sulphate Dosing**			**Lime Dosing**	
Floc Dosing Rate =	7 mg/L effluent treated		Fe 2+/3+ Dosing Rate =	0 mg Fe^{2+}/Fe^{3+}/L effluent treated		Lime Dosing Rate =	3.4 g lime/L effluent treated
Flow Into Floc Tank =	4961 L/min =	1311 USgpm	Plant Feed Rate =	4000 L/min =	1057 USgpm	Lime Dosing Rate =	2.9 g lime (CaO + inerts)/L \treated
Undiluted Floc Flowrate =	7 L/min =	2 USgpm	Solution Concentration =	15 %		Average Plant Feed =	4000 L/minute
Diluted Floc Flowrate =	70 L/min =	18 USgpm	Solution Flowrate =	0 L/min =	0 USgpm	Daily Consumption =	16.7 tonnes/day
Floc Consumption =	50 kg/day =	111 lbs/day	Iron Sulphate Consumption =	0 kg/day =	0 lbs/day	Annual Consumption=	6078 tonnes/year quicklime

Tank Flows

Out Of Lime/Sludge Mix Tank

Solids Mass =	273 kg/min =	603 lbs/min
Solids Volume =	99 L/min =	26 USgpm
Water Flow =	861 L/min =	227 USgpm
Total Slurry Flow =	959 L/min =	253 USgpm
Slurry SG =	1.18	
pH Slurry =	13.5 pH Units	
SG Solids =	2.77	
Slurry % Solids =	24.10 %	

Out Of Rapid Mix Tank

Solids Mass =	282 kg/min =	621 lbs/min
Solids Volume =	101 L/min =	27 USgpm
Water Flow =	4861 L/min =	1284 USgpm
Total Slurry Flow =	4961 L/min =	1311 USgpm
Slurry SG =	1.04	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	5.47 %	

Out Of Reactor Tank

Solids Mass =	281.54 kg/min =	620.80 lbs/min
Solids Volume =	101 L/min =	27 USgpm
Water Flow =	4861 L/min =	1284 USgpm
Total Slurry Flow =	4961 L/min =	1311 USgpm
Slurry SG =	1.04	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	5.47 %	

Out Of Flocculant Tank

Solids Mass =	282 kg/min =	621 lbs/min
Solids Volume =	101 L/min =	27 USgpm
Water Flow =	4931 L/min =	1303 USgpm
Total Slurry Flow =	5031 L/min =	1329 USgpm
Slurry SG =	1.04	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	5.40 %	

Clarifier Flows

Clarifier Overflow

Solids Mass =	0 kg/min =	0 lbs/min
Solids Volume =	0 L/min =	0 USgpm
Water Flow =	4086 L/min =	1079 USgpm
Total Slurry Flow =	4086 L/min =	1079 USgpm
Slurry SG =	1	
pH Slurry =	9.3 pH Units	
SG Solids =	n/a	
Slurry % Solids =	0 %	

Clarifier Underflow

Solids Mass =	282 kg/min =	621 lbs/min
Solids Volume =	101 L/min =	27 USgpm
Water Flow =	845 L/min =	223 USgpm
Total Slurry Flow =	945 L/min =	250 USgpm
Slurry SG =	1.19	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	25.00 %	

Balance Check (Overall)

Total Solids In =	23.46 kg/min	Total Water In =	4156 L/min
Total Solids Out =	23.46 kg/min	Total Water Out =	4156 L/min
% Deviation =	0.00 %	% Deviation =	0.00 %

Balance Check (Clarifier)

Total Solids In =	282 kg/min	Total Water In =	4931 L/min
Total Solids Out =	282 kg/min	Total Water Out =	4931 L/min
% Deviation =	0.00 %	% Deviation =	0.00 %

Operating Cost Estimate

Base Conventional
4000 L/min
11 August, 2002

Reagent	Dose Rate (mg/L plant feed)	Annual Average Plant Flow Rate (L/min)	Annual Reagent Consumption (tonnes/year)	Reagent Unit Cost (US$/tonne)	Annual Reagent Cost (US$/year)
Quicklime	2891	3,750	5698	100	570,000
Flocculant	7	3,750	14	3600	50,000
Iron Sulphate	0	3,750	0	140	0
				Sub-total:	**$620,000**

Item	Annual Consumption	Unit Cost (US$)	Annual Cost (US$/year)
Electric Power	1.31 million kW-hours	0.05	65,000
O & M Capital	3 % of capital cost	2300000	69,000
O & M Manpower	8 man-hours per day	24	70,000
		Sub-total:	**$204,000**

Total Annual Operating Cost:	**$824,000**	**/year**	(US dollars)
Normalized Annual Operating Cost:	**$0.42**	**/m^3**	(US dollars)
	$1.58	**/1000 USgal**	(US dollars)

Discount Interest Rate:	10%	
Expected Plant Lifetime:	20	years
Present Value of Plant Operating Costs:	**$7,015,000**	**US dollars**
Net Present Value of Plant:	**$9,315,000**	**US dollars**

APPENDIX B
EXAMPLE CALCULATION OF A SIMPLE LIME NEUTRALIZATION WATER TREATMENT PLANT DESIGN USING UNPUBLISHED DATA BY HUMBER (1996)

Simple Lime Neutralization Process Design

Use:
1. Complete solids generation
2. Complete operating parameters
3. Complete process design
4. Set pond requirements
5. Complete Process Flowsheet and Equipment List

Base Process
3000 L/min

11 August, 2002
Pre-Feasibility
Rev. #1

Enter the Process Data on this sheet:

General Design Information

Design Flowrate:	3,000 L/min
Solids Generation	487.8 g/L Effluent Treated
Solids SG	2.8
Feed pH	-7.0 pH Units
Flash Mix Tank pH	7 pH Units
Reactor Vessel pH	9.3 pH Units
Settling Pond Solids Density	50 %

Aeration Requirements

Feed Iron Content	6000 mg/L
Percentage Ferrous Iron	0 %
Average Density of Air	1.201 kg/m^3
Oxygen Transfer Efficiency	20 %

Vessel Residence Times:

Flash Mix Tank Res. Time	10 minutes
Reactor Vessel Res. Time	30 minutes
Flocculator Tank	3 minutes

Flocculant Dosing System

Flocculant Dose Rate	100 mg floc/kg solids (range from 50 to 200)
Flocculant Addition Rate	48.8 mg floc/L effluent treated (range from 1 to 10)
Undiluted Floc Concentration	0.5 %

Lime Dosing System

Lime Addition Rate (as $Ca(OH)_2$)	93.9 g Lime/L Effluent Treated
Lime Slurry Concentration	15 % solids
Slurry pH	14 pH Units
Solids SG	2.4
Storage Requirements	6 hours
Available CaO	90.0 %
Lime use	78.9 g lime (CaO)/L effluent

Operating Costs

Lime Cost	100 US$/tonne	
Flocculant Cost	3600 US$/tonne	
Power Cost	100 hp	0.05 US$/kw-hour
Manpower Cost	8 man-hours/day	24 US$/man-hour
O&M Capital	3 % of capital cost	4,000,000 US$ total capital

Water Quality and Sludge Generation Prediction

Simple Lime Neutralization Process Design
Base Process
3000 L/min
11 August, 2002

Ion	Ion Wt. (g/mol)	Hydroxide Formula	Hydroxide Weight (g/mol)	Mass of Ion Present (mg/L)	Mass of OH^- (mg/L)	Mass of Precip. (mg/L)
Al	26.98	$Al(OH)_3$	78.01	0.00	0.00	0.00
Ag	107.87	AgOH	124.88	0.00	0.00	0.00
As	74.92	$As(OH)_3$	125.95	0.00	0.00	0.00
Bi	208.98	$Bi(OH)_3$	260.01	0.00	0.00	0.00
Ca	40.08	$Ca(OH)_2$	74.1	0.00	0.00	0.00
Cd	112.41	$Cd(OH)_2$	146.43	0.00	0.00	0.00
Cu	63.55	$Cu(OH)_2$	97.57	0.00	0.00	0.00
Fe	55.85	$Fe(OH)_3$	106.88	6000.00	5482.18	11482.18
Pb	207.2	$Pb(OH)_2$	241.22	0.00	0.00	0.00
Mg	24.31	$Mg(OH)_2$	58.33	0.00	0.00	0.00
Mn	54.94	MnO_2	86.94	0.00	0.00	0.00
Ni	58.71	$Ni(OH)_2$	92.73	0.00	0.00	0.00
S*	32.06	$CaSO_4.2H_2O$	172.18	0.00	0.00	0.00
Sb	121.75	$Sb(OH)_3$	172.78	0.00	0.00	0.00
Se	78.96	$Se(OH)_4$	147	0.00	0.00	0.00
Si	28.09	$Si(OH)_2$	62.11	0.00	0.00	0.00
Zn	65.38	$Zn(OH)_2$	99.4	0.00	0.00	0.00
SO_4^{2-}*	96.06	$CaSO_4.2H_2O$	172.18	96000.00	0.00	169670.61
CO_3^{2-}	59.98	$CaCO_3$	100.06	0.00	0.00	0.00
TSS	n/a	n/a	n/a	n/a	n/a	280000.00
Total					**5482.18**	**461152.79**

Residual SO_4^{2-} concentration 1340 mg/L (pure solubility range from 1240 - 1435 mg/L)

*** Use either (S) or (SO4).**

Solids Generation = 487.8 g/L
(includes 10.0 % lime enerts)
(includes 20.0 % unreacted lime solids)

