

2

Environmental Life Cycle Considerations for Design Related Decision Making in Minerals Processing

VOLUME 2

Appendices

A thesis submitted to the
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DOCTOR OF PHILOSOPHY
in the Department of Chemical Engineering

by

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Appendix 2: Information Structuring Appendix

Appendix 2-1: Information available on the environmental performance of the South African Minerals Industry

The report "The Situation of Waste Management and Pollution Control in South Africa" (CSIR, 1991) commissioned by the Department of Environmental Affairs and Forestry (DEAF) and completed in January 1991, was based on a number of different questionnaires sent out to different parties within the waste management community. The distribution of questionnaires sent out and those returned is detailed in Table A2.1-1.

Questionnaires sent to	Sent out	Returned	Percentage returned
Producers	986	185	18.76
Controllers and Municipalities	251	105	41.83
Research Organisations	128	109	85.16
Action Groups	49	32	65.31
Total	1414	431	30.48

Table A2.1-1 Distribution of CSIR Questionnaires on Waste Management

The number of questionnaires returned represents the number of institutions returning them, ie., the duplication if a single company returned more than one questionnaire is not reflected. What this shows is that, although 43% of the returned questionnaires originated from the producers of waste, only 19% of the producers approached responded. Producers represent 70% of the original recipients of the questionnaire.

The value of the information concerning the waste generated by the minerals industry alone is discussed here.

Wastes are classified according to whether they impact on water, air or land. This aggregation of the values gives an idea of the effects on the various media but makes it difficult to determine the waste stream generated by one of the sub-sectors of the industry dealt with (say the minerals industry).

Air pollution figures are segmented, there is an estimation of the distribution of the airborne pollution stream in the Eastern Transvaal Highveld (a specific geographical area in the north east of the country) for 1987 as well as an indication of the change in air quality within various urban centres from 1985 to 1989. There is no comprehensive breakdown of the gaseous waste stream by industry

making it difficult to get a total for the amount of air-borne pollution attributable to the minerals industry.

Land affected by solid waste is discussed. The total amount of ore mined per year is quoted (no indication of which year is given) as well as an estimated quantification of the waste stream and a breakdown of this stream by waste type (tailings, slags, etc) are given for 1987 and 1990. The total area affected by accumulated wastes is given for 1990. No indication of land degradation attributable to mining activities is given.

The waste stream entering water does not include the amount of water within the stream (only the components are quantified, not the body of water carrying them). This makes it difficult to assess the impact of the stream as dilution effects can not be accounted for nor is there any indication of the amount of water being affected. An estimate of the amount of water being affected is important as water is a reactive medium and the mixing of wastes with different amounts of water can lead to different types of impacts.

At the same time that the CSIR was conducting their study the Foundation for Research and Development (FRD) was also involved in a study for the DEAF concerned with hazardous waste in South Africa. This study was later managed by the CSIR. It is detailed in the six volumes of "Hazardous Waste in South Africa" (FRD, 1993). Again only the information pertaining to the minerals industry will be discussed.

The information in this report is far more detailed as far as distribution across regions within the country is concerned. Also, because of the desire to categorise the wastes according to hazard potential, the constituents of the waste stream are given in more detail.

There are five hazard groups identified in the report. These range from extremely hazardous (Hazard Group 1) to non-hazardous (Hazard Group 5). This classification of the hazard rating of wastes has given rise to much negative comment. Hazard classification of wastes within South Africa is under review at present (NWMS, 1998).

Again wastes are aggregated according to the media which they affect making it difficult to make an accurate assessment of a single sector. Wastes entering bodies of water do not include the volumes of water within the waste stream or the amount of water being affected by the stream.

Both of these reports, although having a lot of information about specific sections of the waste stream, are deficient in four areas:

- Detail concerning components within the waste stream
- Quantity of water in the waterborne waste stream
- The numbers they include are static and, although given as an estimate in of the situation for 1991, include data covering a number of years thus rendering the final data set inconsistent
- The Minerals Industry did not accept the validity of the information published as they felt that they were not consulted sufficiently about the values used in the calculations

It was thus deemed necessary to calculate a more accurate waste stream for the South African Minerals Industry. It was decided that the best approach to evaluating this waste stream would be by using the tools of process synthesis; thus the waste stream would be calculated using the performance of technologies in place. This approach leads to an analysis of technologies, it no longer views the process as a "black box". This is an important difference as it is the process which produces the waste and thus the process must be analysed if any attempt to decrease the impact of a process on its environment is to be effective.

Appendix 2-2: Data sources and verification of flowsheets and inventories

Data on the performance of technologies in place in the South African Minerals Industry was available in many places and forms. These are detailed below. An initial data gathering exercise which covered all these forms of data was undertaken in order to establish preliminary flowsheets for each sub-sector of the industry. Once these had been evaluated and inventories calculated they were presented to a wide variety of audiences for comment (many of these people being those who had offered comment in the first instance). The comments were then integrated into the flowsheets and the flowsheets were refined until a final form was reached. Due to the iterative nature of the process it is difficult to divorce those sources of information which rendered the initial information from those which were used to establish the veracity of the results generated by the models. For this reason they are presented together. An attempt is made to indicate those sources of information which were used in the first iteration, this is presented on Table A2.2-1.

2.2.1 Journals

Journals give information on two levels, environmental and technical.

The technical journals contain some of the most relevant information available as there are a number of studies of sections of the South African minerals beneficiation industry which detail flowsheets as well as how the technology within these operates and performs. This information was used extensively in drawing up the unit operations as well as the generalised flow diagrams.

Many of the technical journals contained balanced and enlightening articles on the environment and the effect that the industry has on it.

The environmental journals have information of differing usefulness. Some of them are technical and give exciting ideas for alternate technologies. Others are biased against industry as a whole, making no effort to understand the industries involved in various studies. This can make the information they contain slanted.

Environmental journals are a good source of information concerning the growth and development of the environmental initiative world-wide.

2.2.2 Books

These too can be divided into two emphases, environmental and technical.

The technical volumes were used in drawing up the generalised flowsheets as well as giving an idea of how the unit operations work. In some cases, when the information available was limited, they were used to give an estimate of the various operations within the unit and the technicalities involved.

Environmental books give an insight into the philosophies of environmental thinking as well as clarifying concepts. There are many valuable works which detail upgrading technologies as well as alternative technologies for impact minimisation and clean technology.

2.2.3 Newspapers

The daily and weekly papers for the country give an indication of changes within the minerals beneficiation industry. The information from this source is as varied as the newspapers read. Newspapers read were financial, technical and general in emphasis. They have much interesting and current information concerning:

- Economics affecting the industry as well as the economic situation of the country
- Increased capacity within the industry associated with new projects as well as plant upgrades
- Technology shifts within the industry
- The industry's attitude to the environment
- The attitude of both the private and the public sectors to the environmental movement and to the minerals beneficiation sector

2.2.4 Annual Reports

Although these are limited to the companies listed on the JSE (Johannesburg Stock Exchange) they give a fair indication of the performance of the industry as a whole. They contain much information quantifying material processed as well as products from the processes. Some of them detail processes to unit operation level. They obviously give a detailed idea of the economics of different operations.

It was interesting to note the increase over the last five years in reports containing sections devoted to environmental considerations as well as reports containing an explicit environmental statement for the company.

2.2.5 Interviews

Interviews were conducted on two levels, firstly during the initial phase of the project when general information on environmental performance was being sourced. Later, once the generalised flowsheets for the various sub-sectors had been established, and mass and energy balances for these flowsheets had been calculated, interviews were conducted with people in order to establish both the technical veracity of the flowsheets as well as the accuracy of the assumptions used in calculating the mass and energy balances. Table A2.2-1 below details the cross-section of people contacted as well as the types of information they provided. These interviews included a significant number of plant visits. These interviews were conducted before the major re-structuring that has taken place in the industry since the beginning of 1997.

The type of information obtained from the various sources was:

- Academics

These people supplied information on technology in place as well as “cutting edge” technology; they are a good source of information about the current situation in their specific industry.

- Consultants

Consultants in two fields were interviewed. The information they supplied was as detailed as confidentiality around projects allowed them to be. A good source of information concerning unit operations and process particulars.

- Corporate

The corporate employees are obviously the best source of technical information and forthcoming with whatever information they feel is not sensitive.

- Sector Associates

The two groups approached are different in their attitudes but can give interesting perspectives on various issues.

- Government Departments

As the laws in the country change so do the types of information available from the various departments. Accessibility of information differs between the departments but all have a detailed picture of the *status quo* for the country as a whole as falls within their jurisdiction.

- NGOs

The information from these sources is detailed but limited to their areas of interest.

- Research Organisations

A detailed source of information concerning the operation of units within the processes as well as certain aspects of environmental performance.

The interview process, although time consuming, is a worthwhile source of information.

		Tech Staff	Environ Staff	Other Staff	Written Material
Academic	University of Cape Town	5	1	2	yes
	University of the Witwatersrand	2			
	University of Natal - Durban	1			yes
	Other	1			
Consultant	Environmental	5	9	5	yes
	Technical	19		1	yes
Corporate	Anglo American	10	2		yes
	Anglovaal	2			
	Gold Fields	3		1	yes
	Gencor	8	3		yes
	Iskor	2			
	Palaborwa Mining	2			yes
	Richards Bay Minerals	1			
	Sasol	2			
	Eskom	9	2		yes
	Other	19		1	yes
Sector Associates	Trade Unions		2	2	
	Chamber of Mines		2	2	yes
Government Departments	Minerals Affairs	9			yes
	Water Affairs	2	2		yes
	National Health		3		yes
	Environment Affairs		2		yes
	Other		1	1	
NGOs			8		yes
Research Organisations	CSIR	11	1		yes
	Mintek	7	2		yes
	Water Research Commission		2		yes
	Other		1	2	
TOTALS		126	43	19	

Table A2.2-1 Details of people contacted

2.2.6 Information Pamphlets

These pamphlets can be divided into

- Company specific pamphlets which contain general information on the processes within the company and were used in establishing the mass and energy balances over these processes.
- Information on specific pieces of equipment and the operation of this equipment, also used in mass and energy balance calculations.

2.2.7 Environmental Impacts Assessments

There have been a number of EIAs conducted for projects in the minerals beneficiation sector. They are easily accessible in the public domain. As the law governing them becomes more sophisticated they will become easier to use and interpret. They are a good source of technical information, although, as the process is never the focus of the report (environmental impact is), this information is dispersed throughout the report and requires a fair amount of interpretation before it is useful.

The information they contain concerning environmental impact is extremely comprehensive but can be subjective.

2.2.8 Summary of Sources of Information

Table A2.2-2 below details what type of information each source rendered and the value of the information.

Source	Technical Information	Environmental Information	Source of Initial Information
CSIR and FRD reports	None	Detailed within limits; not consistent; static	
Journals	Excellent	Excellent	yes
Books	Excellent	Excellent	yes
Newspapers	Limited to new projects; current information	Interesting general information	
Annual Reports	General	General; usually corporate environmental policy	yes
Interviews and Plant Visits	Excellent	Excellent	yes
Information Pamphlets	Varied; detailed within limits	Varied; usually simplified	
EIAs	Detailed but fairly difficult to interpret	Detailed concerning specific impacts	yes

Table A2.2-2 Details of sources of information

Appendix 2-3: Sub-sectoral Study on Gold

2.3.1 Introduction

South Africa's known gold reserves represent 39% of the world's total. In the 112 years over which records of gold production have been kept, South Africa has produced over 44 000t of gold. Although the amount of gold mined in South Africa has been steadily decreasing since the mid 1970s, in 1992 South Africa still produced 28% of the world's gold. The Witwatersrand Basin delivers 98% of this. Most of the gold in South Africa is associated with iron pyrite (FeS_2) ores, less than 2 percent of the gold is associated with arsenopyrite.

Gold plays a significant part in the South African economy. In 1996 there were 54 active gold mining companies and 13 mining holdings companies listed on the Johannesburg Stock Exchange. Between them they controlled the 55 operating gold mines in the country. Together these mines represent over 2500 years of mining. The tailings from these processes are widely spread over the industrial heart of the country (the Eastern Transvaal Highveld). These wastes present a significant environmental liability to the industry as well as to the country as a whole as many of the older dumps were engineered with scant respect for environmental considerations. It is not possible to quantify the size or the nature of this problem explicitly as there is no complete data-base of the number of these dumps nor their positioning.

During the 1996 production year South Africa produced 497 t of gold at an average of 42 t/month. The revenue from export sales was R 26 billion or just over R2 billion/month.

2.3.2 Generalised flow diagram

Figure A2.3-1 contains the generalised flow diagram for gold. There are three basic process routes. Of these gravity concentration followed by mercury amalgamation was the first used. This process resulted in high mercury emissions from the smelting process and relatively low gold recoveries.

The two main process routes currently used are based on dissolution in a cyanide solution. In older processes dissolution was followed by filtration and then the cementation of gold from a clarified

solution. In more recently commissioned plants the carbon-in-pulp (CIP) adsorption of gold has replaced the clarification and cementation processes. The CIP process is a semi-batch process in which the adsorbed gold is removed from the carbon, the carbon then being regenerated before its return to the main process.

Some plants use gravity concentration on part of the stream from comminution as gold is often concentrated in only a small part of the overall stream. In plants using gravity concentration recoveries in the gravity concentration plant can account for up to 20% of the gold for the process.

The refractory ores of the Barberton reef represent 1.2% of the total material processed. These are the ores associated with arsenopyrite. The ores are roasted to oxidise the gold and render it available for cyanide leaching. SO_x gas emissions are associated with this roasting. One South African plant (Fairview) has replaced the roasting process with bio-leaching.

Sulphuric acid was historically a co-product in the gold industry, however, as transport costs are high and the product value is low, there is no significant market for the acid.

Mine dump sands represent 43% of the ore processed. This includes plants processing sands only and others processing sands along with mined ore. As the gold price drops the percentage of sands in the feed to the process is projected to grow as the cost of getting sands to the plant is far lower than mining ore. Typical grades of gold in the sand are 1-3 ppm (usually recorded as g/ton). Ore grades range from 2-8 g/ton.

The structure of the industry has changed markedly since early 1996. In many cases there has been an unbundling of the mining holding companies (Anglo American, Gencor), in one case there has been an amalgamation of two of the larger role players in the industry (Goldfields and GenGold to form Goldco). These changes will change the management structures in place on the plants, their effects will not be felt when analysing the "hard technologies" in place and will thus not be discussed in this section. Later discussion on management structures and management systems will address this issue.

The industry has ongoing technology improvements in order to improve the recovery of gold. However, there have been no significant technology shifts away from the three process flow routes described in Figure A2-1 and thus the values calculated using models based in this diagram can be seen as representative of the industry as a whole.