Lime Requirements

Based on calcium requirements	75.09	g Ca(OH)2/L effluent	**OR**	(SO_4^{2-} based)
	0.00	g Ca(OH)2/L effluent		(S based)
Based on hydroxide requirements	11.94	g Ca(OH)2/L effluent		

Lime Utilization = 80.0 %
Available CaO = 90.0 %

Lime use = 93.9 g $Ca(OH)_2$/L effluent
Lime use = 78.9 g lime (CaO)/L effluent

Sludge Quality Prediction

Simple Lime Neutralization Process Design
Base Process
3000 L/min
11 August, 2002

Ion	Mass of Ion Present (mg/L)	Mass of OH^- (mg/L)	Mass of Precip. (mg/L)	Mass of Metal (mg/L)	Sludge Composition (%)
Al	0.00	0.00	0.00	0.00	0.00
Ag	0.00	0.00	0.00	0.00	0.00
As	0.00	0.00	0.00	0.00	0.00
Bi	0.00	0.00	0.00	0.00	0.00
Ca	0.00	0.00	0.00	0.00	0.00
Cd	0.00	0.00	0.00	0.00	0.00
Cu	0.00	0.00	0.00	0.00	0.00
Fe	6000.00	5482.18	11482.18	6000.00	1.23
Pb	0.00	0.00	0.00	0.00	0.00
Mg	0.00	0.00	0.00	0.00	0.00
Mn	0.00	0.00	0.00	0.00	0.00
Ni	0.00	0.00	0.00	0.00	0.00
$CaSO_4.2H_2O$	0.00	0.00	0.00	n/a	0.00
Sb	0.00	0.00	0.00	0.00	0.00
Se	0.00	0.00	0.00	0.00	0.00
Si	0.00	0.00	0.00	0.00	0.00
Zn	0.00	0.00	0.00	0.00	0.00
$CaSO_4.2H_2O$	96000.00	0.00	169670.61	n/a	34.78
$CaCO_3$	0.00	0.00	0.00	n/a	0.00
TSS	n/a	n/a	280000.00	n/a	57.40
Lime Inerts	n/a	n/a	26664.52	n/a	5.47
Total		**5482.18**	**487817.32**	**6000.00**	**98.88**

Balance Check: 100.00 %

Solids generation = 487.8 g/L
Ultimate sludge percent solids = 50 %
Sludge pond lifetime = 20 years

Annual Average Data:

Plant feed rate =	2,800 L/minute	
Total dry solids production =	1966.9 tonnes/day	717911.0 tonnes/year
Sludge volume accumulated =	2669.3 m^3/day	974307.8 m^3/year
Volume at ultimate density =	2669.3 m^3/day	974307.8 m^3/year
Pond volume required =	19,486,000 m^3	

Vessel Sizes

			Tank Dimensions (no freeboard included)										aspect ratio
Flash Mix Tank:	53 m^3 =	13990 USgal	D =	4.6 m	or	15.1 ft	H =	3.2 m	or	10.5 ft	1.44
Reactor Tank:	159 m^3 =	41969 USgal	D =	6.5 m	or	21.3 ft	H =	4.8 m	or	15.7 ft	1.36
Flocculation Tank:	16 m^3 =	4192 USgal	D =	3.0 m	or	9.8 ft	H =	2.2 m	or	7.4 ft	1.34
Lime Storage Tank:	685 m^3 =	181007 USgal	H =	10.0 m	or	32.8 ft	W=L =	8.3 m	or	27.2 ft	1.21

Aeration Requirements

Total Iron Content =	6000 mg/L
Percent Ferrous Iron =	0 %
Oxygen Transfer Efficiency =	20 %
Total Flow In =	3000 L/min
Total Ferrous Iron =	0.0 kg/min
=	0.0 kg/hr
Aeration required =	**0** m^3/hour
=	**0** SCFM

Sludge and Reagent Flowrates

Sludge Pond Accumulation Rate

Solids Generation =	1463 kg/min =	3227 lbs/min
Solids Volume =	523 L/min =	138 USgpm
Interstitial Water Flow =	1463 L/min =	387 USgpm
Total Accumulation Rate =	1986 L/min =	525 USgpm
SG Slurry =	1.47	
pH Slurry =	9.3 pH Units	
SG Solids =	2.8	
Slurry % Solids =	50.00 %	

Lime Circuit

Lime Dosing

Solids Mass =	313 kg/min =	690 lbs/min
Solids Volume =	130 L/min =	34 USgpm
Water Flow =	1773 L/min =	468 USgpm
Total Slurry Flow =	1903 L/min =	503 USgpm
Slurry SG =	1.10	
pH Slurry =	14 pH Units	
SG Solids =	2.4	
Slurry % Solids =	15.00 %	

Lime Loop Out Of Storage Tank

Solids Mass =	1251 kg/min =	2759 lbs/min
Solids Volume =	521 L/min =	138 USgpm
Water Flow =	7092 L/min =	1873 USgpm
Total Slurry Flow =	7613 L/min =	2011 USgpm
Slurry SG =	1.10	
pH Slurry =	14 pH Units	
SG Solids =	2.4	
Slurry % Solids =	15.00 %	

Lime Loop Return To Storage Tank

Solids Mass =	939 kg/min =	2070 lbs/min
Solids Volume =	391 L/min =	103 USgpm
Water Flow =	5319 L/min =	1405 USgpm
Total Slurry Flow =	5710 L/min =	1508 USgpm
Slurry SG =	1.10	
pH Slurry =	14 pH Units	
SG Solids =	2.4	
Slurry % Solids =	15.00 %	

Flocculant Dosing

Floc Dosing Rate =	49 mg/L effluent treated	
Flow Into Floc Tank =	5296 L/min =	1399 USgpm
Undiluted Floc Flowrate =	52 L/min =	14 USgpm
Diluted Floc Flowrate =	517 L/min =	136 USgpm
Floc Consumption =	372 kg/day =	820 lbs/day

Lime Dosing

Lime Dosing Rate =	93.9 g lime/L effluent treated
Lime Dosing Rate =	78.9 g lime (CaO + inerts)/L \treated
Average Plant Feed =	3000 L/minute
Daily Consumption =	341.0 tonnes/day
Annual Consumption=	124452 tonnes/year quicklime

Tank Flows

Out Of Flash Mix Tank

Solids Mass =	1463 kg/min =	3227 lbs/min
Solids Volume =	523 L/min =	138 USgpm
Water Flow =	4773 L/min =	1261 USgpm
Total Slurry Flow =	5296 L/min =	1399 USgpm
Slurry SG =	1.18	
pH Slurry =	7 pH Units	
SG Solids =	2.80	
Slurry % Solids =	23.47 %	

Out Of Reactor Tank

Solids Mass =	1463 kg/min =	3227 lbs/min
Solids Volume =	523 L/min =	138 USgpm
Water Flow =	4773 L/min =	1261 USgpm
Total Slurry Flow =	5296 L/min =	1399 USgpm
Slurry SG =	1.18	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	23.47 %	

Out Of Flocculator Tank

Solids Mass =	1463 kg/min =	3227 lbs/min
Solids Volume =	523 L/min =	138 USgpm
Water Flow =	5290 L/min =	1397 USgpm
Total Slurry Flow =	5812 L/min =	1535 USgpm
Slurry SG =	1.16	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	21.67 %	

Out Of Settling Pond

Solids Mass =	0 kg/min =	0 lbs/min
Solids Volume =	0 L/min =	0 USgpm
Water Flow =	5290 L/min =	1397 USgpm
Total Slurry Flow =	5290 L/min =	1397 USgpm
Slurry SG =	1.00	
pH Slurry =	9.3 pH Units	
SG Solids =	2.80	
Slurry % Solids =	0.00 %	

Operating Cost Estimate

Base Process
3000 L/min
11 August, 2002

Reagent	Dose Rate (mg/L treated)	Annual Average Plant Flow Rate (L/min)	Annual Reagent Consumption (tonnes/year)	Reagent Unit Cost (US$/tonne)	Annual Reagent Cost (US$/year)
Quicklime	78927	2,800	116154.9	100	11,615,000
Flocculant	49	2,800	71.8	3600	258,000
				Sub-total	**$11,873,000**

Item	Annual Consumption	Unit Cost (US$)	Annual Cost (US$/year)
Electric Power	0.65 million kW-hours	0.05	0
O & M Capital	3 % of capital cost	4,000,000	120,000
O & M Manpower	8 man-hours per day	24	70,000
		Sub-total:	**$190,000**

Total Annual Operating Cost	**$12,063,000**	**/year**	(US dollars)
Normalized Annual Operating Cost	**$8.20**	**/m^3**	(US dollars)
	$31.03	**/1000 USgal**	(US dollars)