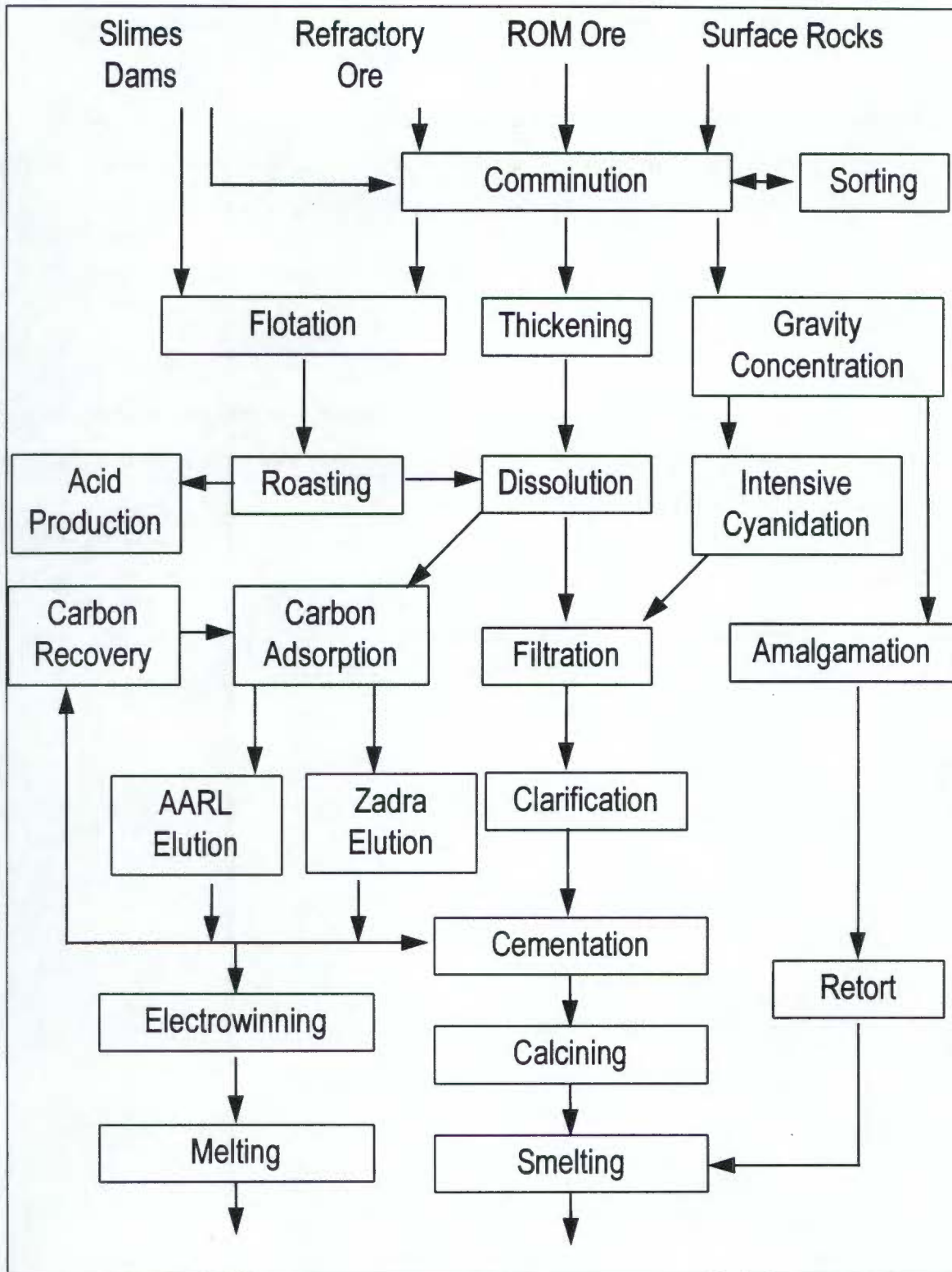


Figure A2.3-1: Generalised Flow Diagram - Gold

2.3.3 Waste Inventory

The complete waste inventory for the generalised gold process as described in Figure A2.3-1 is discussed in this section. The recovery of gold for the industry as modelled is 86% of total gold in the feed stream to the industry. The *Excel* spreadsheet also included in this appendix.

2.3.3.1 Predicted Values

The total waste stream predicted for the South African gold industry in the production of the 496 tonnes of gold produced in 1996 was 420 Mt. The waste stream is detailed by component in Table A2.3-4.

Component	Mass (kt/annum)
Water	270 000
Ore	150 000
Steam	310
Dissolved Salts	160
Sulphur	120
Iron	110
CaO	78
Carbon	21
HCl	17
NaCN	13
NaOH	13
Other Metals	0.39
Au	0.08
Ag	0.01
Hg	5.20E-06
TOTAL	420 000

Table A2.3-4: Breakdown of waste stream by Component for the production of 496 t of gold in 1996

It should be noted that, despite the extreme differences in the orders of magnitude of these numbers, the environmental effect of a waste is not always determined by the mass discharged. Environmental impact also relates to the toxicity of the component. It is vitally important to retain as much detail regarding stream components as possible.

The main components in this waste stream are 150 Mt of ore and 270 Mt of water. The next major contributors are the reagents used in the elution of gold from carbon during the CIP process. The values calculated for the dissolved salts are a barest minimum and would have to be verified by speciation of streams within the process. Some cyanide also leaves the process. (Note: Totals are rounded).

Table A2.3-5 gives a breakdown of the waste stream according to the unit operation which produces it.

Unit	Mass (kt/annum)
Flotation	110000
Calcining	88000
Filtration	69000
Thickener	51000
CIP	49000
Elution	33000
Gravity Concentration	16000
Crush & Sort	2700
Roasting	790
Amalgamation	0.50
Smelting	0.40
Electrowinning	0.04
TOTAL	420000

Table A2.3-5: Breakdown of waste stream by Unit
Operation for the production of 496 t of gold in 1996

The waste stream leaving the Calcining unit has a high mass flow, most of this is due to the high water usage of the units preceding it in the process. This is merely the units in which most of the final concentration of gold takes place, so it is at this point that the waste leaves the process. The last refining stages give rise to little or no waste. This is due to the size of the unit operations and the fact that by this point in the process there are no more materials which can be regarded as waste to be discharged from the system.

As has been stated, environmental impact can not necessarily be inferred from the mass flow rate of the stream. The ore in the waste stream has the potential to impact on the environment by producing an acid leachate. This acid leachate is produced as an oxidation reaction of the FeS_2 in the presence of water. This is seen as the greatest environmental impact associated with wastes from the industry. Water in the industry is run in closed circuit with drainage from the mine dumps being returned to the process as process water. This keeps much of the acid and associated dissolved salts within the process. The solution is diluted with a addition of clean water and by rainfall. The longer term impact associated with the leachate generated by the dumps post-closure will be discussed in a later section.

Other components in the waste stream, such as elution reagents and cyanide, are retained in the closed water loop. There are assertions that cyanide is broken down under ultra-violet and thus there should only be trace elements in the dumps. This has still to be verified quantitatively. There are no reported human deaths from exposure to cyanide in the dumps. The volumes of water used in the industry serve, to a great extent, as dilution agents for the contaminants in the stream.

The other issue concerning the dumps which should be addressed is the radiation naturally occurring in the ores. While there was still a significant demand for uranium it was produced as a co-product from the industry. It has been recognised that there is a potential environmental impact associated with radiation in the dumps but this also has still to be quantified.

2.3.3.2 Comparison with other information

As has already been stated, the variables and quantities used in the calculation of these values were verified by a number of sources. However, the only two sources which quantified the waste stream from the minerals industry are the DEA and CSIR reports (CSIR, 1991; FRD, 1993). The values quoted in these reports have been aggregated in Table A2.3-6.

These reports quantified the waste stream for the gold industry in South Africa as 192 Mt/ annum. This is 75% of the value predicted by the model. However, it can be expected that the tailings stream alone should approximate the quantity of ore mined during the year (figures for the amount of water in the stream having been omitted from the FRD/CSIR information). Comparison of the tailings stream 190 Mt/annum (Table A2.3-6), with the ore waste, a value of 150 Mt/annum (Table A2.3-4), creates a better perspective of the values. Gold production in 1991, the year for which the values in Table A2.3-6 were evaluated, was 631 tonnes, in 1996 it was 494 tonnes. This would account for the discrepancy between the reported and predicted amounts of ore in the waste stream.

The omission of water in the waste streams of the CSIR and FRD reports explains further the discrepancies between the waste stream figures in the referenced works and those predicted.

Category	Waste Stream (kt/annum)	Hazardous Waste Groups			Generation To		
		Haz Waste (Grp 1-3)	Potential Haz (Grp 4)	Non- Haz (Grp 5)	Air	Water	Solid/ Liquid
Tailings	190181						
Effluent	1013						
Uranium waste	529						
Catalyst for H ₂ SO ₄	0.06						
Smelting waste	2.1						
TOTAL	191726	1013	531	190181	0	1538	190188

Table A2.3-6: Waste Stream generated by the South African gold mining industry (FRD, 1993; CSIR, 1991)

2.3.3.3 Discussion

While the accuracy of the models depends on accurate assumption values it is more important that the same level of confidence could be placed in all of the values. Here an 80-20 rule has been applied. A more consistent output is achieved with 80% confidence in all of the assumptions than with 100% confidence in 80% of the numbers.

The results from the models are reasonable if the amount of ore processed, the recoveries attained and the final production rate are considered. The quantities calculated are valuable on a macro-scale; they can be used in macro-scale scenario planning and can indicate trends within the industry.

Confidence levels

Flow Sheet	95%
Assumptions	80%
Stream Splits	75%

A lower confidence level was quoted for the stream split information because the processes studied are dynamic and change daily according to conditions at each plant. In particular the amount of sands being recovered and fed to the processes can change markedly with time.

2.3.4 Using the Model - Historical technology advances

Gold mining is one of the oldest industries in the South African economy and the development of technology within the industry can be traced back over more than 100 years. Initially mercury amalgamation was used for gold recovery. Ore was crushed and then passed over a bed of mercury which concentrated the gold. This mixture or amalgam was retorted (boiled) to recover the gold.

The next process to be used was cyanide dissolution of the gold from finely milled ore. The liquor was then clarified and the gold recovered from solution by cementation. The solid product from cementation was smelted to produce a gold product for refining to final purity at Rand Refineries. In 1996 a small gold refinery was commissioned by one of the mines in the Free State. Previously governmental controls limited companies in their ability to refine their own gold. There is potential for this smaller scale refining of gold to play a larger role in the technology suite of the industry.

The third most notable technology shift was to Carbon-in-Pulp (CIP) gold recovery. The gold-cyanide solution is brought into contact with activated carbon onto which the gold adsorbs. The gold is then eluted or washed from the carbon, usually in a semi-batch process, which renders a cleaner solution for the downstream hydrometallurgy refining steps. The carbon has to be reactivated before entering the process again. In general, higher recoveries are achieved than in the normal clarification / cementation route.

This case study highlights the three technology-specific process routes. The base case presented is that of the status quo for 1996 where all the stream splits and unit assumptions are pertinent to what actually occurred in the South African industry in that year.

In the case of Amalgamation the same quantity of ore at the same average grade as was mined in 1996 was passed through the amalgamation process only. For Filtration alone all the ore was passed through the dissolution/filtration route only. The stream splits up-stream of the dissolution unit were assumed to correspond to those of 1996. Finally the CIP process route was followed. Here again the upstream stream splits from 1996 were used.

Figure A2.3-2 and Figure A2.3-3 graphically compare the three process routes described above. With respect to inputs to the process, there is the expected move from amalgamation reagents to CIP reagents.

Figure A2.3-3 shows how the waste generation profile alters with changing technology. The CIP process as a whole generates a greater quantity of waste than do the other process routes adopted. The nature of the waste also changes from a mercury-rich stream to one containing dissolved salts and cyanide.

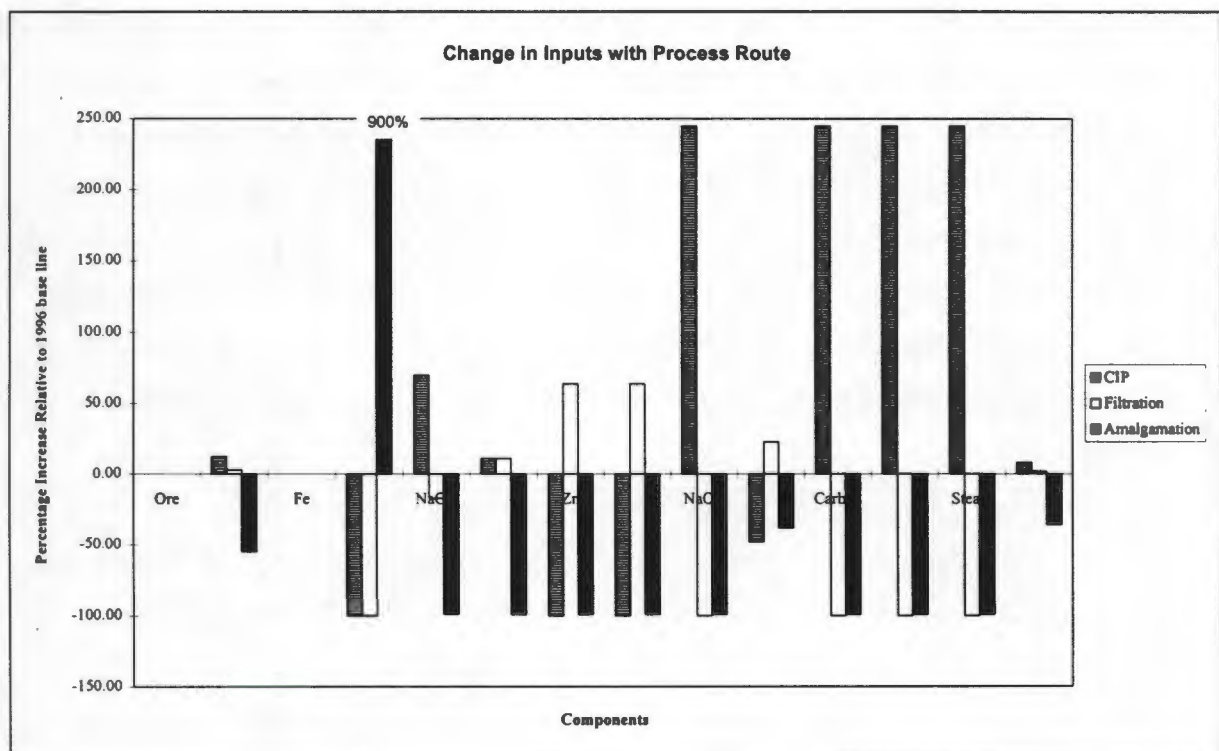


Figure A2.3-2: Change in Inputs to the Gold industry with change in Process Route

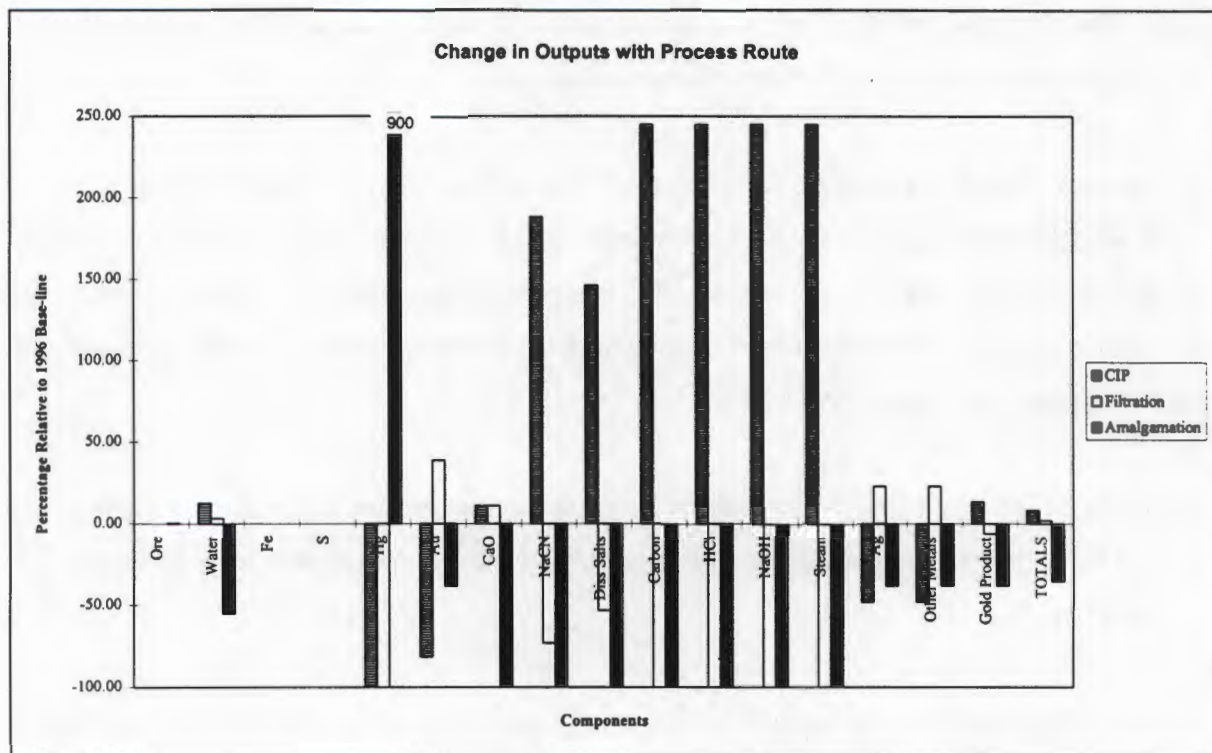


Figure A2.3-3: Change in Outputs from the Gold industry with change in Process Route

This scenario demonstrates how, on a macro-scale, the modelling technique can be employed within a generalised flowsheet to compare the waste streams generated by different technologies. The results gained from this example can be used to demonstrate what could happen should as yet unused gold mining rights be ceded to small mining ventures. This is a distinct possibility with the release of the Government White Paper on Minerals (the new policy document for the Minerals Industry, DMEA, 1998) which proposes that mineral rights no longer rest in the private sector. Should this be the case it is postulated that these rights will be re-issued to formally disadvantaged sectors of the economy. Unless they could afford to invest in the capital intensive routes of filtration or CIP, they might settle on mercury amalgamation to recover gold from ore. This would result in the mercury-rich waste stream illustrated.