APPENDIX C
DAM HEIGHT CALCULATION MACRO

Macro to Calculate Dam Height

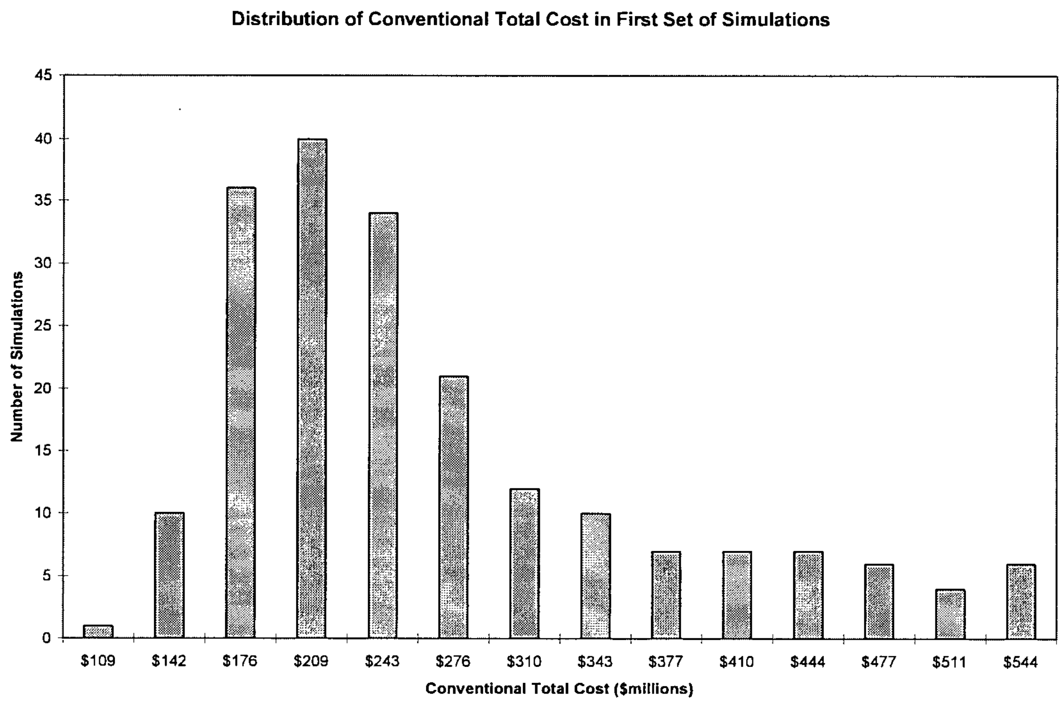
```
Sub Conventional_TMA()

'Declaring Variables for Conventional Tailings

Dim TailsVolume As Single  'Volume of tailings calculated in spreadsheet
Dim TMAVolume As Single    'Volume of TMA calculated in Macro
Dim DamHeight As Single
Dim FloorWidth As Single   'Width of TMA bottom - specified by user
Dim TMALength As Single
Dim WallSlope As Integer   'Slope angle of TMA sides - specified by user
Dim MaxHeight As Integer   'Maximum height of dam
Dim MaxLength As Integer   'Maximum length of TMA
Dim bEnough As Boolean
Dim n As Integer           'Counter for dam height
Dim i As Integer           'Counter for TMA length

'Declaring Variables for Autoclave Tailings

Dim TailsVolumea As Single  'Volume of tailings calculated in spreadsheet
Dim TMAVolumea As Single    'Volume of TMA calculated in Macro
Dim DamHeighta As Single
Dim FloorWidtha As Single   'Width of TMA bottom - specified by user
Dim TMALengtha As Single
Dim WallSlopea As Integer   'Slope angle of TMA sides - specified by user
Dim MaxHeighta As Integer   'Maximum height of dam
Dim MaxLengtha As Integer   'Maximum length of TMA
Dim bEnougha As Boolean
Dim j As Integer            'Counter for dam height
Dim k As Integer            'Counter for TMA length

'Initializing Variables

TailsVolume = Worksheets("TMA").Range("C24").Value
FloorWidth = Worksheets("TMA").Range("F33").Value
WallSlope = Worksheets("TMA").Range("F30").Value
MaxHeight = Worksheets("TMA").Range("F31").Value
MaxLength = Worksheets("TMA").Range("F37").Value
bEnough = False
TMAVolume = 0
DamHeight = 0
TMALength = 0
n = 1
i = 0

'Initializing Variables for Autoclaved Tailings

TailsVolumea = Worksheets("TMA").Range("L34").Value
FloorWidtha = Worksheets("TMA").Range("O44").Value
WallSlopea = Worksheets("TMA").Range("O41").Value
MaxHeighta = Worksheets("TMA").Range("O42").Value
MaxLengtha = Worksheets("TMA").Range("O48").Value
bEnougha = False
TMAVolumea = 0
```

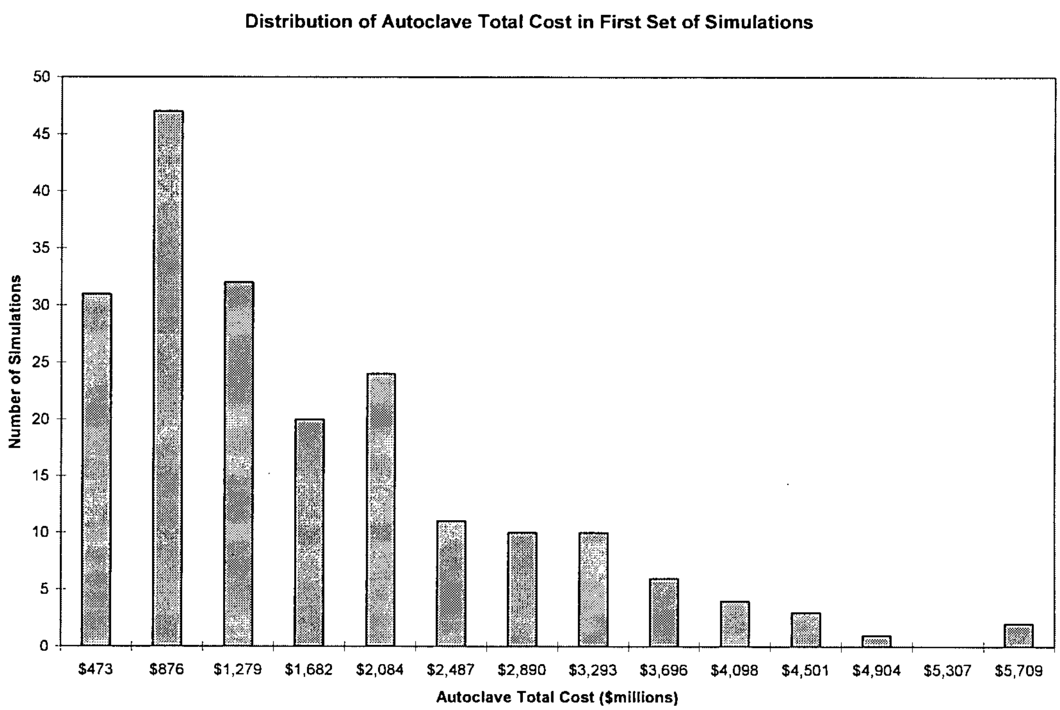
```
DamHeighta = 0
TMALengtha = 0
j = 1
k = 0

'Determining Dam Height and TMA Length for Conventional Tailings

For n = 1 To MaxHeight
  If bEnough = False Then
    DamHeight = n
    For i = 1 To MaxLength
      TMALength = i
      TMAVolume = (DamHeight * TMALength * FloorWidth) + (DamHeight * Tan((90 - WallSlope) *
        Pi / 180) * DamHeight * TMALength) + (0.5 * DamHeight * Tan((90 - WallSlope) * Pi / 180) *
        DamHeight * FloorWidth)
      If TMAVolume > TailsVolume Then
        bEnough = True
        Exit For
      End If
    Next i
  End If
Next n

'Determining Dam Height and TMA Length for Autocalved Tailings

For j = 1 To MaxHeighta
  If bEnougha = False Then
    DamHeighta = j
    For k = 1 To MaxLengtha
      TMALengtha = k
      TMAVolumea = (DamHeighta * TMALengtha * FloorWidtha) + (DamHeighta * Tan((90 -
        WallSlopea) * Pi / 180) * DamHeighta * TMALengtha) + (0.5 * DamHeighta * Tan((90 -
        WallSlopea) * Pi / 180) * DamHeighta * FloorWidtha)
      If TMAVolumea > TailsVolumea Then
        bEnougha = True
        Exit For
      End If
    Next k
  End If
Next j

Worksheets("TMA").Range("D45").Value = DamHeight
Worksheets("TMA").Range("D49").Value = TMALength
Worksheets("TMA").Range("M56").Value = DamHeighta
Worksheets("TMA").Range("M60").Value = TMALengtha
Worksheets("Record").Range("A6").Value = Worksheets("Input Parameters").Range("C3").Value
Worksheets("Record").Range("B6").Value = Worksheets("Input Parameters").Range("C4").Value
Worksheets("Record").Range("C6").Value = Worksheets("Input Parameters").Range("C5").Value
Worksheets("Record").Range("D6").Value = Worksheets("Input Parameters").Range("C6").Value
Worksheets("Record").Range("E6").Value = Worksheets("TMA").Range("C24").Value
Worksheets("Record").Range("F6").Value = TMAVolume
Worksheets("Record").Range("G6").Value = Worksheets("Cost Summary-
    conventional").Range("C14").Value
Worksheets("Record").Range("H6").Value = Worksheets("Cost Summary-
    conventional").Range("C22").Value
```

```
Worksheets("Record").Range("I6").Value = Worksheets("Cost Summary-
    conventional").Range("C24").Value
Worksheets("Record").Range("J6").Value = Worksheets("Cost Summary-autoclave").Range("C16").Value
Worksheets("Record").Range("K6").Value = Worksheets("Cost Summary-
    autoclave").Range("C27").Value
Worksheets("Record").Range("L6").Value = Worksheets("Cost Summary-autoclave").Range("C29").Value
zt = (1750 - 500) / (5000 - 500) 'daily tonnage rate
zm = (8 - 5) / (12 - 5)  'mine life
pt = (0.07 - 0.02) / (0.15 - 0.02) 'sulphide content in tailings
R = Rnd() 'generate a random number for dailing tonnage and mine life
Rp = Rnd()  'generates a random number for sulphide in tailings

If R <= zt Then
  Worksheets("Input Parameters").Range("C3").Value = 500 + (5000 - 500) * Sqr(R * zt)
Else
Worksheets("Input Parameters").Range("C3").Value = 500 + (5000 - 500) * (1 - Sqr((1 - R) * (1 - zt)))
End If

If R <= zm Then
  Worksheets("Input Parameters").Range("C42").Value = 5 + (12 - 5) * Sqr(R * zm)
Else
Worksheets("Input Parameters").Range("C42").Value = 5 + (12 - 5) * (1 - Sqr((1 - R) * (1 - zm)))
End If

If Rp <= pt Then
  Worksheets("Input Parameters").Range("C4").Value = 0.02 + (0.15 - 0.02) * Sqr(Rp * pt)
Else
Worksheets("Input Parameters").Range("C4").Value = 0.02 + (0.15 - 0.02) * (1 - Sqr((1 - Rp) * (1 - pt)))
End If

End Sub
```