The alternative to small separate processing plants is the establishment of a centralised processing plant where small operators could bring their material for beneficiation. This has the advantage of higher recoveries. The disadvantage would be dependence on the sampling techniques used to determine the amount of gold contributed by each operator as well as problems associated with transport.

A complete impact assessment (the third stage of LCA) would be required to highlight the trade-offs to be made between the higher water usage of the cyanide dissolution routes and the hazardous mercury waste associated with the amalgamation route.

2.3.5 Using the Model - Effect of changing operating variables

As has been pointed out, the flowsheets are structured to facilitate a change in a single process assumption and to observe the effect of this on the waste inventory of the process as a whole. It is also possible to see how such a change affects the waste generation of units down-stream from that unit. Effects of changing unit efficiency on the waste stream as a whole are made transparent.

Two main classes of variables exist within the system: continuous variables such as the grade of gold in the ore entering the process, and discrete variables such as an operating condition specific to a single unit. Changing a continuous variable makes determination of the waste inventory's sensitivity to that particular value possible. Changing a discrete variable reveals the sensitivity of the process to the efficiency of a specific unit operation.

To ascertain the system's sensitivity to these variables, assumptions within the flowsheets were changed by an arbitrarily determined set amount (a ten percent decrease in the values assumed for the process in 1996). Only those results in which the waste inventory changes markedly have been detailed here.

Two sets of results are included, these are presented in Figure A2.3-4 and Figure A2.3-5, show the percentage increases in waste generation concomitant to various changes in assumptions governing these unit operations.

Wash water is the water used in the washing of the filter cake, increasing the wash water will increase the recovery of gold for the process as a whole. Increasing the water reporting to the slimes dam is essentially decreasing the operating efficiency of the unit. This demonstrates that decreasing the efficiency of a single unit (the filtration unit) may lead to a increase in efficiency in downstream units. However, the streams leaving the downstream units have far higher environmental impact and thus the trade-off between the two competing efficiencies may be worthwhile. The balance here is the economic viability of the change.

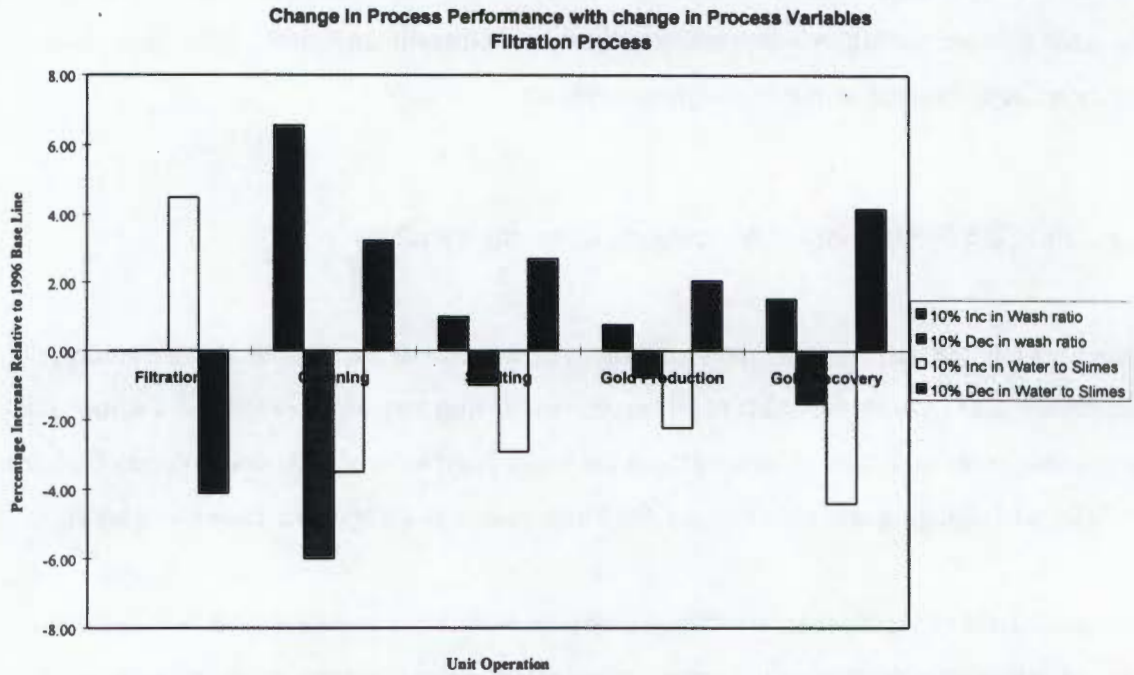


Figure A2.3-4: Percentage change in waste generated with change in operating conditions - Filtration Process

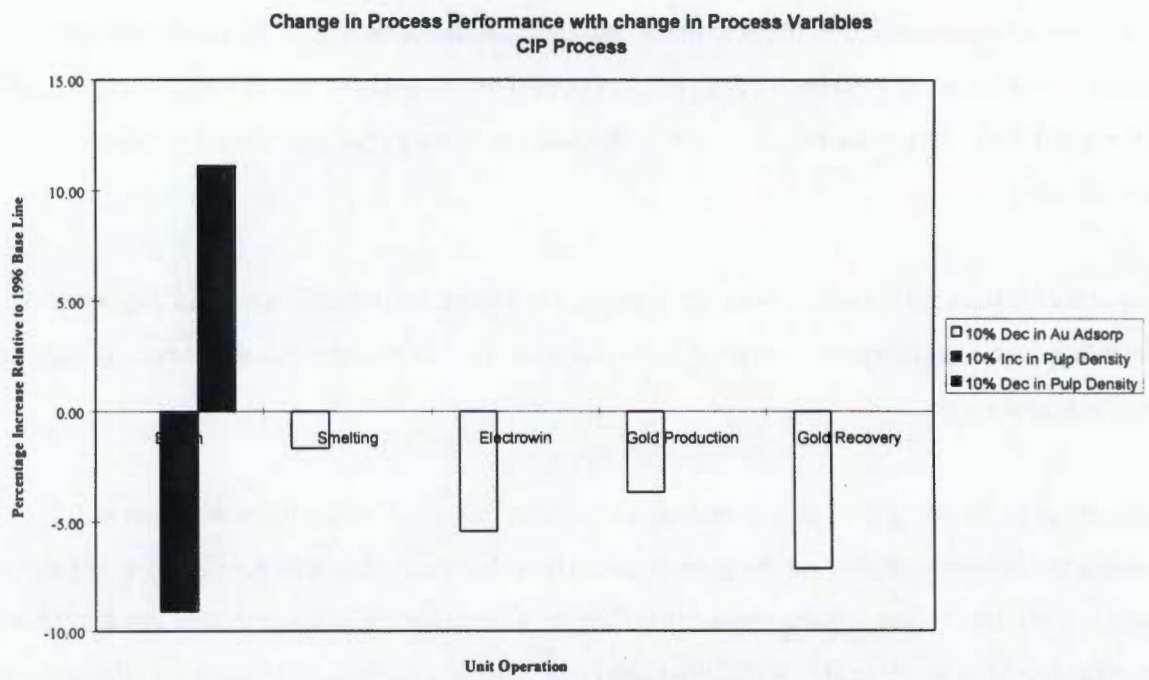


Figure A2.3-5: Percentage change in waste generated with change in operating conditions - CIP Process

These operating conditions reflect the changes in process performance which could result as the process ages. Again there is a trade-off to be made between increasing the efficiency of a single unit and decreasing the efficiency of another.

These results demonstrate the interlinking of the unit operations within a process. Such scenarios have some major implications uses :

- It is possible to infer what will happen during the life-time of a plant where unit operation efficiency is not stable; being low during commissioning, working up to a peak and then tailing off as a process ages.
- The models could also be used as a training tool to show operators how profoundly they can affect the waste generation potential of a process. Each decision could be related to a base case and it would be possible to quantify the effect on the waste inventory, of decisions made in the day-to-day running of the process.
- Also possible is linking of decisions, as they would occur within an hierarchical management structure, to their postulated environmental impact. For example, a change in process route or technology choice may be made by the higher echelons of management, and a change in operating unit efficiency by senior plant personnel, while a change in a single process variable for one unit operation may be decided by a plant operator. Figure A2.3-6 details the effect of three such changes, the first in a process route, the second in the efficiency of the CIP unit and the third in an increased wash ratio in the Filtration unit.

This figure shows how the waste stream generated would change according to the decisions detailed. They clarify how important the position of the environmental manager is within the corporation and shows that the post should not be created arbitrarily. They also underline the necessity for environmental training throughout the entire corporate structure if the environmental performance of the process is to improve.

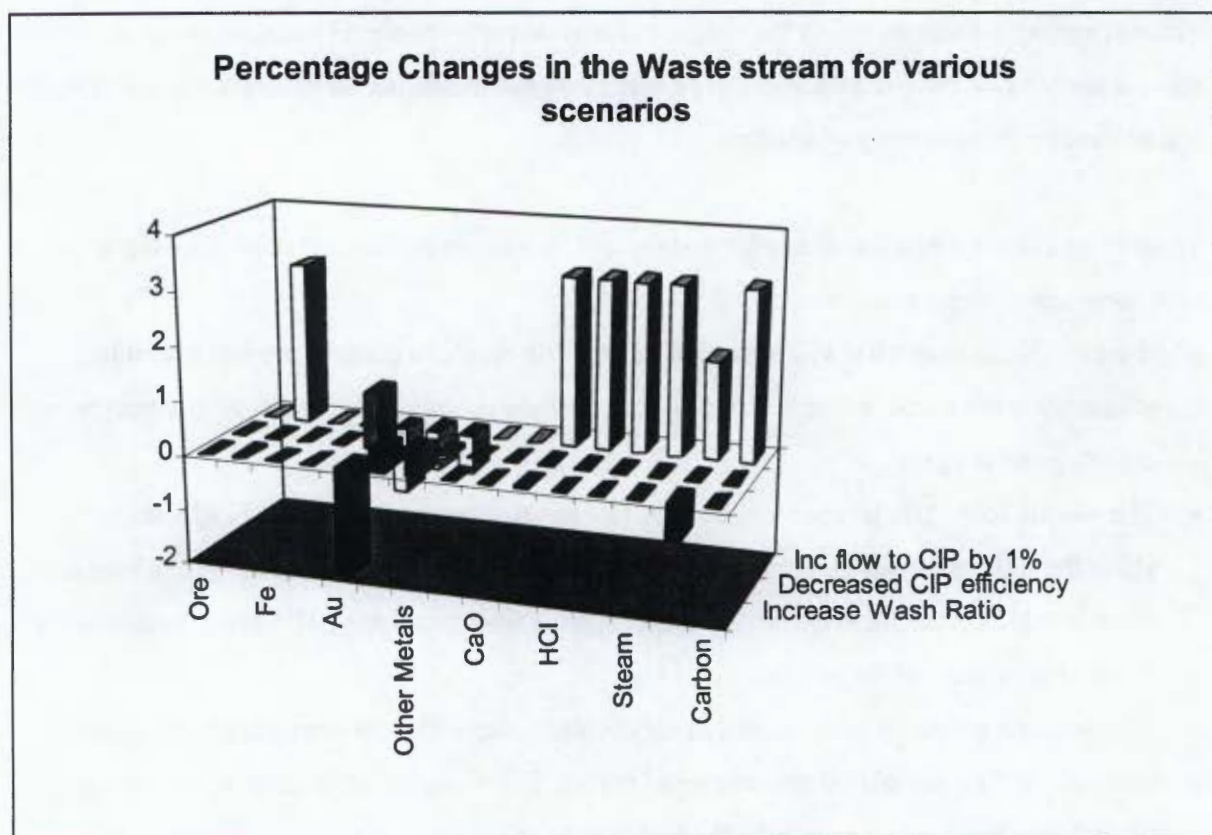


Figure A2.3-6: Changes in Waste stream for different levels of decision making

2.3.6 Printout of Spreadsheet-based Mass Balance

This printout includes:

- The complete assumption set
- The flowsheet which controls the stream splits
- Inputs to the process
- Wastes from the process
- Products from the process
- Unit operation mass balances

2.3.6.1 Assumption Set

Assumptions

Assumptions

Stream Splits

Mill to Grav Conc	10.00 to Grav Conc
Amal and Int Cyan	35.00 to Int Cyan
CIP and Filtration	32.19 to CIP
AARL and Zadra	100.00 to AARL
CIP to Electrowin	65.00 to Electrowin

Crush & Sort

Bypassing Crusher	10.00 % of feed
Water Usage	0.10 m ³ per ton crushed
Material sorted out	3.93 % of feed
<i>This is 7% of the Wits feed</i>	
Material re-crushed	50 % of feed less sorted
Energy consumption	2.5 kWh/t ore output from entire section

Milling

Mass percent to mill	70 %
Density from 3 ⁰ cyclone	1.5
Iron consumed	1.5 kg/ton for ball milling
	0.5 kg/ton for ROM milling
Energy required	25 kWh/ton ore milled

Gravity Separation

Rel Density	1.50
Grade to circuit	12.00 g/t
Final conc	16.00 %Au
Recovery	60.00 % of incoming Au
<i>NB Accounts for 20-50% of Au recovery in plants using this method</i>	
Prod H ₂ O mass %	40.00 %
Fe reporting to prod	1.00 %

Split between Amalgamation and Intensive Cyanidation

Intensive Cyanidation	35.00 % of total from Gravity Concentration
-----------------------	---