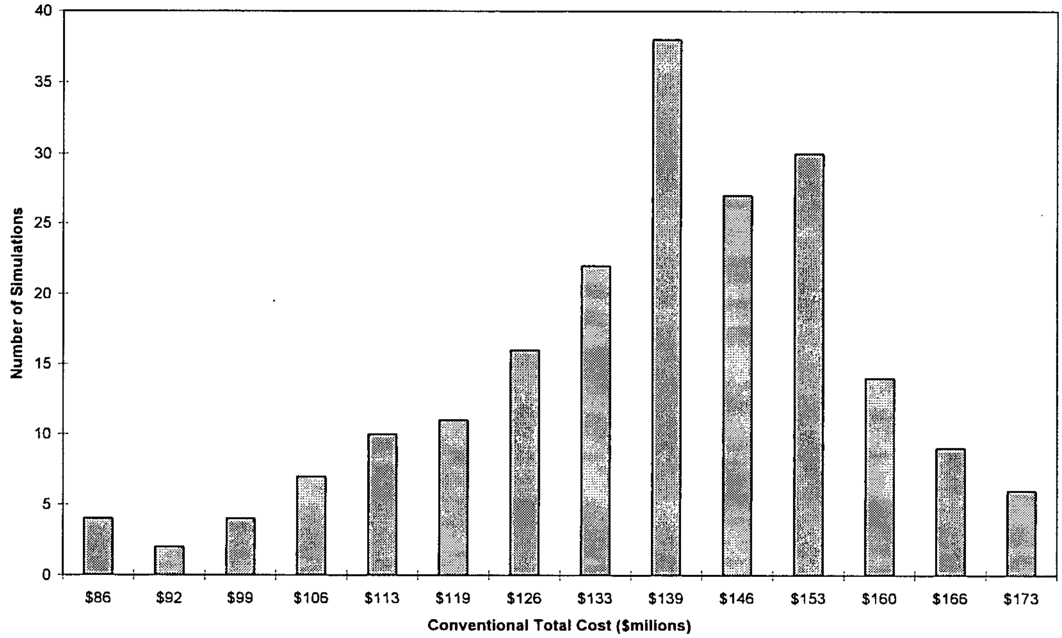
APPENDIX D
SIMULATION RESULTS

Distribution of Simulation Results





Distribution of Conventional Total Cost in Second Set of Simulations



Distribution of Autoclave Total Cost in Second Set of Simulations

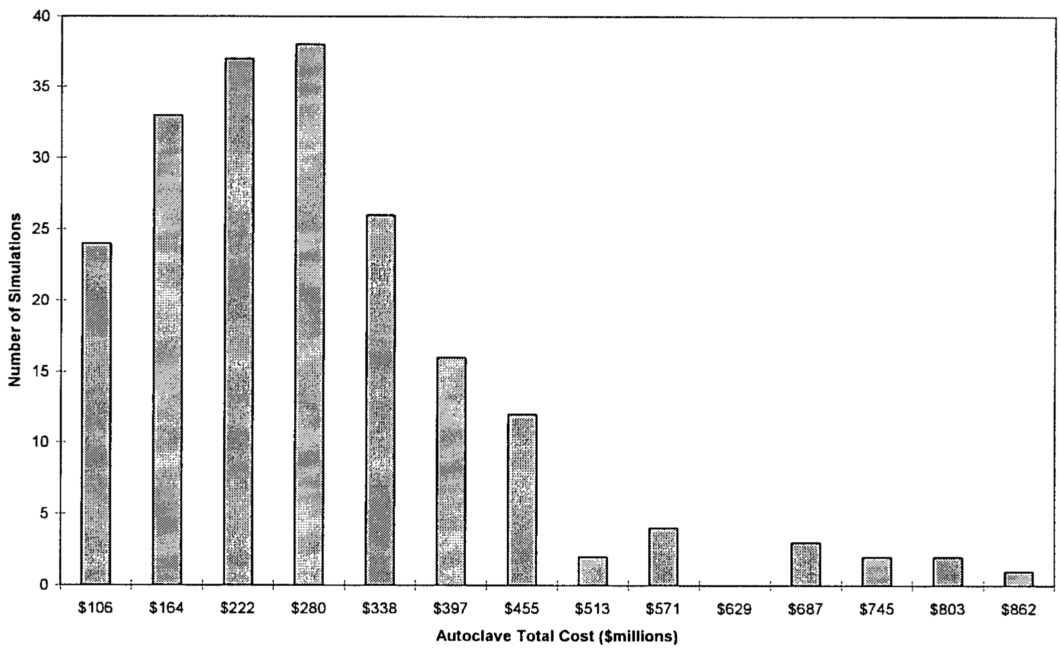


Table D-1
Simulation Results

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
5	741	5.14%	$94.1	$0.6	$94.7	$18.5	$31.1	$49.6
5	650	7.27%	$80.2	$0.5	$80.8	$19.0	$38.1	$57.1
5	597	10.40%	$74.7	$0.5	$75.2	$21.4	$49.2	$70.7
5	730	12.80%	$92.4	$0.6	$93.0	$22.1	$74.0	$96.1
6	1166	3.70%	$87.5	$0.8	$88.3	$18.1	$42.2	$60.3
6	976	3.88%	$133.3	$0.8	$134.2	$15.8	$32.1	$47.9
6	825	5.15%	$107.6	$0.7	$108.3	$20.4	$34.7	$55.1
6	1079	5.98%	$79.1	$0.7	$79.8	$20.8	$60.1	$80.9
6	1069	6.23%	$78.1	$0.7	$78.8	$21.0	$62.0	$83.0
6	1144	6.37%	$85.4	$0.7	$86.1	$22.4	$67.7	$90.2
6	783	6.38%	$100.7	$0.7	$101.4	$21.1	$40.4	$61.5
6	1076	6.50%	$78.8	$0.7	$79.5	$21.5	$65.0	$86.5
6	809	6.69%	$105.0	$0.7	$105.7	$22.2	$43.8	$66.0
6	992	7.14%	$136.0	$0.9	$136.9	$20.9	$57.1	$78.0
6	993	7.50%	$136.3	$0.9	$137.1	$21.4	$60.0	$81.4
6	1224	8.10%	$93.4	$0.8	$94.2	$26.8	$91.6	$118.4
6	1233	8.68%	$94.3	$0.8	$95.1	$27.8	$98.7	$126.5
6	963	8.95%	$131.1	$0.8	$131.9	$22.6	$69.2	$91.8
6	769	8.97%	$98.6	$0.7	$99.2	$24.5	$55.3	$79.9
6	907	9.57%	$121.2	$0.8	$122.0	$22.3	$69.5	$91.8
6	899	10.77%	$119.9	$0.8	$120.6	$23.7	$76.7	$100.4
6	997	11.56%	$137.0	$0.9	$137.9	$26.6	$91.3	$117.9
7	1386	3.99%	$110.5	$0.9	$111.4	$21.8	$53.7	$75.5
7	1751	4.38%	$105.5	$1.1	$106.5	$24.2	$82.7	$106.9
7	1497	4.66%	$122.9	$1.0	$123.9	$21.6	$67.0	$88.7
7	1688	5.25%	$145.4	$1.3	$146.6	$25.0	$92.8	$117.8
7	1599	5.72%	$134.7	$1.2	$135.9	$24.7	$95.4	$120.1
7	1518	5.92%	$125.2	$1.0	$126.2	$24.0	$83.9	$107.8
7	1428	6.00%	$115.1	$0.9	$116.1	$26.6	$79.9	$106.5
7	1403	6.09%	$112.4	$0.9	$113.3	$26.3	$79.6	$106.0
7	1519	6.29%	$125.3	$1.0	$126.3	$25.1	$88.9	$114.0
7	1286	6.35%	$99.8	$0.8	$100.6	$24.8	$75.9	$100.7
7	1581	6.87%	$132.5	$1.2	$133.7	$26.9	$112.6	$139.5
7	1780	6.87%	$108.0	$1.1	$109.1	$29.8	$126.9	$156.7
7	1758	6.89%	$106.1	$1.1	$107.2	$29.5	$125.7	$155.2
7	1555	6.94%	$129.6	$1.2	$130.7	$26.7	$111.9	$138.6

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary Capital ($millions)	Conventional Operating ($millions)	Conventional Total ($millions)	Autoclave Cost Summary Capital ($millions)	Autoclave Operating ($millions)	Autoclave Total ($millions)
7	1759	7.21%	$106.2	$1.1	$107.2	$30.5	$131.3	$161.8
7	1312	7.27%	$102.5	$0.9	$103.4	$27.2	$88.4	$115.5
7	1526	7.76%	$126.2	$1.0	$127.2	$27.6	$109.5	$137.0
7	1728	7.85%	$103.6	$1.0	$104.6	$31.2	$140.2	$171.3
7	1705	8.16%	$147.4	$1.3	$148.7	$31.4	$143.6	$174.9
7	1621	8.41%	$137.3	$1.2	$138.5	$30.5	$140.7	$171.1
7	1651	8.49%	$140.9	$1.2	$142.2	$31.1	$144.7	$175.7
7	1578	8.57%	$132.2	$1.2	$133.4	$30.0	$139.4	$169.4
7	1438	8.91%	$116.3	$1.0	$117.2	$28.4	$118.1	$146.4
7	1728	9.22%	$103.6	$1.0	$104.6	$33.5	$164.0	$197.5
7	1665	9.29%	$142.6	$1.3	$143.9	$32.6	$159.2	$191.8
7	1491	9.36%	$122.1	$1.0	$123.1	$29.9	$128.5	$158.4
7	1700	9.55%	$146.8	$1.3	$148.1	$33.9	$167.0	$200.9
7	1609	9.71%	$135.9	$1.2	$137.1	$32.3	$160.7	$193.1
7	1775	9.93%	$107.6	$1.1	$108.6	$32.9	$181.1	$214.0
7	1684	10.52%	$144.9	$1.3	$146.2	$35.1	$180.4	$215.5
7	1772	10.87%	$107.3	$1.1	$108.4	$34.2	$196.2	$230.4
7	1681	12.73%	$144.6	$1.3	$145.8	$35.7	$217.6	$253.3
8	2007	2.42%	$127.4	$1.2	$128.7	$21.8	$55.3	$77.1
8	2203	3.44%	$144.4	$1.5	$145.9	$24.4	$91.3	$115.6
8	1958	3.86%	$123.2	$1.2	$124.4	$25.6	$82.2	$107.8
8	2312	3.88%	$119.1	$1.4	$120.4	$26.6	$107.0	$133.6
8	1800	4.17%	$109.7	$1.1	$110.8	$24.4	$81.3	$105.7
8	2076	4.18%	$133.4	$1.3	$134.7	$24.9	$93.9	$118.9
8	1852	4.19%	$114.1	$1.1	$115.3	$25.1	$84.0	$109.1
8	1842	4.97%	$113.3	$1.1	$114.5	$27.1	$97.9	$125.0
8	1983	4.99%	$125.4	$1.2	$126.6	$26.2	$105.8	$132.0
8	2314	5.36%	$119.3	$1.4	$120.6	$30.4	$142.2	$172.6
8	1982	5.50%	$125.3	$1.2	$126.6	$27.0	$113.9	$140.9
8	1930	5.64%	$120.8	$1.2	$122.0	$26.7	$113.7	$140.5
8	2428	5.66%	$127.3	$1.4	$128.7	$30.6	$157.3	$187.9
8	1355	5.94%	$130.4	$1.2	$131.6	$23.5	$92.0	$115.5
8	1828	5.96%	$112.1	$1.1	$113.2	$28.7	$113.4	$142.2
8	1826	6.38%	$111.9	$1.1	$113.0	$29.5	$121.2	$150.7
8	1964	6.61%	$123.8	$1.2	$125.0	$29.4	$134.8	$164.2
8	2090	6.67%	$134.5	$1.3	$135.8	$31.0	$144.8	$175.8
8	2196	6.76%	$143.8	$1.5	$145.3	$32.5	$168.9	$201.4

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
8	1850	6.80%	$114.0	$1.1	$115.1	$30.7	$130.5	$161.2
8	1979	7.11%	$125.0	$1.2	$126.2	$30.5	$145.7	$176.3
8	2294	7.15%	$152.5	$1.6	$154.1	$32.6	$186.1	$218.8
8	2373	7.27%	$123.4	$1.4	$124.8	$33.8	$195.8	$229.6
8	1876	7.44%	$116.2	$1.1	$117.3	$29.8	$144.5	$174.4
8	1977	7.45%	$124.8	$1.2	$126.0	$31.1	$152.3	$183.4
8	2278	7.46%	$151.1	$1.6	$152.6	$33.1	$192.6	$225.7
8	1990	7.68%	$126.0	$1.2	$127.2	$31.7	$158.0	$189.8
8	2068	7.86%	$132.7	$1.3	$134.0	$33.4	$168.1	$201.5
8	2418	8.07%	$126.6	$1.4	$128.0	$35.9	$220.9	$256.8
8	2088	8.15%	$134.4	$1.3	$135.7	$34.2	$175.6	$209.8
8	1931	8.18%	$120.9	$1.2	$122.1	$31.9	$163.2	$195.0
8	2035	8.23%	$129.8	$1.3	$131.1	$33.7	$172.8	$206.5
8	2321	8.28%	$119.7	$1.4	$121.1	$35.2	$217.2	$252.4
8	2059	8.34%	$132.0	$1.3	$133.2	$34.2	$177.2	$211.4
8	2347	8.41%	$121.5	$1.4	$122.9	$36.1	$223.2	$259.3
8	2035	8.42%	$129.8	$1.3	$131.1	$34.0	$176.8	$210.8
8	2095	8.69%	$135.1	$1.3	$136.4	$35.3	$187.8	$223.1
8	1282	8.82%	$119.5	$1.1	$120.6	$27.0	$116.6	$143.6
8	2426	9.18%	$127.2	$1.4	$128.6	$38.6	$251.2	$289.7
8	1906	9.27%	$118.8	$1.2	$119.9	$33.7	$181.8	$215.6
8	2167	9.40%	$141.3	$1.5	$142.7	$35.7	$229.8	$265.6
8	2129	9.60%	$138.0	$1.4	$139.4	$35.6	$230.5	$266.1
8	2068	9.88%	$132.6	$1.3	$133.9	$37.4	$209.9	$247.3
8	1863	10.00%	$115.1	$1.1	$116.2	$34.2	$189.8	$224.0
8	2459	10.23%	$129.5	$1.4	$131.0	$39.3	$280.7	$320.0
8	2080	10.41%	$133.7	$1.3	$135.0	$36.2	$220.4	$256.5
8	2449	10.48%	$128.8	$1.4	$130.2	$39.6	$286.4	$326.0
8	2061	10.50%	$132.1	$1.3	$133.4	$38.2	$220.4	$258.6
8	1439	10.51%	$143.5	$1.3	$144.8	$32.8	$168.8	$201.6
8	2024	10.71%	$128.9	$1.2	$130.1	$38.0	$220.7	$258.7
8	2028	10.85%	$129.3	$1.2	$130.5	$38.3	$224.0	$262.4
8	2120	10.87%	$137.1	$1.4	$138.6	$37.5	$257.1	$294.5
8	2213	11.29%	$145.2	$1.5	$146.8	$39.7	$278.7	$318.4
8	2054	12.73%	$131.4	$1.3	$132.7	$39.8	$265.7	$305.5
8	1796	13.19%	$109.3	$1.1	$110.4	$38.6	$240.7	$279.3
8	1918	13.29%	$119.8	$1.2	$121.0	$40.8	$259.1	$299.9

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary Capital ($millions)	Conventional Cost Summary Operating ($millions)	Conventional Cost Summary Total ($millions)	Autoclave Cost Summary Capital ($millions)	Autoclave Cost Summary Operating ($millions)	Autoclave Cost Summary Total ($millions)
8	1833	13.60%	$112.5	$1.1	$113.6	$39.8	$253.3	$293.2
9	2749	2.97%	$151.0	$1.6	$152.7	$25.9	$99.7	$125.6
9	2575	3.40%	$137.9	$1.5	$139.5	$25.7	$105.5	$131.2
9	2501	3.80%	$132.6	$1.5	$134.0	$28.7	$113.6	$142.3
9	3181	4.13%	$152.6	$1.9	$154.5	$32.2	$168.3	$200.5
9	2663	4.39%	$144.6	$1.6	$146.1	$29.6	$138.2	$167.8
9	2627	4.41%	$141.9	$1.6	$143.4	$29.3	$136.8	$166.1
9	2880	4.68%	$132.6	$1.7	$134.3	$31.1	$171.1	$202.1
9	2751	5.16%	$151.2	$1.6	$152.9	$32.6	$162.9	$195.5
9	3022	5.25%	$142.0	$1.8	$143.7	$33.7	$196.7	$230.3
9	2602	5.52%	$140.0	$1.5	$141.5	$32.0	$164.4	$196.4
9	3016	5.54%	$141.5	$1.8	$143.3	$34.7	$206.4	$241.0
9	2475	5.64%	$130.6	$1.5	$132.1	$31.0	$159.6	$190.6
9	2908	5.72%	$134.5	$1.7	$136.2	$34.1	$205.4	$239.6
9	2944	6.10%	$136.8	$1.7	$138.5	$35.4	$221.1	$256.5
9	2913	6.18%	$134.8	$1.7	$136.5	$35.3	$221.5	$256.8
9	2637	6.46%	$142.5	$1.6	$144.1	$35.0	$194.1	$229.1
9	3122	6.48%	$148.7	$1.8	$150.5	$36.9	$248.6	$285.6
9	2476	6.52%	$130.7	$1.5	$132.2	$33.0	$183.9	$216.9
9	2571	6.63%	$137.6	$1.5	$139.2	$34.6	$193.9	$228.5
9	2503	6.73%	$132.7	$1.5	$134.1	$34.1	$191.6	$225.7
9	3172	6.82%	$152.1	$1.9	$153.9	$38.3	$265.7	$304.0
9	2730	6.86%	$149.6	$1.6	$151.2	$35.1	$212.9	$248.0
9	2708	7.27%	$148.0	$1.6	$149.6	$35.8	$223.4	$259.1
9	2509	7.31%	$133.1	$1.5	$134.6	$35.4	$208.2	$243.6
9	1599	7.35%	$116.9	$1.3	$118.2	$27.8	$143.9	$171.7
9	2914	7.37%	$134.8	$1.7	$136.5	$38.5	$262.8	$301.3
9	3026	7.44%	$142.2	$1.8	$144.0	$38.3	$275.8	$314.1
9	3082	7.48%	$146.0	$1.8	$147.8	$39.0	$282.3	$321.3
9	2741	7.48%	$150.4	$1.6	$152.0	$36.9	$232.6	$269.5
9	1561	7.65%	$112.5	$1.2	$113.7	$27.7	$135.3	$163.0
9	2723	7.91%	$149.1	$1.6	$150.7	$37.6	$243.8	$281.4
9	2591	7.92%	$139.1	$1.5	$140.7	$38.0	$232.2	$270.2
9	2556	8.02%	$136.5	$1.5	$138.0	$37.8	$232.1	$269.9
9	2624	8.03%	$141.6	$1.6	$143.2	$36.8	$238.6	$275.4
9	2508	8.37%	$133.0	$1.5	$134.5	$37.9	$237.2	$275.2
9	3140	8.39%	$149.9	$1.8	$151.7	$42.0	$321.7	$363.7