Amalgamate and Retort

Mass percent from mill	60.00 %
Feed grade	5.00 t Au /t ore
Fe removed	100 %
Dewater to	15.00 % H ₂ O
Amalgam contains	40.00 % Au by mass, rest Hg
Mass Hg lost	0.30 %
Recovery of Au in Hg	99.99 %
Recovery Au in Retort	100 %

Intensive Cyanidation

Water to solid ratio	3.00
Grade	5.00 g/ton
Dissolution	99.99 %
Cyanide conc	5.00 mass % NaCN

Assumptions

Flotation

Density	1.65 kg/m ³
Mass % water conc	50.00 %
Au in conc	42.50 g/t
S in conc	33.75 %
U in conc	600.00 g/t
Au recovery	59.88 %
S recovery	81.88 %
U recovery	26.25 %

Roasting & Acid production

Density	1.30 kg/m ³	Conversion factor	MJ/cal	
SO ₂ prod:S cons	1.00	0.0041868		
S converted SO ₂	100.00 %	Molar Masses	t/mol	
Scrubber etc eff	75.00 %	H ₂ O	1.80E-05 S	3.21E-05
H ₂ SO ₄ prod:SO ₂ cons	100.00 %	Fe	5.59E-05 Au	1.97E-04
Ambient temperature	20.00 °C		U	2.38E-04
Roasting temperature	625.00 °C			
ΔH rock	121.00 cal/g (°C)	293.00 K		
ΔH H ₂ O	605.00 cal/mol (K)	898.00 K		
ΔH Fe	4797.22 cal/mol (K)	5.07E-07 MJ/t		
ΔH S	2714.95 cal/mol (K)	4.56E-05 MJ/t		
ΔH Au	3912.85 cal/mol (K)	1.12E-03 MJ/t		
ΔH U	4017.20 cal/mol (K)	3.64E-04 MJ/t		
		3.23E-03 MJ/t		
		4.00E-03 MJ/t		

Thickener

Feed w:s	7.00
U/f water to solids	0.65
Lime dosing	30 g/t solid
Mass reporting to u/f	99 %

Dissolution

Feed w:s	0.65
NaCN addition	0.24 kg/ton ore
Require lime	1.10 kg/ton ore
Undissolved Au	0.50 % undissolved
Pachuca w:s	1.10
Feed grade	8.08 g/t Au
Solids Dissolving	0.5 %

Filtration & Clarification

Wash ratio	1.15 ton per ton solid
Water in solid product	30 %
Recycle Ratio	60 % of water to tailings is recycled

Cementation

Zn	40.00 g/m ³ soln
Nitrate	8.50 g/m ³ soln
Precipitation of Au	99.96 %
Density	1.00 t/m ³
Conc in solids	35 % Au
	22 % Zn
	5 % Ag
	3 % S
	10 % Pb
	2% % other

Assumptions

Calcining

H ₂ SO ₄ consumption	1.40 kg (100%)/kg Zn
H ₂ SO ₄ concentration	25.00 %
Decant to	50 mass % H ₂ O
Dewater to	10 mass % H ₂ O
Zn dissolution	100.00 %
Calcine has no solids loss	
Moisture after calcine	0.00 %

Carbon Adsorption

Au absorbed max	99.96 %	
Feed w:s	1.15	
Au in barren	0.010 g/t	
Au in feed	8.07 g/t	
Au in conc	4000.00 g/t	
C loss	0.22 %/day	30 days per month
	6.6 %/month	
NaCN	100.00 g/t	
C	20 g/ litre	equiv kg/m ³
Pulp density	1.45 t/m ³	
Other metals on C	0.10 %	

Elution - AARL

Water required	1.00 bed vol for flooded C feed		
Acid wash	1.00 bed vol	3.00 % HCl	
Water wash	2.00 bed vol water		
NaCN pretreat	0.60 bed vol	3.00 % NaCN	Density 1.00
3% NaCN equivalent to	60.00 kg NaCN/t C		These are per ton carbon eluted
NaOH pretreat		2.00 % NaOH	18.3 kg/t NaCN
Soft water strip	5.00 bed vol		52.8 kg/t HCL
Stripping efficiency	98.00 %		42.2 kg/t NaOH
Regeneration requires	1.00 kg steam/kg C		103.3 m ³ /t RWB
Adsorption of salts	50 %		80.9 kg/t Carbon
			Ratio C:Material
			0.20

Elution - Zadra

Water required	1.00 bed vol for flooded C feed	
Eluate required	34.00 bed vol	
NaCN	0.00 %	
NaOH	2.2 %	
NaCN equivalent	3 % NaCN is	60.00 kg NaCN/t C
Stripping efficiency	98.00 %	
Regeneration requires	1.00 kg steam/kg C	
Adsorption of metals	50.00 %	

Electrowinning

Efficiency	99.95 %			
Amperes	650 A			
Volts	4 V			
For	150 l/min	30 days a month		
equivalent to	6480 t/month			
Cathodes	6 weighing	1000 g		
Anodes	7			
Au uses	3 electrons	Molar Masses		
Fe uses	2 electrons	Au	196.97 g/mol	
Diss salts won	50 %	Ag	107.87 g/mol	
Faraday's Constant	96485.1343 C/mol charge	Base Metal	112.04 g/mol	Pb/Cu/Zn

Assumptions

Calcine smelting

Product	86 % Au
	10 % Ag
	4 % Base metals
Gold lost to atmos	0 %

Cathode Smelting

Product	86 % Au
	9 % Ag
	4 % Base metals
Gold lost to atmos	0 %

2.3.6.2 Flowsheet

Flow Sheet

Total production		15478057 t/month			
		133293.9072			
Wits ores		56.09	56.0929	All Sands	42.71 %
	t/m	8682083.33			42.7062549
Au Cont	g/t	5.00		Au Cont	t/m 6610098.333
U ₃ O ₈ Cont	g/t	78.53		U ₃ O ₈ Cont	g/t 2.50
S Cont	%	0.92		S Cont	g/t 93.33
					% 0.45
<u>% of Wits ores</u>				<u>% of Wits Sands</u>	
Sort	%	7.00		Mill Conventional	% 0.00
Crush & Screen	%	0.08		Float	% 100
ROM Mill	%	18.35		Conventional	% 100
Conventional Mill	%	81.65		S Acid production	% 100
Gravity	%	10.00		Calcine leach	% 100
Float	%	0.03			
CIP	%	32.19		Barberton ores	1.20 %
Conventional	%	67.81			1.200893652
S Acid production	%	0.03			
Calcine leach	%	0.03		Au Cont	t/m 185875.00
AARL elution	%	100.00		U ₃ O ₈ Cont	g/t 5.00
CIP to Electrowin	%	65.00		S Cont	% 0.92
					111304.6

Feed to Crush & Sort

Wits Ores	t/m	7088901.49	97.4449 %
Barberton Ores	t/m	185875.00	2.56 %
Total	t/m	7274776.49	100.00 %

Other inputs to Crush & Sort

Water	t/m	727477.65
Total	t/m	727477.65
Energy	MJ	62902191.80

Process stream from Crush & Sort

Wits Ore	t/m	6810555.80
Barberton Ore	t/m	178576.62
Water	t/m	727477.65
Total	t/m	7716610.07

BAL 0.00

Waste stream from Crush and Sort

Wits Ore	t/m	278345.69
Barberton Ore	t/m	7298.38
Water	t/m	0.00
Total	t/m	285644.07

Process Stream to Milling

Wits ore	t/m	6810555.80	97.44 % of ore
Barberton Ore	t/m	178576.62	2.56 % of ore
Water	t/m	727477.65	
ROM	t/m	1593181.84	
Total	t/m	7716610.07	

Other inputs to milling

Water	t/m	20012208.23
Fe	t/m	11280.29
Total	t/m	20023488.52
Energy	MJ	772408283.70

Process Stream from Milling

Wits Ore	t/m	8363030.87	97.44
Barberton Ore	t/m	219283.39	2.56
Water	t/m	20739685.88	
Fe	t/m	11280.29	
Total	t/m	29333280.43	

BAL 0.00

Process stream to Flotation

Sands	t/m	6610098.33
Barberton Ore	t/m	219283.39
Water	t/m	529911.69
Fe	t/m	288.22
Total	t/m	7359581.63
S (weighted avg)	%	0.47

Other inputs to Flotation

Water	t/m	10504893.47
Total	t/m	10504893.47

Process stream from Flotation

Ore	t/m	51014.65
Water	t/m	77052.76
S	t/m	26005.31
Au	t/m	2.17
U	t/m	30.63
Total	t/m	154105.51

BAL 0.00

Waste stream from Flotation

Ore	t/m	6746572.07
Water	t/m	10957752.40
Fe	t/m	288.22
S	t/m	5756.90
Total	t/m	17710369.59

Flow Sheet

Process stream to Roasting

Ore	t/m	51014.65
Water	t/m	77052.76
S	t/m	26005.31
Au	t/m	2.17
U	t/m	30.63
Total	t/m	154105.51

Other Inputs to Roasting

Energy	MJ	13.15
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Process stream from Roasting

Rock	t/m	51014.65
Au	t/m	2.17
U	t/m	30.63
Total	t/m	51047.45

Waste stream from Roasting

Water	t/m	77052.76
S	t/m	6501.33
Total	t/m	83554.08

Product from Roasting

S	t/m	19503.98
Total	t/m	19503.98

BAL

0.00

Process stream to Gravity Conc

Wits ore	t/m	836303.09
Water	t/m	2020977.419
Fe	t/m	1099.207124
Total	t/m	2858379.71

Other Inputs to Gravity Conc

Water	t/m	0.00
Total	t/m	0.00

Process stream from Gravity Conc

Wits ore	t/m	37.63
Water	t/m	32.42
Fe	t/m	10.99
Total	t/m	81.04

Waste stream from Gravity Conc

Wits ore	t/m	836265.45
Water	t/m	2020945.00
Fe	t/m	1088.22
Total	t/m	2858298.67

BAL

0.00

Process stream to Int Cyanidation

Wits ore	t/m	13.17
Water	t/m	11.35
Fe	t/m	3.85
Total	t/m	28.36

Other Inputs to Int Cyanidation

Water	t/m	39.71
NaCN	t/m	5.54
Total	t/m	45.25

Process stream from Int Cyanidation

Ore	t/m	13.17
Water	t/m	51.06
NaCN	t/m	5.54
Fe	t/m	3.85
Au soln	t/m	0.000066
Total	t/m	73.61

BAL

0.00

Process stream to Amalgamation

Wits ore	t/m	24.46
Water	t/m	21.07
Fe	t/m	7.14
Total	t/m	52.68

Other Inputs to Amalgamation

Hg	t/m	0.0000006
Total	t/m	0.0000006

Waste stream from Amalgamation

Ore	t/m	24.46
Water	t/m	21.07
Fe	t/m	7.14
Hg	t/m	0.00
Au	t/m	0.00
Total	t/m	52.68

Product from Amalgamation

Au	t/m	0.000122
Total	t/m	0.000122

BAL

0.000122309

Process stream to Thickener

Wits ore	t/m	7526727.78
Water	t/m	18188796.77
Fe	t/m	9892.86
Total	t/m	25725417.42

Other Inputs to Thickener

CaO	t/m	226.10
Total	t/m	226.10

Waste stream from Thickener

Ore	t/m	75267.28
Water	t/m	13338834.42
Fe	t/m	98.93
Total	t/m	13414200.63

Process stream from Thickener

Ore	t/m	7451460.51
Water	t/m	4849962.35
Fe	t/m	9793.94
CaO	t/m	226.10
Total	t/m	12311442.89

BAL

0.00

Flow Sheet

Stream from Roasting and Stream from Thickening

Process stream to Dissolution

Ore	t/m	7502507.96
Water	t/m	4849962.35
Fe	t/m	9793.94
CaO	t/m	226.10
Total	t/m	12362490.34

Other inputs to Dissolution

Water	t/m	3411625.73
CaO	t/m	8026.66
NaCN	t/m	1800.60
Total	t/m	3421452.99

Liquid stream from Dissolution

Water	t/m	8261588.08
Au	t/m	60.31
NaCN	t/m	1800.60
Diss Salts	t/m	37512.24
Total	t/m	8300961.22

Solid stream from Dissolution

Ore	t/m	7464935.11
Fe	t/m	9793.94
Au	t/m	0.30
CaO	t/m	8252.76
Total	t/m	7482982.11

BAL

0.00

Percentage to CIP

32.19 %

Liquid stream to CIP

Water	t/m	2659088.34
Au	t/m	19.41
NaCN	t/m	579.54
Diss Salts	t/m	12073.75
Total	t/m	2671761.05

Solid stream to CIP

Ore	t/m	2402676.31
Fe	t/m	3152.29
Au	t/m	0.10
CaO	t/m	2656.25
Total	t/m	2408484.95

Other inputs to CIP

Water	t/m	97996.64
Carbon	t/m	2187.26
Total	t/m	100183.90

Process stream from CIP

Au	t/m	19.40
Diss Salts	t/m	12.07
Total	t/m	31.48

Waste stream from CIP

Ore	t/m	2402676.31
Water	t/m	2757084.98
Fe	t/m	3152.29
Au	t/m	0.11
CaO	t/m	2656.25
NaCN	t/m	579.54
Diss Salts	t/m	12061.68
Carbon	t/m	2187.26
Total	t/m	5180398.42

Carbon Load 33140.36 t

BAL

0.00

Percentage to AARL elution

100.00 %

Process stream to AARL elution

Au	t/m	19.40
Diss Salts	t/m	12.07
Total	t/m	31.48

Other inputs to AARL elution

Water	t/m	3423399.49
HCl	t/m	1749.81
NaCN	t/m	606.47
NaOH	t/m	1398.52
Steam	t/m	33140.36
Total	t/m	3460294.65

Carbon load 33140.36 t

Process stream from AARL elution

Water	t/m	0.00
Au	t/m	19.01
Diss salts	t/m	6.04
Total	t/m	25.05

Waste stream from AARL elution

Water	t/m	3423399.49
Au	t/m	0.39
Diss salts	t/m	6.04
HCl	t/m	1749.81
NaCN	t/m	606.47
NaOH	t/m	1398.52
Steam	t/m	33140.36
Total	t/m	3460301.08

BAL

0.00

Process stream to Zadra Elution

Au	t/m	0.00
Diss Salts	t/m	0.00
Total	t/m	0.00

Other inputs to Zadra elution

Water	t/m	0.00
NaCN	t/m	0.00
NaOH	t/m	0.00
Steam	t/m	0.00
Total	t/m	0.00

Carbon Load 0.00 t

Flow Sheet

Process stream from Zadra elution

Water	t/m	0.00
Au	t/m	0.00
Diss salts	t/m	0.00
NaCN	t/m	0.00
NaOH	t/m	0.00
Total	t/m	0.00

BAL

Waste stream from Zadra elution

Water	t/m	0.00
Au	t/m	0.00
Diss salts	t/m	0.00
Steam	t/m	0.00
Total	t/m	0.00

0.00

Process stream from CIP overall

Au	t/m	19.01
Diss salts	t/m	6.04
Water	t/m	0.00
NaCN	t/m	0.00
NaOH	t/m	0.00
Total	t/m	25.05

Balance of Dissolution and all Int Cyanidation

67.81

Liquid Stream to Filtration

Water	t/m	5602550.79
Au	t/m	40.90
NaCN	t/m	1226.59
Diss Salts	t/m	25438.49
Total	t/m	5629256.77