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
9	2583	8.53%	$138.5	$1.5	$140.1	$37.4	$248.9	$286.3
9	2551	8.57%	$136.2	$1.5	$137.7	$38.9	$247.1	$286.0
9	2524	8.60%	$134.2	$1.5	$135.7	$38.6	$245.2	$283.8
9	2492	8.61%	$131.9	$1.5	$133.3	$38.2	$242.3	$280.5
9	2519	8.80%	$133.8	$1.5	$135.3	$39.0	$250.2	$289.2
9	2993	9.04%	$140.0	$1.7	$141.7	$42.6	$329.7	$372.4
9	2980	9.35%	$139.1	$1.7	$140.9	$43.1	$339.3	$382.5
9	2612	9.36%	$140.7	$1.6	$142.3	$39.7	$275.7	$315.3
9	3154	9.56%	$150.8	$1.9	$152.7	$44.5	$366.7	$411.3
9	2526	9.70%	$134.4	$1.5	$135.9	$39.3	$276.0	$315.3
9	3046	9.95%	$143.5	$1.8	$145.3	$45.5	$368.7	$414.2
9	2861	10.01%	$131.4	$1.7	$133.0	$43.0	$345.1	$388.1
9	2786	10.14%	$153.9	$1.7	$155.6	$43.9	$315.4	$359.3
9	2744	10.53%	$150.7	$1.6	$152.3	$44.2	$322.3	$366.5
9	2529	10.53%	$134.6	$1.5	$136.1	$40.7	$297.2	$337.9
9	2687	10.90%	$146.3	$1.6	$147.9	$44.2	$326.8	$371.0
9	2620	11.10%	$141.3	$1.6	$142.9	$43.7	$324.4	$368.2
9	2795	11.62%	$154.6	$1.7	$156.3	$45.7	$362.1	$407.7
9	3121	12.75%	$148.6	$1.8	$150.4	$51.1	$478.5	$529.6
9	2665	13.05%	$144.7	$1.6	$146.3	$46.9	$387.6	$434.5
9	2776	13.08%	$153.2	$1.7	$154.8	$48.4	$404.6	$453.0
9	2722	13.60%	$149.0	$1.6	$150.6	$48.9	$412.2	$461.1
10	3254	3.52%	$157.7	$1.9	$159.6	$30.5	$148.4	$179.0
10	3519	4.52%	$151.1	$1.9	$153.0	$34.7	$202.4	$237.1
10	3667	4.90%	$160.4	$2.2	$162.6	$36.2	$242.9	$279.1
10	3270	5.37%	$158.8	$1.9	$160.7	$35.1	$217.2	$252.3
10	3389	5.69%	$143.1	$1.8	$144.9	$37.1	$238.2	$275.3
10	3785	5.76%	$147.7	$2.1	$149.8	$39.8	$287.2	$327.0
10	3377	5.78%	$142.3	$1.8	$144.2	$37.2	$240.8	$278.0
10	3767	5.80%	$146.7	$2.1	$148.7	$39.7	$287.8	$327.5
10	3817	6.29%	$149.6	$2.1	$151.7	$40.7	$315.2	$355.9
10	3324	6.49%	$139.1	$1.8	$140.9	$38.9	$265.0	$304.0
10	3803	6.55%	$148.7	$2.1	$150.8	$41.3	$326.6	$368.0
10	3530	6.55%	$151.8	$1.9	$153.7	$39.7	$284.3	$324.0
10	3788	6.70%	$147.9	$2.1	$150.0	$41.6	$332.7	$374.3
10	8021	6.92%	$205.2	$3.1	$208.2	$64.2	$681.3	$745.5
10	3477	7.16%	$148.5	$1.9	$150.4	$41.1	$305.3	$346.4

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
10	3416	7.17%	$144.7	$1.9	$146.6	$40.5	$300.1	$340.7
10	3265	7.67%	$158.5	$1.9	$160.4	$41.6	$306.4	$348.0
10	3628	7.96%	$157.9	$2.1	$160.1	$44.2	$376.4	$420.6
10	3779	7.98%	$147.4	$2.1	$149.5	$45.7	$393.4	$439.1
10	3412	8.19%	$144.5	$1.9	$146.3	$43.8	$341.3	$385.1
10	1830	8.58%	$145.6	$1.5	$147.1	$31.9	$191.6	$223.5
10	1824	9.57%	$144.8	$1.5	$146.3	$33.8	$212.4	$246.2
10	3398	9.59%	$143.6	$1.9	$145.5	$47.2	$396.6	$443.9
10	3848	10.11%	$151.4	$2.1	$153.5	$51.3	$500.2	$551.5
10	3220	10.34%	$155.4	$1.9	$157.3	$46.7	$401.1	$447.9
10	9976	10.35%	$217.9	$3.5	$221.5	$88.2	$1,244.0	$1,332.2
10	3739	10.70%	$145.1	$2.1	$147.1	$51.5	$514.0	$565.5
10	3263	11.15%	$158.3	$1.9	$160.2	$49.2	$437.9	$487.2
10	1945	11.31%	$124.3	$1.5	$125.8	$37.1	$282.4	$319.5
10	3846	12.61%	$151.2	$2.1	$153.3	$56.8	$622.5	$679.4
10	1870	13.55%	$150.9	$1.7	$152.5	$39.5	$324.9	$364.4
10	1239	14.51%	$166.0	$1.4	$167.5	$34.9	$216.1	$250.9
10	4566	14.82%	$174.3	$2.2	$176.5	$68.4	$812.9	$881.4
11	2103	4.06%	$141.5	$1.7	$143.2	$26.1	$116.8	$142.9
11	2198	4.70%	$152.4	$1.9	$154.3	$29.3	$147.8	$177.1
11	3956	6.09%	$157.6	$2.2	$159.8	$41.4	$316.7	$358.1
11	4607	6.10%	$161.6	$2.5	$164.1	$45.7	$390.3	$436.0
11	4445	6.29%	$168.2	$2.5	$170.7	$44.9	$388.2	$433.2
11	4241	6.61%	$156.7	$2.3	$159.0	$45.1	$367.5	$412.6
11	2041	8.25%	$134.7	$1.6	$136.3	$35.0	$219.5	$254.5
11	2244	8.53%	$129.8	$1.7	$131.5	$36.8	$263.5	$300.3
11	2241	8.85%	$129.4	$1.7	$131.1	$37.7	$272.5	$310.2
11	4517	10.02%	$156.8	$2.4	$159.3	$55.7	$615.0	$670.7
11	4373	10.10%	$164.1	$2.5	$166.6	$55.0	$600.4	$655.5
11	4453	11.35%	$168.7	$2.5	$171.2	$58.7	$686.3	$745.0
11	4261	12.36%	$157.9	$2.3	$160.1	$59.4	$676.0	$735.5
11	4489	13.11%	$170.7	$2.5	$173.2	$63.4	$798.2	$861.5
11	4104	13.74%	$166.5	$2.3	$168.8	$61.8	$722.9	$784.7
12	2518	5.64%	$157.7	$2.1	$159.8	$32.5	$207.6	$240.1
12	2535	6.75%	$159.4	$2.1	$161.6	$35.6	$248.6	$284.2
12	4645	7.59%	$163.6	$2.5	$166.2	$50.3	$486.3	$536.6
12	2274	9.21%	$132.7	$1.7	$134.4	$37.3	$287.7	$325.0
12	4670	10.92%	$165.0	$2.5	$167.5	$59.1	$692.4	$751.4
12	2370	14.44%	$142.2	$1.8	$144.1	$48.1	$463.8	$511.9