Solid stream to Filtration

Ore	t/m	5062271.98
Fe	t/m	6645.49
Au	t/m	0.21
CaO	t/m	5596.51
Total	t/m	5074514.18

Other inputs to Filtration

Water	t/m	5821612.77
Total	t/m	5821612.77

Process stream from Filtration

Au	t/m	33.11
Diss Salts	t/m	20595.82
Water	t/m	9249371.86
NaCN	t/m	993.09
Total	t/m	9270993.88

Waste stream from Filtration

Ore	t/m	5062271.98
Water	t/m	2174791.70
Fe	t/m	6645.49
Au	t/m	7.99
CaO	t/m	5596.51
NaCN	t/m	233.50
Diss Salts	t/m	4842.67
Total	t/m	7254389.84

BAL

0.00

Percentage CIP to Electrowin

65.00 %

Process stream to Electrowinning

Au	t/m	12.36
Diss salts	t/m	3.92
Water	t/m	0.00
NaCN	t/m	0.00
NaOH	t/m	0.00
Total	t/m	16.28

Other Inputs to Electrowinning

Fe	t/m	2.34
----	-----	------

Process stream from Electrowinning

Au	t/m	12.35
Diss salts	t/m	1.96
Total	t/m	14.32

Waste stream from Electrowinning

Liquid	t/m	0.00
Au	t/m	0.01
Diss salts	t/m	1.96
Fe	t/m	2.34
Total	t/m	4.30

BAL

0.00

Process stream to Melting

Au	t/m	12.35
Diss salts	t/m	1.96
Total	t/m	14.32

BAL

Waste stream from Melting

Au	t/m	0.00
Diss salts	t/m	0.09
Total	t/m	0.09

0.00

Product from Melting

Au	t/m	12.35
Ag	t/m	1.29
Base metals	t/m	0.57
Total	t/m	14.22

Flow Sheet

Balance of CIP and all of Filtration

Process stream to Cementation

Au	t/m	39.77
Diss salts	t/m	20597.93
Water	t/m	9249371.86
NaCN	t/m	993.09
NaOH	t/m	0.00
Total	t/m	9271002.65

35.00 %

Other inputs to Cementation

Zn	t/m	369.97
Nitrate	t/m	78.62
Total	t/m	448.59

Process stream from Cementation

Liquid	t/m	9271337.63
Au	t/m	39.77
Zn	t/m	25.00
Silver	t/m	5.68
Sulphur	t/m	3.41
Pb	t/m	11.36
Other	t/m	28.40
Total	t/m	9271451.24

BAL

0.00

Process stream to Calcining

Liquid	t/m	9271337.63
Au	t/m	39.77
Zn	t/m	25.00
Silver	t/m	5.68
Sulphur	t/m	3.41
Pb	t/m	11.36
Other	t/m	28.40
Total	t/m	9271451.24

Other inputs to Calcining

Liquid	t/m	104.98
H ₂ SO ₄	t/m	34.99
Total	t/m	139.97

Process stream from Calcining

Au	t/m	39.77
Zn	t/m	0.00
Ag	t/m	5.68
S	t/m	3.41
Pb	t/m	11.36
Other	t/m	28.40
Total	t/m	88.62

BAL

0.00

Waste stream from Calcining

Liquid	t/m	9271502.60
Total	t/m	9271502.60

Process stream to Smelting

Au	t/m	39.77
Zn	t/m	0.00
Ag	t/m	5.68
S	t/m	3.41
Pb	t/m	11.36
Other	t/m	28.40
Total	t/m	88.62

BAL

0.00

Waste stream from Smelting

Au	t/m	0.00
Ag	t/m	1.06
Other	t/m	41.32
Total	t/m	42.38

Product from Smelting

Au	t/m	39.77
Ag	t/m	4.62
Other	t/m	1.85
Total	t/m	46.24

Flow Sheet

Check block 1

Inputs		Outputs	
Process stream from Dissolution		Process stream from CIP	
	15783943.33		31.48
Process stream from Int Cyanide		Waste from CIP	
	73.61		5180398.42
Input to CIP		Waste from Filtration	
	100183.90		7254389.84
Input to Filtration		Process stream from Filtration	
	5821612.77		9270993.88
Total	21705813.62		21705813.62
Difference	0.00		

Stream from Dissolution + Stream from Int Cyanide	
	15784016.95
Stream to CIP + Stream to Filtration	
	15784016.95
Difference	0.00

Check block 2

Inputs		Outputs	
Process stream to Gravity Conc		Waste from Gravity Conc	
	2858379.71		2858298.67
Inputs to Gravity Conc		Process stream from Int Cyanide	
	0.00		73.61
Inputs to Int Cyanide		Waste from Amalgamation	
	45.25		52.68
Inputs to Amalgamation		Product from Amalgamation	
	0.00		0.00
Total	2858424.96		2858424.96
Difference	0.00		

Stream from Grav Conc	
	81.04
Stream to Int Cyanide + Stream to Amalgamate	
	81.04
Difference	0.00

Check block 3

Inputs		Outputs	
Process stream from CIP		Process stream from AARL Elution	
	31.48		25.05
Other inputs to AARL Elution		Waste stream from AARL Elution	
	3460294.65		3460301.08
Other inputs to Zadra Elution		Process Stream from Zadra Elution	
	0.00		0.00
		Waste stream from Zadra Elution	
			0.00
Total	3460326.13		3460326.13
Difference	0.00		

Stream from CIP	
	31.48
Stream to AARL + Stream to Zadra	
	31.48
	0.00
Stream from AARL + Stream from Zadra	
	25.05
Stream from CIP	
	25.05
Difference	0.00

Flow Sheet

Check block 4

Inputs		Outputs	
Stream to Electrowin		Waste from Electrowin	
	16.28		4.30
Inputs to Electrowin		Waste from Melting	
	2.34		0.09
Stream to Cementation		Product from Melting	
	9271002.65		14.22
Inputs to Cementation		Waste from Calcining	
	448.59		9271502.60
Inputs to Calcining		Waste from Smelting	
	139.97		42.38
		Product from Smelting	
			46.24
Total	9271609.83		9271609.83
Difference	0.00		

Stream from CIP + Stream from Filtration

9271018.93

Stream to Electrowin + Stream to Cementation

9271018.93

Difference

0.00

2.3.6.3 Inputs, Products and Wastes

Products

Products	Crush & S	Milling	Flotation	Roasting	Grav Co	Int Cyan	Amalgama	Thickener	Dissoluti	CIP	Elution	Filtration	Electrowi	Melting	Cement	Calcining	Smelting
S				19503.98													
Au							0.000122								12.35		39.77
Ag															1.29		4.62
Other Metals															0.57		1.85
TOTALS	0	0	0	19503.98	0	0	0.000122	0	0	0	0	0	0	0	14.22	0	46.24

OVERALL PRODUCT 19.56 kt/month 60.46 t/month Gold

OVERALL OUTPUT 43747.80 kt/month

OVERALL INPUT 43747.80 kt/month

OVERALL BALANCE 0.00 kt/month

Gold Recovery Gold in 60.87 t/month 3.93 average g/t
 Gold out 52.12 t/month
 Gold recov 85.63 %
 recover 85.153 %
 which is kg
 amount in is kg

Waste Streams

Waste	Crush & Sort	Milling	Flotation	Roasting	Grav Conc	Int Cya	Amalgamat	Thickener	Dissolu	CIP	Elution	Filtration	Electrowi	Melting	Cement	Calcining	Smelting
Ore	285644.07		6746572.07		836265.45		24.46	75267.28		2402676.31		5062271.98					
Water	0.00		4383100.96	77052.76	808378.00		21.07	5335533.8		2757084.98	3423399.49	2174791.70	0.00				
Fe			288.22		1088.22		7.14	98.93		3152.29		6645.49	2.34				
S			5756.90	650 1.33													
Hg							0.0000006										
Au							0.000			0.11	0.39	7.99	0.01	0.00			0.00
CaO										2656.25		5596.51					
NaCN										579.54	606.47	233.50					
Diss Salts										12061.68	6.04	4842.67	1.96	0.09			
Carbon										2187.26							
HCl											1749.81						
NaOH											1398.52						
Steam											33140.36						
Ag																	1.06
Other Metals																	41.32
TOTALS	285644.07	0.00	11135718.1	33554.08	1645731.67	0.00	52.68	541090.0	0.00	5180398.42	3460301.08	7254389.84	4.30	0.09		0.00	42.38
OVERALL WASTE		43728.24	kl/month														

2.3.6.4 Unit operation mass balances

Input Streams

Input	Feed	Crush & Sort	Milling	Flotation	Grav Con	Int Cyan	Amalgamate	Thickene	Dissolution	CIP	Elution	Filtration	Electro	Cement	Calcining
Ore	15478056.67														
Water		727477.65	4221689.1	10504893	0.00	39.71			3411625.73	97996.64	3423399.5	5821612.8			104.98
Fe			11280.29		0.00								2.34		
Hg							0.0000006								
NaCN						5.54			1800.60		606.47				
CaO								226.10	8026.66						
Zn														369.97	
Nitrate														78.62	
NaOH											1398.52				
H2SO4															34.99
Carbon										2187.26					
HCl											1749.81				
Steam											33140.36				
TOTALS	15478056.67	727477.65	4232969.4	10504893	0.00	45.25	0.0000006	226.10	3421452.99	100183.9	3460294.7	5821612.8	2.34	448.59	139.97
OVERALL INPUT		43747.80	kt/month												
TOTAL FEED		15478.06	kt/month												
INPUTS - FEED		28269.75	kt/month												

Milling

Assumptions:

Mass: percent to mill	70 %	Densities
Feed from crushing	6989132.42 t/month ore	2.70
	727477.65 t/month H ₂ O	1.00
ROM Feed	1593181.84 t/month	7.80
Iron consumed	1.5 kg/ton for ball milling	
	0.5 kg/ton for ROM milling	
Density from 3 ^o cyclone	1.5	90.57
Energy required	25 kWh/ton ore milled	

Mass: in	29333280.43	0.00
Mass: out	29333280.43	

		Mass %					
		1	2	3	4	5	6
		Feed	Make-up H ₂ O	Iron Addition	To Mill	Cyclone H ₂ O	From cyclone
t/month	Ore	8582314.26	0.00	0.00	8582314.26	0.00	8582314.26
t/month	Water	727477.65	2950657.04	0.00	3678134.68	17061551.20	20739685.88
t/month	Fe	0.00	0.00	11280.29	11280.29	0.00	11280.29
	Total	9309791.91	2950657.04	0.00	12271729.24	17061551.20	29333280.43
	Energy				772408283.70		

Input Streams

	1	2	3	5
Ore	8582314.26	0.00	0.00	0.00
Water	727477.65	2950657.04	0.00	17061551.20
Fe	0.00	0.00	11280.29	0.00
	9309791.91	2950657.04	11280.29	17061551.20

Output Streams

	6
Ore	8582314.26
Water	20739685.88
Fe	11280.29
Total	29333280.43
Rel density	

Flotation

Assumptions:

Density	1.65 kg/m ³	38.23 mass percent
Feed	6829381.73 t/month ore	
	529911.69 t/month H ₂ O	Densities
	288.22 t/month Fe	2.70
	0.47 % S	1.00
Mass % water conc	50.00 %	7.80
Au in conc	42.50 g/t	
S in conc	33.75 %	
U in conc	600.00 g/t	
Au recovery	59.88 %	
S recovery	81.88 %	% H ₂ O
U recovery	26.25 %	50.00

Mass in	17864475	0.00
Mass out	17864475	% S
		0.47
		Density
		1.65
		S %
		33.75

		1	2	3	4	5	6
		Feed	Make up H ₂ O	To Float	Concentrate	Au conc	Tails
t/month	Ore	6797619.52		6797619.52	51047.45	51014.65	6746572.07
t/month	Water	529911.69	10504893.47	11034805.16	77052.76	77052.76	10957752.40
t/month	Fe	288.22		288.22			288.22
t/month	S	31762.20		31762.20	26005.31	26005.31	5756.90
t/month	Au					2.17	
t/month	U					30.63	
	Total	7359581.63	10504893.47	17864475.10	154105.51	154105.51	17710369.59
		2.58		1.65	1.85		1.65

WASTE

0.465062075 38.23045404

Input Streams

	1	2
Ore	6797619.52	
Water	529911.69	10504893.47
Fe	288.22	
S	31762.20	
Au		
U		
Total	7359581.63	10504893.47

Output Streams

	5	6
Ore	51014.65	6746572.07
Water	77052.76	10957752.40
Fe		288.22
S	26005.31	5756.90
Au	2.17	
U	30.63	
Total	154105.51	17710369.59

WASTE

Roasting & Acid

Assumptions:

Density	1.30 kg/m ³	Densities	Masses
Feed	51014.65 t/month rock	2.70	32.06 S
	77052.76 t/month H ₂ O	1.00	16.00 O
	0.00 t/month Fe	7.80	1.01 H
	26005.31 t/month S		
	2.170 t/month Au	2169.52 kg/month	
	30.628 t/month U	30628.47 kg/month	

SO ₂ prod:S cons	1.00
S converted SO ₂	100.00 %
Scrubber etc eff	75.00 %
H ₂ SO ₄ prod:SO ₂ cons	100.00 %
Weighted average DIH	8.53297E-05 MJ/t

Mass in	154105.5	0.00
Mass out	154105.5 % S	

		1	2	3	4	5	6
		Feed	Roast (s)	Roast (g)	Gas Cleaners	To atmos	Acid plant
t/month	Rock	51014.65	51014.65				
t/month	Water	77052.76		77052.76		77052.76	
t/month	Fe						
t/month	S	26005.31		26005.31	19503.98	6501.33	19503.98
t/month	Au	2.170	2.170				
t/month	U	30.628	30.628				
	Total	154105.51	51047.45	103058.06	19503.98	83554.08	19503.98
	Energy	13.15				WASTE	PROD
t/month	SO ₂				38968.30		
t/month	H ₂ SO ₄						59658.88

Input Streams

	1	Energy
Rock	51014.65	
Water	77052.76	
Fe		
S	26005.31	
Au	2.17	
U	30.63	
Total	154105.51	13.15

Output Streams

	2	5	6
Rock	51014.65		
Water		77052.76	
Fe			
S		6501.33	19503.98
Au	2.170		
U	30.628		
Total	51047.45	83554.08	19503.98
		WASTE	PROD

Gravity

Assumptions:

Rel Density	1.50
Feed	836303.09 t/month ore
	2020977.42 t/month H ₂ O
	1099.21 t/month Fe
Grade to circuit	12.00 g/t
Final conc	16.00 %Au
Recovery	60.00 % of incoming Au
<i>NB Accounts for 20-50% of Au recovery in plants using this method</i>	
Prod H ₂ O mass %	40.00 %
Fe reporting to prod	1.00 %
Mass in	2858380 0.00
Mass out	2858380

Densities

2.70
1.00
7.80

Input Streams

	1	2
Ore	836303.09	0.00
Water	2020977.42	0.00
Fe	1099.21	0.00
Total	2858379.71	0.00

Output Streams

	4	5
Ore	37.634	836265.45
Water	32.417	2020945.00
Fe	10.992	1088.22
Total	81.043	2858298.67

WASTE
to slimes

	Grade	Rel Density	Recovery		
	12.00	1.50	60.000		
	1	2	3	4	5
	To circuit	Make up H ₂ O	To tables etc	Product	From tables
t/month	Ore	836303.09	836303.09	37.634	836265.45
t/month	Water	2020977.42	0.00	2020977.42	32.417
t/month	Fe	1099.21	0.00	1099.21	10.992
	Total	2857280.51	0.00	2858379.71	81.043
ton/month	Au	10.04	10.04	6.021	4.01
grade g/ton		12	12	160000	4.80