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary Capital ($millions)	Conventional Operating ($millions)	Conventional Total ($millions)	Autoclave Cost Summary Capital ($millions)	Autoclave Operating ($millions)	Autoclave Total ($millions)
12	2504	15.38%	$156.2	$2.1	$158.3	$50.9	$544.7	$595.6
12	2537	24.48%	$159.7	$2.1	$161.8	$63.9	$876.5	$940.5
13	2626	5.50%	$144.7	$2.0	$146.7	$33.4	$211.5	$244.9
13	2547	15.03%	$137.5	$1.9	$139.4	$51.0	$541.4	$592.4
13	2701	20.41%	$151.6	$2.2	$153.8	$60.6	$810.2	$870.8
14	3148	5.39%	$152.6	$2.4	$155.0	$35.8	$268.6	$304.5
14	3026	9.45%	$160.0	$2.4	$162.4	$45.7	$445.2	$490.9
14	2972	10.29%	$155.5	$2.3	$157.7	$46.9	$454.6	$501.6
14	2986	11.32%	$156.7	$2.3	$159.0	$48.6	$501.8	$550.4
14	3098	12.25%	$166.0	$2.5	$168.5	$51.4	$583.9	$635.4
14	2969	13.34%	$155.2	$2.3	$157.5	$51.8	$587.2	$639.0
14	3087	15.60%	$165.1	$2.5	$167.5	$57.6	$736.0	$793.6
15	3578	4.33%	$169.1	$2.7	$171.8	$35.9	$261.2	$297.1
15	3431	7.39%	$158.6	$2.6	$161.2	$43.0	$410.3	$453.3
15	3333	11.16%	$166.8	$2.5	$169.3	$51.1	$572.7	$623.9
15	3394	12.85%	$156.0	$2.5	$158.6	$55.4	$692.5	$747.9
15	3540	17.11%	$166.3	$2.7	$169.0	$65.1	$955.8	$1,020.9
15	3504	20.54%	$163.8	$2.6	$166.5	$71.2	$1,132.8	$1,204.0
16	3724	7.39%	$165.4	$2.7	$168.1	$46.1	$445.4	$491.4
16	3956	8.09%	$168.2	$2.9	$171.1	$49.3	$531.3	$580.7
16	3662	8.10%	$161.2	$2.7	$163.9	$47.3	$478.5	$525.8
16	3664	9.05%	$161.4	$2.7	$164.0	$49.3	$533.6	$583.0
16	3651	9.36%	$174.3	$2.8	$177.1	$50.2	$549.3	$599.5
16	3846	9.54%	$173.5	$2.9	$176.4	$52.2	$606.6	$658.8
16	3937	10.36%	$167.1	$2.9	$169.9	$54.7	$667.4	$722.1
16	3719	11.18%	$165.0	$2.7	$167.7	$55.0	$660.9	$715.9
16	3767	11.76%	$168.2	$2.8	$171.0	$56.6	$704.3	$761.0
16	3854	17.52%	$174.0	$2.9	$176.9	$69.7	$1,096.3	$1,166.1
16	3807	20.20%	$170.9	$2.9	$173.8	$74.4	$1,244.7	$1,319.1
16	3936	21.90%	$167.0	$2.9	$169.9	$79.3	$1,395.9	$1,475.2
17	4374	3.77%	$182.6	$3.2	$185.8	$37.7	$296.5	$334.3
17	4214	3.96%	$173.0	$3.0	$175.9	$37.8	$291.1	$328.9
17	4073	4.49%	$175.6	$3.0	$178.6	$38.8	$316.4	$355.2
17	4306	5.35%	$178.6	$3.1	$181.7	$43.0	$397.0	$440.0
17	4104	5.56%	$177.6	$3.0	$180.6	$42.6	$383.3	$425.9
17	4150	6.79%	$180.5	$3.0	$183.5	$47.2	$470.1	$517.3
17	4051	7.58%	$174.2	$3.0	$177.2	$48.9	$510.8	$559.6
17	4498	8.14%	$179.3	$3.2	$182.5	$53.4	$622.9	$676.3
17	4221	11.12%	$173.4	$3.0	$176.4	$59.2	$767.6	$826.8

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
17	4080	11.93%	$176.0	$3.0	$179.0	$59.9	$795.7	$855.6
17	4249	12.75%	$175.0	$3.0	$178.1	$63.5	$884.8	$948.3
17	4310	13.31%	$178.8	$3.1	$181.9	$65.4	$960.7	$1,026.0
17	4431	18.91%	$175.5	$3.1	$178.6	$79.2	$1,394.9	$1,474.2
18	5072	2.42%	$192.4	$3.6	$196.0	$35.3	$235.0	$270.3
18	4876	4.57%	$191.0	$3.5	$194.5	$43.6	$403.3	$446.9
18	4898	4.97%	$183.2	$3.4	$186.6	$45.6	$439.0	$484.6
18	4572	6.96%	$183.6	$3.3	$186.9	$50.3	$544.0	$594.4
18	4985	13.46%	$187.8	$3.5	$191.3	$72.3	$1,148.9	$1,221.2
18	4807	14.73%	$187.1	$3.4	$190.5	$73.7	$1,211.6	$1,285.3
18	4681	15.38%	$180.1	$3.3	$183.4	$73.9	$1,198.3	$1,272.2
18	4555	20.14%	$182.7	$3.2	$185.9	$83.2	$1,522.2	$1,605.5
19	5081	5.22%	$192.9	$3.6	$196.5	$47.1	$467.6	$514.7
19	5641	6.82%	$197.7	$3.8	$201.5	$57.2	$686.0	$743.2
19	5146	8.45%	$187.9	$3.5	$191.5	$59.3	$756.1	$815.4
19	5182	8.86%	$189.8	$3.6	$193.3	$60.8	$796.7	$857.5
19	5279	9.68%	$194.8	$3.6	$198.5	$64.1	$885.5	$949.7
19	5658	10.24%	$198.5	$3.8	$202.3	$68.8	$1,014.3	$1,083.1
19	5566	16.99%	$201.5	$3.8	$205.3	$87.7	$1,642.0	$1,729.7
19	5262	17.45%	$193.9	$3.6	$197.5	$85.4	$1,562.5	$1,647.9
19	5678	18.32%	$199.5	$3.9	$203.3	$89.1	$1,806.4	$1,895.4
19	5379	20.82%	$192.0	$3.7	$195.7	$91.6	$1,938.1	$2,029.7
20	5710	3.06%	$201.0	$3.9	$204.9	$41.6	$333.3	$374.9
20	6034	4.60%	$209.6	$4.0	$213.6	$51.0	$512.5	$563.6
20	6224	6.71%	$211.8	$4.2	$215.9	$60.8	$758.4	$819.3
20	5715	6.74%	$201.2	$3.9	$205.1	$57.5	$687.1	$744.6
20	6353	7.80%	$211.4	$4.2	$215.6	$66.0	$895.9	$962.0
20	5812	8.56%	$206.0	$4.0	$210.0	$64.6	$881.9	$946.6
20	5713	9.27%	$201.2	$3.9	$205.0	$66.3	$936.3	$1,002.6
20	6023	9.97%	$209.0	$4.0	$213.1	$71.3	$1,060.7	$1,132.0
20	5967	10.50%	$206.3	$4.0	$210.3	$72.5	$1,096.6	$1,169.1
20	6162	14.49%	$208.8	$4.1	$212.9	$87.0	$1,586.1	$1,673.1
20	5749	15.69%	$202.9	$3.9	$206.8	$86.2	$1,565.9	$1,652.0
21	6470	4.87%	$216.8	$4.3	$221.1	$54.8	$590.2	$645.0
21	6982	5.10%	$223.8	$4.6	$228.4	$58.6	$662.5	$721.0
21	6899	5.29%	$220.1	$4.5	$224.7	$58.9	$678.6	$737.5
21	6911	5.67%	$220.6	$4.6	$225.2	$60.9	$726.8	$787.7
21	6654	5.70%	$220.2	$4.4	$224.6	$59.1	$692.8	$751.9
21	6836	5.91%	$222.7	$4.6	$227.3	$61.3	$747.9	$809.3