Int Cyanide

Assumptions:

Water to solid ratio	3.00
Feed	13.17 t/month ore
	11.35 t/month H2O
	3.85 t/month Fe
Grade	5.00 g/ton
Dissolution	99.99 %
Cyanide conc	5.00 mass % NaCN

Densities

	2.70
	1.00
	7.80

Masses

Na	23
C	12
N	14
Total NaCN	49

w:s

Dissolution %

3	99.000
---	--------

Mass in	73.61	0.00
Mass out	73.61	

Input Streams

	1	2	4
Ore	13.17	0.00	0.00
Water	11.35	39.71	0.00
NaCN	0.00	0.00	5.54
Fe	3.85	0.00	0.00
Au soln	0.00	0.00	0.00
Total	28.36	39.71	5.54

Output Streams

	6
Ore	13.17
Water	51.06
NaCN	5.54
Fe	3.85
Au soln	0.00
Total	73.61

		1	2	3	4	5	6
		To circuit	Make up H2O	To tanks	Add NaCN	Tanks	To Filter
t/month	Ore	13.17		13.17		13.17	13.17
t/month	Water	11.35	39.71	51.06		51.06	51.06
t/month	NaCN				5.54	5.54	5.54
t/month	Fe	3.85		3.85		3.85	3.85
t/month in solution	Au						0.000066
	Total	28.36	39.71	68.08	5.54	73.61	73.61
ton/month	Au	0.00	0.00			0.00	0.00
grade g/ton		5				5	0.00

Amalgamate & Retort

Assumptions:

Mass percent from mill
Feed

60.00 %
24.46 t/month ore Ore
21.07 t/month H2O H2O
7.14 t/month Fe Fe
5.00 t Au /t ore Au

Densities Unit Vol
2.70
1.00
7.80
19.30

Feed grade

Fe removed

Dewater to

Amalgam contains

Mass Hg lost

Recovery of Au in Hg

Recovery Au in Retort

100.00 %
15.00 % H2O
40.00 % Au by mass, rest Hg
0.30 %
99.99 %
100.00 %

Mass in

Mass out

52.68 0.00
52.68

Recovery Mass % Au recov
100.000 60.00 99.99

Input Streams

1 2

Ore 24.46
Water 21.07
Fe 7.14
Hg 0.0000006
Au 0.00
Total 52.68 0.00

Output Streams

3 6 8 9

Ore 24.46
Water 21.07
Fe 7.14 0.00
Hg 0.0000006
Au 0.00012
Total 7.14 45.53 0.00 0.00
Au recov Hg loss
100.00 0.3

	1	2	3	4	5	6	7	8	9
	Feed	Hg make-up	Removal of Fe	To amalgam	Amalgam	Liquid	Retort	Hg loss	Product
t/month Ore	24.46			24.46		24.46			
t/month Water	21.07			21.07		21.07			
t/month Fe	7.14		7.14	0.000		0.00			
t/month Hg		0.0000006		0.00018	0.00018		0.00018	0.0000006	
t/month Au	0.0001223			0.00012	0.00012		0.00012		0.00012
t/month Total	52.68	0.00	7.14	45.533	0.00031	45.53	0.00031	0.00	0.00

WASTE

40

Thickening

Assumptions:

Feed w:s	7
Feed	7526727.78 t/month ore
	18188796.77 t/month H ₂ O
	9892.86 t/month Fe
U/f water to solids	0.65
Lime dosing	30 g/t solid
Mass reporting to u/f	99 %

Densities

2.70
1.00
7.80

Input Streams

	1	2
Ore	7526727.78	
Water	18188796.77	
Fe	9892.86	
CaO		226.099
Total	25725417.42	226.099

Output Streams

	4	5
Ore	7451460.51	75267.28
Water	4849962.35	13338834.42
Fe	9793.94	98.93
CaO	226.099	
Total	12311442.89	13414200.63

Mass in	25725643.52	0.00	% of mass
Mass out	25725643.52		99.00

g/t solid	w:s
30.00	0.65

		1	2	3	4	5
		Feed	Lime add	To thickener	Underflow	Overflow
t/month	Ore	7526727.78		7526727.78	7451460.51	75267.28
t/month	Water	18188796.77		18188796.77	4849962.35	13338834.42
t/month	Fe	9892.86		9892.86	9793.94	98.93
t/month	CaO		226.099	226.099	226.099	
	Total	25725417.42	226.099	25725643.52	12311442.89	13414200.63

Dissolution

Assumptions:

Feed w:s	0.65
Feed	7502507.96 t/month ore
	4849962.35 t/month H ₂ O
	9793.94 t/month Fe
	226.10 t/month CaO
NaCN addition	0.24 kg/ton ore
Require lime	1.10 kg/ton ore
Undissolved Au	0.50 % undissolved
Pachuca w:s	1.10
Feed grade	8.08 g/t Au
Solids Dissolving	0.50 %

Densities

2.70 Ore
1.00 Water
7.80 Fe

Input Streams

	1	2	3
Ore	7502507.96		
Water	4849962.35	3411625.73	
Fe	9793.94		
Au			
CaO	226.099		8026.66
NaCN			1800.60
Diss Salts			
Total	12362490.34	3411625.73	9827.26

Output Streams

	5	6
Ore		7464935.11
Water	8261588.08	
Fe		9793.94
Au	60.305	0.303
CaO		8252.76
NaCN	1800.602	
Diss Salts	37512.24	
Total	8300961.22	7482982.11

Mass in	15783943	0.00
Mass out	15783943	

NaCN kg/t

0.24

CaO kg/t

1.10

w:s ratio

1.10

Solid Diss

0.5

Au undiss

0.50

NaCN

1800.602

Diss Salts

37512.24

Total

8300961.22

7482982.11

	1	2	3	4	5	6
	Feed	Water add	Reagent add	To pachuca	Liquid	Solid
t/month Ore	7502507.96			7502447.35		7464935.11
t/month Water	4849962.35	3411625.73		8261588.08	8261588.08	
t/month Fe	9793.94			9793.94		9793.94
t/month Au				60.608	60.305	0.303
t/month CaO	226.10		8026.66	8026.66		8252.76
t/month NaCN			1800.60	1800.60	1800.60	
t/month Diss Salts					37512.24	
Total	12362490.34	3411625.73	9827.26	15783717.23	8300961.22	7482982.11

Filtration and Clarification

Assumptions:

Feed - liquid	5602550.79 t/month H ₂ O
	40.90 t/month Au
	1226.59 t/month NaCN
	25438.49 t/month salts
Feed - solid	5062271.98 t/month ore
	6645.49 t/month Fe
	0.21 t/month Au
	5596.51 t/month CaO
Wash ratio	1.15 ton per ton solid
Water in solid produ	30 %

Densities

2.70
1.00
7.80

Input Streams

	1	2	3
Ore		5062271.98	
Water	5602550.79		5821612.77
Fe		6645.49	
Au	40.90	0.21	
CaO		5596.51	
NaCN	1226.59		
Diss Salts	25438.49		
Total	5629256.77	5074514.18	5821612.77

Output Streams

	6	7
Ore		5062271.98
Water	9249371.86	2174791.70
Fe		6645.49
Au	33.11	7.99
CaO		5596.51
NaCN	993.09	233.50
Diss Salts	20595.82	4842.67
Total	9270993.88	7254389.84

Mass in	16525384	0.00
Mass out	16525384	

		1	2	3	4	5	6	7	
		Feed liq	Feed solid	Wash H ₂ O	Liquid phase	Liquid loss	Liquid prod	Solid prod	
				1.15		0.19		30.00	
t/month	Ore		5062271.98					5062271.98	
t/month	Water	5602550.79		5821612.77	11424163.56	2174791.70	9249371.86	2174791.70	
t/month	Fe		6645.49					6645.49	
t/month	Au	40.895	0.206		40.895	7.785	33.11	7.991	
t/month	CaO		5596.51					5596.51	
t/month	NaCN	1226.59			1226.59	233.504	993.09	233.50	
t/month	Diss Salts	25438.49			25438.49	4842.67	20595.82	4842.67	
t/month	Total	5629256.77	5074514.18	5821612.77	11450869.54	2179875.66	9270993.88	7254389.84	

Cementation

Assumptions:

Zn
Nitrate
Precipitation of Au
Feed

40.00 g/m³ soln
8.50 g/m³ soln
99.96 %
9249371.86 t/month H₂O
39.77 t/month Au
993.09 t/month NaCN
0.00 t/month NaOH
20597.93 t/month Diss Salts
1.00 t/m³
35.00 % Au
22.00 % Zn
5.00 % Ag
3.00 % S
10.00 % Pb
25.00 % Other

Mass

Element

207.19 Pb
14.0067 N
15.9994 O

Input Streams

	1	2
Water	9249371.86	
Au	39.77	
NaCN	993.09	
NaOH	0.00	
Zn		369.97
Nitrate		78.62
Diss Salts	20597.93	
Silver		
Sulphur		
Pb		
Other		
Total	9271002.65	448.59

Output Streams

	4	5
Water	9249371.86	
Au		39.77
NaCN	993.09	
NaOH	0.00	
Zn	344.98	25.00
Nitrate	78.62	
Diss Salts	20549.08	
Silver		5.68
Sulphur		3.41
Pb		11.36
Other		28.40
Total	9271337.63	113.61

	1	2	3	4	5
	Feed	Reagent Add	To Cementation	Liquid prod	Solid prod
t/month	9249371.86		9249371.86	9249371.86	
t/month	39.77		39.77		39.77
t/month	993.09		993.09	993.09	
t/month	0.00		0.00	0.00	
t/month		369.97	369.97	344.98	25.00
t/month		78.62	78.62	78.62	
t/month	20597.93		20597.93	20549.08	
t/month					5.68
t/month					3.41
t/month					11.36
t/month					28.40
t/month	9271002.65	448.59	9271451.24	9271337.63	113.61

Calcining

Assumptions:

Total liquid feed	9271337.63 t/month
Feed	39.77 t/month Au
	25.00 t/month Zn
	5.68 t/month Ag
	3.41 t/month S
	11.36 t/month Pb
	28.40 t/month Other
H ₂ SO ₄ consumption	1.40 kg (100%)/kg Zn
H ₂ SO ₄ concentration	25.00 %
Decant to	50.00 mass % H ₂ O
Dewater to	10.00 mass % H ₂ O
Zn dissolution	100.00 %
Calcine has no solids loss	
Moisture after calcine	0.00 %

Mass in	9271591	0.00
Mass out	9271591	

s/b

35.00 Au
22.5 Zn
3 Ag
2.5 S
10 Pb

Input Streams

	1	4
Liquor	9271337.63	104.98
Au	39.77	
Zn	25.00	
Ag	5.68	
S	3.41	
Pb	11.36	
Other	28.40	
H ₂ SO ₄		34.99
Total	9271451.24	139.97

Output Streams

	2	7	8	9
Liquor	9271224.01	268.74	9.85	
Au				39.77
Zn				0.00
Ag				5.68
S				3.41
Pb				11.36
Other				28.40
H ₂ SO ₄				
Total	9271224.01	268.74	9.85	88.62

Zn dissolved
100.00
t H₂SO₄/t Zn
1.40

mass % H₂O % H₂SO₄
50.00 25.00
mass % H₂O 10

		1	2	3	4	5	6	7	8	9
		Feed	Decant	To acid treat	Acid add	To dewater	From Dewater	H ₂ O removed	Calcine - g	Calcine s
t/month	Liquor	9271337.63	9271224.01	113.61	104.98	278.58	9.85	268.74	9.85	
t/month	Au	39.77		39.77		39.77	39.77			39.77
t/month	Zn	25.00		25.00		0.00	0.00			0.00
t/month	Ag	5.68		5.68		5.68	5.68			5.68
t/month	S	3.41		3.41		3.41	3.41			3.41
t/month	Pb	11.36		11.36		11.36	11.36			11.36
t/month	Other	28.40		28.40		28.40	28.40			28.40
t/month	H ₂ SO ₄					34.99				
t/month	Total	9271451.24	9271224.01	227.23	139.97	367.20	98.47	268.74	9.85	88.62

return

return

Carbon Adsorption

Assumptions:

Au absorbed max	99.96 %		
Feed w:s	1.15		
Feed - Solid	2402676.31 t/month ore		
	3152.29 t/month Fe		
	0.10 t/month Au		
	2656.25 t/month CaO		
Feed - Liquid	2659088.34 t/month H ₂ O		
	19.41 t/month Au		
	579.54 t/month NaCN		
	12073.75 t/month diss salts		
Au in barren	0.010 g/t		
Au in feed	8 g/t		
Au in conc	4000 g/t		
C loss	0.220 %/day	30 days per month	
	6.60 %/month		
NaCN	100 g/t		
C	20 g/ litre	equiv kg/m ³	
Pulp density	1.45 t/m ³		
Other metals on C	0 %		

Mass in	5180430	0.00	w:s
Mass out	5180430		1.15

Input Streams

	1	2	3	5
Ore	2402676.31			
Water		2659088.34	97996.64	
Fe	3152.29			
Au	0.098	19.4100		
CaO	2656.25			
NaCN		579.54		
Diss Salts		12073.75		
Carbon				2187.26
Total	2408484.95	2671761.05	97996.64	2187.26

Output Streams

	6	7	8
Ore			2402676.31
Water		2757084.98	
Fe			3152.29
Au	19.4022	0.0078	0.098
CaO			2656.25
NaCN		579.54	
Diss Salts	12.07	12061.68	
Carbon			2187.26
Total	31.48	2769726.21	2410672.21

	1	2	3	4	5	6	7	8
	Feed - s	Feed - l	H ₂ O add	Liquid	C make-up	Carbon phase	Liquid	Solid
t/month Ore	2402676.31							2402676.31
t/month Water		2659088.34	97996.64	2757084.98			2757084.98	
t/month Fe	3152.29							3152.29
t/month Au	0.098	19.40996357		19.4100		19.4022	0.0078	0.098
t/month CaO	2656.25							2656.25
t/month NaCN		579.54		579.54			579.54	
t/month Diss Salts		12073.75		12073.75		12.07	12061.68	
t/month Carbon					2187.26	33140.36		2187.26
Total	2408484.95	2671761.05	97996.64	2769757.69	2187.26	33171.84	2769726.21	2410672.21

Elution - AARL

Assumptions:

Feed - Carbon Pha	19.40 t/month Au
	12.07 t/month diss salts
	33140.36 load C
Water required	103.30 m ³ /ton C
NaCN	18.30 kg/ton
HCL	52.80 kg/ton
NaOH	42.20 kg/ton

Stripping efficiency	98 %
Regeneration requi	1 kg steam/kg C
Adsorbion of salts	50 %

Mass in	3493466	0.00
Mass out	3493466	

Input streams

	1	3	4	5	6	8
Water	0	0	3423399	0	0	
Au	19					
Diss salts	12					
Carbon	33140					
HCl		1750				
NaCN				606		
NaOH				1399		
Steam						33140
Total	33172	1750	3423399	2005	0	33140

Output Streams

	2	3	4	5	7	8	9
Water	0	0	3423399	0	0		
Au					19		0
Diss salts					6		6
Carbon							33140
HCl		1750					
NaCN				606			
NaOH				1399			
Efficiency							33140
Steam kg/k							33140
98	0	1750	3423399	2005	25	33147	33140

	1	2	3	4	5	6	7	8	9
	Feed	Drain	Acid Wash	Water wash	NaCN/NaO	Soft water	Product	C reactivat	C return
t/month	Water	0.00	0.00	3423399.486	0.00	0.00	0.00		
t/month	Au	19.40					19.01	0.388	
t/month	Diss salts	12.07					6.04	6.04	
t/month	Carbon	33140.36							33140.36
t/month	HCl		1749.81						
t/month	NaCN				606.47				
t/month	NaOH				1398.52				
t/month	Steam						33140.36		
t/month	Total	33171.84	0.00	1749.81	3423399.486	2004.99	0	25.05	33140.36

Electrowinning

Assumptions:

Feed	12.36 t/month Au				196.97 Au
	3.92 t/month diss salts				55.85 Fe
	0.00 t/month liquid				
Efficiency	99.95 %	Power Consumed			
Amperes	650 A	2600 W			
Volts	4 V	112.32 MJ for		30 days	
For	150 l/min	30 days a month			
equivalent to	6480 t/month				
Cathodes	6 weighing	1000 g			
Anodes	7				
Au uses	3 electron				
Fe uses	2 electrons				
For 1 ton Au	0.19 tons Fe are used				
Diss salts won	50 %	Diss salts			
Faraday's Constant	96485.13432 C/mol charge	50			
e-s tranfered	1 e- per mole r metal	ton Fe/ton Au			
Mass in	18.62	0.00		0.19	
Mass out	18.62		Efficiency		

Masses

Input streams

	1	2
Liquid	0.00	
Au	12.36	
Diss salts	3.92	
Fe		2.34
Total	16.28	2.34

Output Streams

	3	4
Liquid		0.00
Au	12.35	0.0062
Diss salts	1.96	1.9620
Fe		2.34
Total	14.32	4.30

	1	2	3	4
	Feed	Fe make-up	Product	Tails
t/month	Liquid	0.00		0.00
t/month	Au	12.36	12.35	0.0062
t/month	Diss metals	3.92	1.96	1.96
t/month	Fe		2.34	2.34
	Total	16.28	2.34	14.32
Energy				
moles			80905.11116	

Cathode smelting

Assumptions:

Feed	12.35 t/month Au			
	1.96 t/month diss salts			
Product	86 % Au		% Base Met	
	9 % Ag			4
	4 % Base metals		% Ag	
Gold lost to atmos	0 %			9
Mass in	14.32	0.00	% Au	
Mass out	14.32			86
		1	2	3
	Feed		Gases	Product
t/month	Au	12.35	0.00	12.35
t/month	Diss salts	1.96	0.09	
t/month	Ag			1.293
	Base metals			0.575
	Total	14.32	0.09	14.36

WASTE

Input streams

	1	
Au	12.35	
Diss salts	1.96	
Ag		
Base metals		
Total	14.32	

Output Streams

	2	3
Au	0.00	12.35
Diss salts	0.09	
Ag		1.293
Base metals		0.575
Total	0.09	14.22

Calcine Smelting

Assumptions:

Feed	39.77 t/month Au
	0.00 t/month Zn
	5.68 t/month Ag
	3.41 t/month S
	11.36 t/month Pb
	28.40 t/month Other
Product	86 % Au
	10 % Ag
	4 % Base metals
Gold lost to a	0 %

Mass in	88.62	0.00
Mass out	88.62	

% Base metals
4.00

% Ag
10.00

% Au
0 86

		1 Feed	2 Heat - gas	3 Product
t/month	Au	39.77	0.00	39.77
t/month	Zn	0.00		
t/month	Ag	5.68	1.06	4.62
t/month	S	3.41		
t/month	Pb	11.36		
t/month	Other	28.40	41.32	1.850
t/month	Total	88.62	42.38	46.24

Input Streams

	1
Au	39.77
Zn	0.00
Ag	5.68
S	3.41
Pb	11.36
Other	28.40
Total	88.62

Output Streams

	2	3
Au	0.00	39.77
Zn		
Ag	1.06	4.62
S		
Pb		
Other	41.32	1.850
Total	42.38	46.24

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Appendix 2-4: Sub-sectoral Study on Coal

2.4.1 Introduction

More than seventy collieries operate within South Africa, mining three distinct geological occurrences. Most of the coal produced in South Africa is bituminous, although a small percentage of anthracite is also mined. In 1993 South Africa ranked sixth in world coal production. The coal reserves still to be exploited are estimated to be 55 000 Mt or 10% of world reserves.

In 1996 South Africa's coal industry produced 203 Mt of coal, of this 60 Mt were exported. Anthracitic coal represents about 1% of the country's total coal production.

South African coal is characteristically low in sulphur though most coal requires washing to reach the required export value of approximately 1% sulphur. The ash content of the coal is high. Coal buyers dictate the required proportion of ash and sulphur which directly effects the amount of beneficiation undertaken at the processing plant. Thus the amount of waste produced by the process is dependent on market demand. The lower the sulphur and ash levels demanded by the market, the more the coal will have to be washed prior to selling, and the greater the amount of discard produced.

A large amount of duff (or waste) coal, usually extremely fine, has been stock-piled. Some of it is burning (spontaneously) and the remainder lies fallow. The quality and quantity of this coal have been established in various reports on the subject and a lot of effort has gone into evaluating the possibility of reclaiming this material. Technological advancements in dealing with fine material are needed to make reuse of this coal possible. From a clean production point of view these duff coal reserves will have to be exploited at some point as they represent a vast source of raw materials already mined.

2.4.2 Generalised flow diagram

Figure A2.4-1, the generalised flowsheet modelled for coal, details five levels of coal cleaning. The end use of the coal dictates the level of cleaning required. For this reason the complete flow of coal in South Africa is included in Figure A2.4-2.

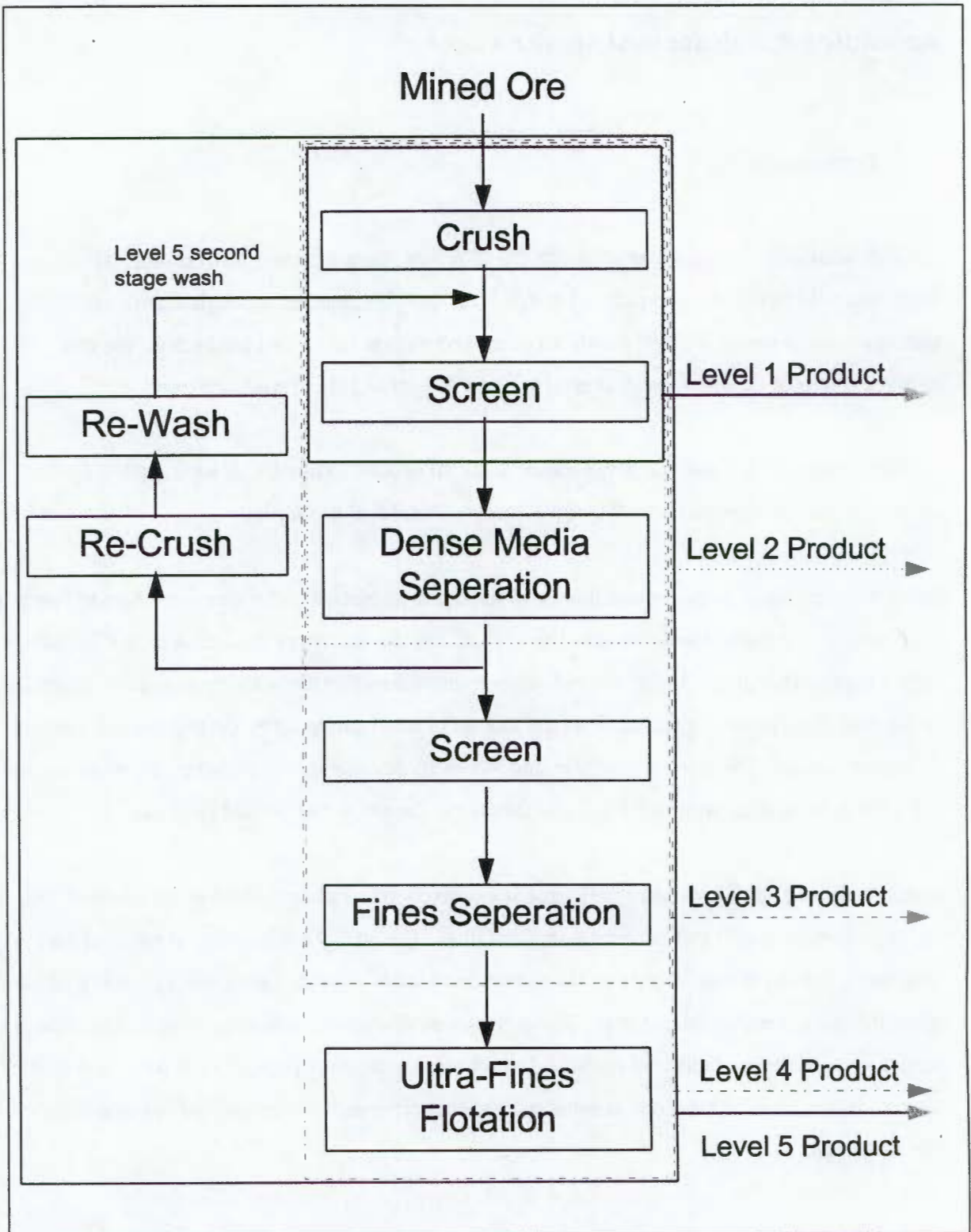


Figure A2.4-1: Generalised Flow Diagram for the South African Coal Industry

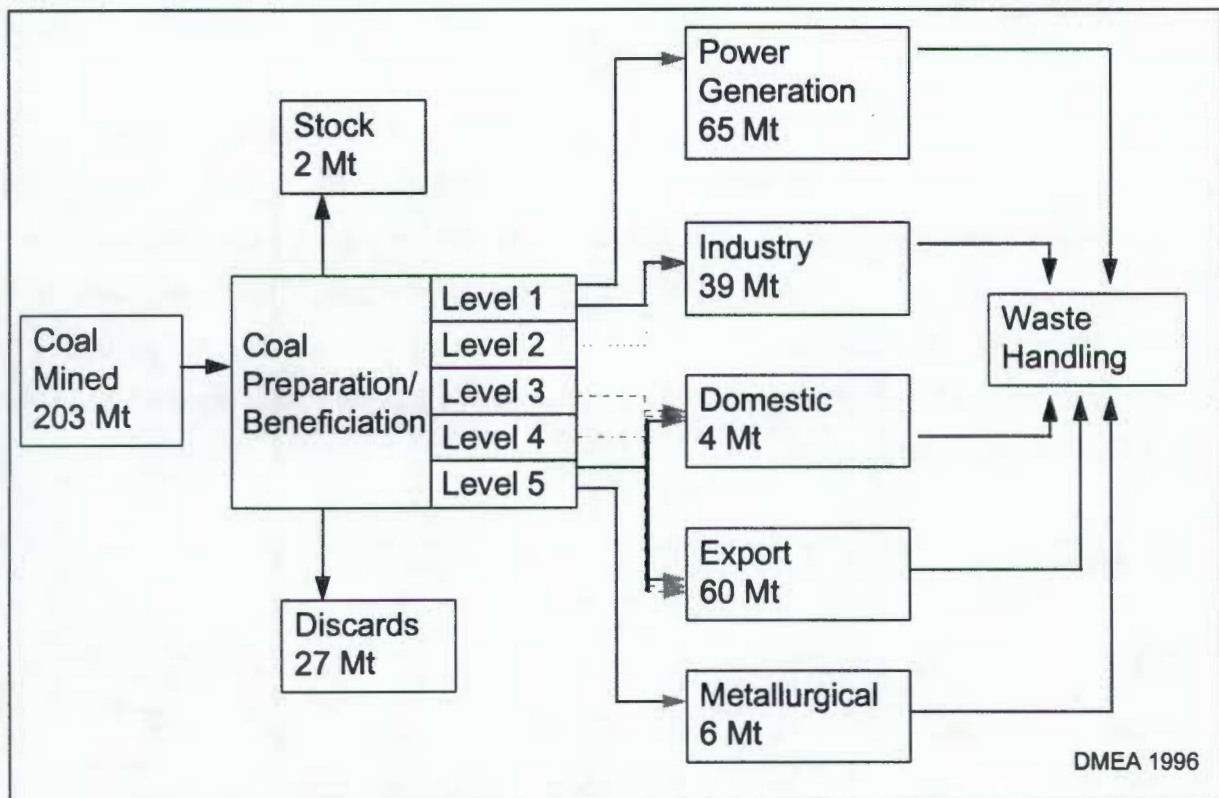


Figure A2.4-2: South African Coal Life Cycle

From this pair of diagrams one can see that products from Level 1 processes, essentially crush and screen procedures, report solely to the power generation sector, Eskom. Most of the coal used by industry, a Level 2 product with a single washing stage, is consumed by Sasol in the production of oil from coal. Metallurgical coal, usually a Level 5 product coal which has undergone the most washing, is used principally in the manufacture of Ferro-alloy products.

Scrutiny of these diagrams discloses firstly that coal is the basis of the power sector in South Africa, providing both electricity and petrol. Life Cycle Assessment (LCA) includes the environmental effects of all services such as electricity and transport. Thus the LCA of coal will form part of the data base feeding into all LCAs in South Africa.

Secondly, coal beneficiation is purely physical, and linear changes in unit operations dictated by market requirements will cause linear changes in the waste inventory for the process. The environmental impacts associated with the beneficiation of coal are thus difficult to minimise as they are dictated by market demand. They are also small in comparison to the environmental impacts associated with coal use. It is for these reasons that coal will be used to illustrate how the methodology developed fits into the LCA framework.

2.4.3 Waste Inventory

The complete waste inventory for the South African coal industry is presented here. The most important assumptions made in modelling the process were those determining what percentage of the stream reached which level of beneficiation, i.e., the stream splits between the unit operations. The three sources of these splits are listed on Table A2.4-1 below. Percentage set 1988b was chosen to model the process as it rendered a product split closest to what was expected. This was checked primarily by cross-referencing the amount of material produced in Level 1 processes with the amount of coal used by Eskom in that year.

Percentage Set	Level					Information Source
	1	2	3	4	5	
1988a	46%	7%	26%	15%	5%	DEA Duff Coal - 1988
1988b	35%	26%	15%	19%	6%	DEA Operating Coal mines - 1988
1984	46%	11%	30%	5%	8%	Reported values - 1984

Table A2.4-1: Percentage Splits over the five Coal processing levels

2.4.3.1 Predicted Values

203 Mt of coal was sold in 1996 and 2 Mt was stockpiled. Processing this coal generated a total waste stream of 54 Mt. Waste coal constituted nearly 90% of this stream.

There are three main constituents in the total waste stream detailed in Table A2.4-2 below.

Ascertaining the dissolution of minerals into the water in the stream would require the execution of a complex speciation. These figures, however, give an indication of the amount of water affected.

Coal (kt/annum)	Water (kt/annum)	Magnetite (kt/annum)
48 000	5 700	45

Table A2.4-2: Constituents in the waste stream from the South African coal industry

2.4.3.2 Comparison with other information

The values for the amount of waste generated by the coal industry as quoted the CSIR and DEA reports are aggregated in Table A2.4-3.

These numbers are comparable with the results obtained from the model of the industry. The main difference can be accounted for by the omission of the amount of water in the waste stream. These results also substantiate the split of the waste over the various media.