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
21	6679	6.05%	$221.4	$4.4	$225.8	$61.0	$736.1	$797.1
21	6535	7.03%	$213.7	$4.3	$218.0	$64.3	$833.1	$897.4
21	7017	8.26%	$225.4	$4.6	$230.0	$72.8	$1,062.3	$1,135.1
21	7065	8.47%	$227.6	$4.7	$232.2	$74.2	$1,096.7	$1,170.9
21	6875	8.57%	$224.5	$4.6	$229.1	$73.1	$1,079.1	$1,152.2
21	6805	12.88%	$221.3	$4.5	$225.9	$88.3	$1,583.3	$1,671.7
21	6421	14.36%	$214.5	$4.3	$218.8	$89.3	$1,637.3	$1,726.6
21	6834	14.70%	$222.6	$4.6	$227.2	$95.0	$1,812.8	$1,907.8
21	6864	21.47%	$224.0	$4.6	$228.6	$112.6	$2,638.5	$2,751.1
22	7511	4.36%	$232.3	$4.9	$237.2	$58.5	$636.3	$694.8
22	7499	5.30%	$231.8	$4.9	$236.7	$62.8	$749.4	$812.3
22	7164	7.45%	$226.8	$4.7	$231.4	$70.6	$981.1	$1,051.7
22	7447	8.48%	$234.3	$4.9	$239.2	$77.4	$1,173.1	$1,250.5
22	7118	9.28%	$224.7	$4.6	$229.4	$78.0	$1,208.3	$1,286.3
22	7376	10.79%	$231.2	$4.8	$236.0	$85.8	$1,439.4	$1,525.3
22	7761	13.15%	$238.7	$5.0	$243.7	$99.3	$1,868.3	$1,967.5
22	7175	14.43%	$227.2	$4.7	$231.9	$94.2	$1,868.0	$1,962.2
22	7267	14.65%	$231.4	$4.8	$236.1	$95.9	$1,920.7	$2,016.6
22	7712	18.54%	$236.6	$5.0	$241.5	$114.7	$2,602.5	$2,717.2
22	7576	19.05%	$235.2	$4.9	$240.1	$114.5	$2,627.0	$2,741.5
22	7165	21.01%	$226.8	$4.7	$231.5	$115.2	$2,694.1	$2,809.3
23	7960	4.63%	$242.9	$5.1	$248.1	$62.7	$712.9	$775.6
23	8169	5.46%	$247.8	$5.3	$253.1	$68.1	$850.6	$918.7
23	7955	6.37%	$242.8	$5.1	$247.9	$71.4	$948.8	$1,020.2
23	8470	8.04%	$252.7	$5.4	$258.2	$83.9	$1,282.9	$1,366.8
23	8152	14.38%	$247.1	$5.3	$252.3	$104.8	$2,172.9	$2,277.7
23	8260	16.23%	$247.6	$5.3	$252.9	$113.1	$2,471.1	$2,584.3
23	8300	18.31%	$249.3	$5.3	$254.7	$121.5	$2,801.2	$2,922.7
23	8113	19.39%	$245.4	$5.2	$250.5	$123.0	$2,863.9	$2,986.9
23	8159	22.47%	$247.4	$5.3	$252.7	$135.0	$3,371.8	$3,506.8
24	8812	5.91%	$259.9	$5.6	$265.5	$75.3	$990.2	$1,065.5
24	9008	7.68%	$264.7	$5.7	$270.5	$86.7	$1,320.3	$1,407.0
24	8830	8.53%	$260.7	$5.7	$266.3	$89.6	$1,433.0	$1,522.6
24	8790	11.91%	$259.0	$5.6	$264.5	$101.5	$1,943.4	$2,045.0
24	8653	12.27%	$256.8	$5.5	$262.3	$101.7	$1,970.4	$2,072.2
24	8833	13.51%	$260.8	$5.7	$266.5	$109.2	$2,238.1	$2,347.3
24	8522	14.99%	$255.0	$5.5	$260.4	$111.6	$2,366.8	$2,478.4
24	8963	16.67%	$262.8	$5.7	$268.5	$124.1	$2,785.9	$2,910.0
24	9108	18.23%	$269.0	$5.8	$274.8	$132.7	$3,096.2	$3,228.9

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
25	9488	3.04%	$275.2	$6.0	$281.1	$62.2	$591.5	$653.7
25	9764	4.11%	$283.8	$6.2	$290.0	$71.7	$810.3	$881.9
25	9803	5.57%	$282.4	$6.2	$288.5	$81.2	$1,063.0	$1,144.2
25	9816	8.63%	$282.9	$6.2	$289.1	$99.5	$1,629.0	$1,728.5
25	9185	8.97%	$268.8	$5.8	$274.6	$95.0	$1,565.7	$1,660.7
25	9517	9.87%	$276.4	$6.0	$282.4	$103.3	$1,782.3	$1,885.5
25	9182	10.52%	$268.7	$5.8	$274.5	$102.5	$1,814.8	$1,917.4
25	9427	13.69%	$275.8	$6.0	$281.8	$117.1	$2,420.1	$2,537.1
25	9617	14.22%	$280.6	$6.1	$286.7	$122.0	$2,590.2	$2,712.1
25	9466	18.39%	$274.2	$6.0	$280.2	$138.9	$3,246.9	$3,385.8
25	9800	21.97%	$282.2	$6.2	$288.4	$160.8	$4,044.8	$4,205.6
26	10481	6.77%	$299.8	$6.6	$306.4	$94.8	$1,384.9	$1,479.6
26	10099	7.39%	$292.0	$6.4	$298.3	$95.1	$1,441.4	$1,536.4
27	10690	4.47%	$306.0	$6.7	$312.7	$81.2	$968.5	$1,049.7
27	10613	8.92%	$302.8	$6.7	$309.4	$109.5	$1,834.6	$1,944.1
27	10877	13.26%	$309.0	$6.8	$315.8	$134.1	$2,758.8	$2,893.0
27	10696	14.33%	$306.3	$6.7	$313.0	$137.2	$2,928.9	$3,066.2
27	10623	16.44%	$303.2	$6.7	$309.9	$147.6	$3,321.3	$3,468.9
28	11424	3.86%	$325.1	$7.2	$332.3	$82.5	$911.0	$993.5
28	11537	7.02%	$327.7	$7.2	$334.9	$106.7	$1,593.3	$1,700.0
28	11752	7.38%	$332.7	$7.4	$340.0	$111.7	$1,716.9	$1,828.5
28	11439	8.10%	$325.7	$7.2	$332.9	$113.3	$1,814.8	$1,928.2
28	11599	8.11%	$330.3	$7.2	$337.6	$115.3	$1,842.5	$1,957.7
28	11683	13.18%	$331.8	$7.3	$339.1	$145.4	$2,991.8	$3,137.2
29	12326	4.38%	$349.3	$7.7	$357.0	$94.7	$1,119.5	$1,214.3
29	12494	8.15%	$354.6	$7.8	$362.4	$126.1	$2,022.8	$2,148.9
29	12047	10.47%	$341.2	$7.5	$348.7	$132.7	$2,455.0	$2,587.7
29	12044	13.43%	$341.1	$7.5	$348.6	$152.5	$3,142.4	$3,294.9
29	12379	16.02%	$351.5	$7.8	$359.3	$170.0	$3,855.5	$4,025.6
30	13092	4.27%	$373.5	$8.2	$381.7	$101.2	$1,170.2	$1,271.4
30	13300	6.54%	$379.3	$8.3	$387.6	$122.9	$1,749.5	$1,872.4
30	12755	8.97%	$362.3	$8.0	$370.3	$132.5	$2,267.7	$2,400.2
30	13153	15.18%	$374.6	$8.2	$382.8	$170.4	$3,906.7	$4,077.1
30	12780	16.06%	$363.4	$8.0	$371.3	$172.0	$3,990.8	$4,162.8
31	13377	3.97%	$381.2	$8.3	$389.5	$100.8	$1,116.5	$1,217.3
31	14027	4.77%	$402.4	$8.7	$411.2	$116.1	$1,397.5	$1,513.6
31	13779	7.98%	$394.3	$8.6	$402.9	$138.6	$2,212.7	$2,351.3
31	14060	8.01%	$403.8	$8.8	$412.6	$142.6	$2,266.4	$2,409.0
31	13489	8.62%	$384.6	$8.4	$393.0	$139.9	$2,321.3	$2,461.1

Table D-1 (cont'd)

Mine Life	Mill Tonnage	%S in Tailings	Conventional Cost Summary			Autoclave Cost Summary		
			Capital ($millions)	Operating ($millions)	Total ($millions)	Capital ($millions)	Operating ($millions)	Total ($millions)
31	13405	14.29%	$382.4	$8.4	$390.7	$168.4	$3,768.0	$3,936.4
32	14488	6.14%	$417.7	$9.0	$426.8	$134.7	$1,812.6	$1,947.3
32	14328	6.57%	$413.0	$8.9	$421.9	$136.3	$1,903.2	$2,039.5
32	14651	8.81%	$423.7	$9.1	$432.9	$149.4	$2,603.3	$2,752.7
32	14574	16.28%	$421.4	$9.1	$430.5	$180.6	$4,691.6	$4,872.2
33	15030	4.60%	$437.2	$9.4	$446.5	$126.1	$1,453.7	$1,579.8
33	15039	7.46%	$437.6	$9.4	$447.0	$143.6	$2,271.4	$2,415.0
33	15070	10.30%	$438.1	$9.4	$447.4	$157.5	$3,093.5	$3,251.1
33	14889	11.78%	$432.0	$9.3	$441.3	$163.7	$3,492.1	$3,655.7
33	15421	12.39%	$450.8	$9.6	$460.4	$168.2	$3,819.9	$3,988.1
34	15474	4.81%	$453.1	$9.7	$462.8	$132.4	$1,567.3	$1,699.7
34	15667	4.90%	$460.0	$9.8	$469.8	$133.5	$1,613.8	$1,747.3
34	16136	6.34%	$477.9	$10.1	$488.0	$143.5	$2,100.2	$2,243.6
34	15851	7.90%	$467.4	$9.9	$477.3	$148.1	$2,552.6	$2,700.8
35	16325	4.81%	$485.1	$10.2	$495.3	$134.4	$1,661.4	$1,795.8
35	16272	10.49%	$483.3	$10.2	$493.5	$162.2	$3,429.3	$3,591.5
36	17376	8.41%	$528.0	$10.9	$538.9	$155.0	$2,995.3	$3,150.3
36	16949	15.51%	$510.4	$10.6	$521.0	$187.1	$5,261.8	$5,448.9
37	17682	12.01%	$532.5	$11.0	$543.5	$174.1	$4,292.0	$4,466.1
37	17766	15.47%	$532.5	$11.0	$543.6	$190.0	$5,519.3	$5,709.3
38	18537	5.00%	$532.7	$11.3	$544.0	$140.4	$1,975.3	$2,115.6
38	18950	7.57%	$532.8	$11.5	$544.3	$154.2	$2,968.2	$3,122.3