Category	Waste Stream (Mt/annum)	Hazardous Waste Groups			Generation To		
		Haz Waste (Grp 1-3)	Potential Haz (Grp 4)	Non- Haz (Grp 5)	Air	Water	Solid/ Liquid
Spoil	34.20						
Slurry	11.40						
TOTAL	45.60	0	34.20	11.40	0	0	4.,60

Table A2.4-3: Waste Stream generated by the South African coal mining industry
(References: FRD and CSIR)

2.4.3.3 Discussion

The values calculated for the waste stream generated by the coal industry compare favourably with those found in literature. The assumptions used in calculating these numbers are general because the changing demands of the purchasers result in changes in the waste generated by a process. Process performance also changes on a daily basis. However, on a macro-scale at least, the results are satisfactory.

Confidence levels

Flow Sheet	95%
Assumptions	70%
Stream Splits	85%

Confidence in the generalised flowsheet is high as it was verified by many people in industry. Confidence in the assumption values is difficult to quantify. The unit operations, being purely physical separation methods, are easy to model. As such cyclones, jigs and other separation methods which depend on density differences can be modelled in the same unit using aggregated assumptions. On the other hand, these values can fluctuate widely in the course of a year. Confidence placed in the stream splits is high as the product split was confirmed by existing information.

2.4.4 Using the model - Application of LCA to Coal-Fired Power Generation

ESKOM, the South African parastatal power generation company, consumed 32% of total coal production in 1996 (Figure A2.4-2) in generating electricity for the country. Over 90% of electricity generated in South Africa is produced in coal-based plants. The coal is milled and injected into furnaces where it is burnt. Water is heated and used to drive turbines which generate the electricity. There are electro-static precipitators installed on the plants, these remove particulates from the off-gases. These particulates have little economic value and are disposed of to landfill site. There is a potential for these sites to leach metals into the groundwater system.

There is little removal of the SO_x products of combustion. In spite of the fact that South African coal has a low sulphur content, there are potential environmental impacts associated with this SO_x production and efforts to remove Sulphur from the energy production cycle is necessary.

There are various technologies available for the removal of sulphur, they can be divided into three discrete groups:

- Pre-combustion removal of the sulphur, e.g., additional beneficiation of the coal to remove sulphur at the mine. Referring to Figure A2.4-1, this would imply that more material would report to stage 2 beneficiation. Alternatively, microbial washing could be used. Using this approach would result in additional waste coal production at the processing plant. There is also the potential to lose energy in the coal to the waste stream. There is the advantage that washing reduces the ash content of the coal and would decrease the ash produced at the plant.
- It is possible to remove sulphur from the process during combustion. This is achieved by injecting sorbents into the combustion chamber. It is also possible to run the furnace as a fluidised bed roaster. This has the advantage of increasing efficiency. However, there is the production of waste material carrying the sulphur for which there is no definite market at present.

- Post-combustion removal of SO_x from the off-gases, these technologies are well developed and widely utilised. The off-gases are scrubbed with water producing a dilute sulphuric acid stream. It is possible to concentrate this stream and sell it but, as with the gold industry, this is seldom economically viable. Retro-fitting of these technologies also results in decreased efficiency.

In the work of Notten and Diekmann (1996), the following technology options were investigated:

- Dense medium washing of the coal fed to the pulverised fuel boiler,
- Microbial desulphurisation of the coal fed to the pulverised fuel boiler,
- Feeding run-of-mine coal to a fluidized bed boiler,
- Flue gas desulphurisation of the flue gases exiting the pulverised fuel boiler,
- Feeding run-of-mine coal to a pulverised fuel boiler (standard process).

An LCA profile for each of these technology suites was developed using the LCA software package PEMS. Within the study the boundary for the LCA was drawn around the flow of coal from mining and beneficiation to power generation. It was assumed that the use of the electricity would not change with changing production technologies, and thus the environmental impacts of downstream use of electricity was not included in the LCA boundary as they are independent of technology choice. This is the concept of a **background** system (that which is not affected by the technology under investigation) and the **foreground** system (that which changes within the assessment) as defined by Clift (1997). Figure A2.4-3 is included to clarify this concept.

An inventory analysis (the second stage of the LCA, comprising a mass balance for the system) was conducted over the foreground system identified in Figure A2.4-3, this was a combination of the beneficiation model detailed in Figure A2.4-1 and a simplified combustion model. The assumption set for the beneficiation model was changed according to the potential technology used (i.e., to reflect the performance of the process with increased washing of the coal). Figure A2.4-4 contains details of the inventories calculated.

The impact assessment is then evaluated from the inventory. Here inputs and outputs are classified and aggregated according to their contribution to the environmental impact categories identified in section 1.3. The impacts are normalised for the sake of ease of comparison. In Figure A2.4-5, the effects have been normalised with respect to annual global production of the respective impacts. In this figure the impact characterisation data has been taken from Heijungs et al.(1992). This normalisation is consistent with a global view of environmental impact. However, in order to make

the study more meaningful to the South African context, it would be preferable to normalise these effects with respect to national impact values. Such an impact data-base has still to be developed.

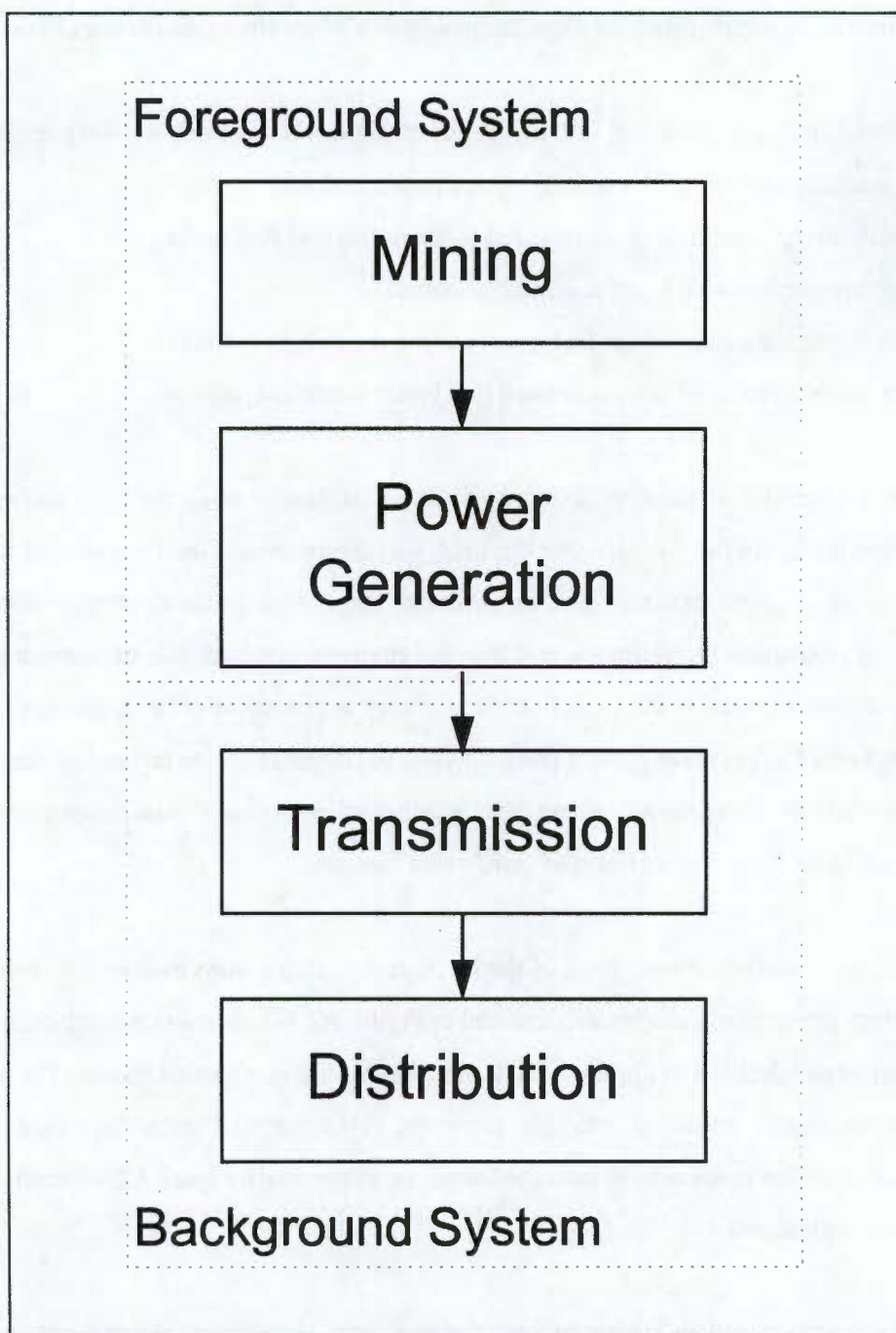


Figure A2.4-3: Differentiation between background and foreground system for the Power Generation Life Cycle

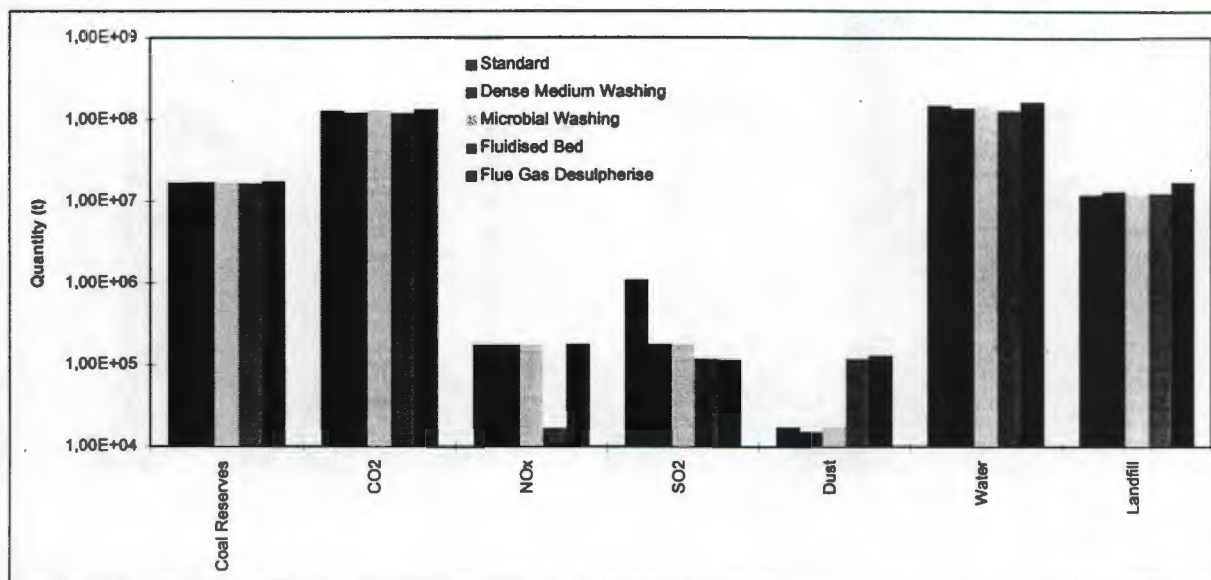


Figure A2.4-4: Life Cycle Inventory for various technology suites

In this figure all the impact categories have been weighted evenly. Weighting of impact categories is extremely subjective and affects the results of the LCA markedly. The manipulation of these weightings will be discussed in the next section of this thesis. For clarity and convenience at this point, equal weightings have been assigned to the impact categories.

This figure shows the difference in environmental performance between the various technology suites identified. From these results it could be inferred that fluidised bed combustion with combined removal of sulphur is the best environmental option. However, weightings which reflect the South African context would give a different result.

This case study illustrates the concept of linking the process models to an LCA. The dynamic nature of the models is should be highlighted at this point as it is possible to show improvement in environmental performance with changing operating conditions.

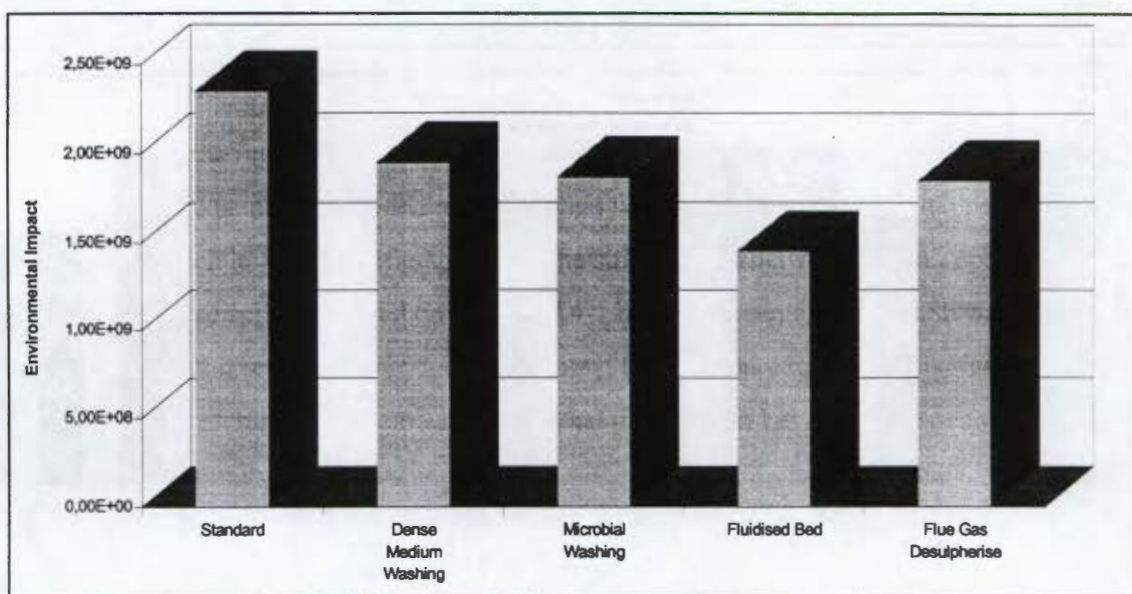


Figure A2.4-5: Valuation graph for the Coal Life Cycle

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Appendix 2-5: Sub-sectoral study on the Platinum Group Metals industry

2.5.1 Introduction

The platinum group metals (PGMs) include six elements - platinum, palladium, rhodium, ruthenium, iridium and osmium. These are mainly used in jewellery manufacture and in catalytic converters for the automobile industry.

During 1996 the platinum group industry, within the sector ranking second to the gold industry in foreign exchange generation, produced 188 tonnes of PGMs and sold products with a value of R 7580 million. By far the greatest known in the world, South African platinum reserves are estimated to be 58 800 tonnes. Four major companies are involved in the processing.

The ore processed is drawn from three main bodies - the Platreef, UG-2 and Merensky reefs. The latter two ore bodies differ from the former with respect to the presence of iron and chrome as co-deposited metals.

The industry produces several base metals as by-products. The most significant are copper, cobalt and nickel. The production of these base metals has been considered here (rather than in the base metals section), as the processes which produce them are integral to the entire PGM production operation. However, flowsheets developed are similar to those used in the base metals industry.

2.5.2 Generalised flow diagram

The flowsheet modelled for the PGM sub-sector is included in Figure A2.5-1 below. The process has three main sections. The plants are often physically separated so they have been kept as discrete sections within the models. The precious metals refinery is located far from the other two sections. Usually the products from more than one base metals refinery are processed at a single precious metals refinery.

The first stage of refining is the initial beneficiation stage where primary concentration occurs. The two main products from this part of the process are a PGM-rich stream which is sent directly to the precious metals refinery, and a smelter matte which is directed to the base metals refinery. The initial

beneficiation process is detailed in Figure A2.5-2. Essentially separation at the concentrator depends on the physical nature of the material.

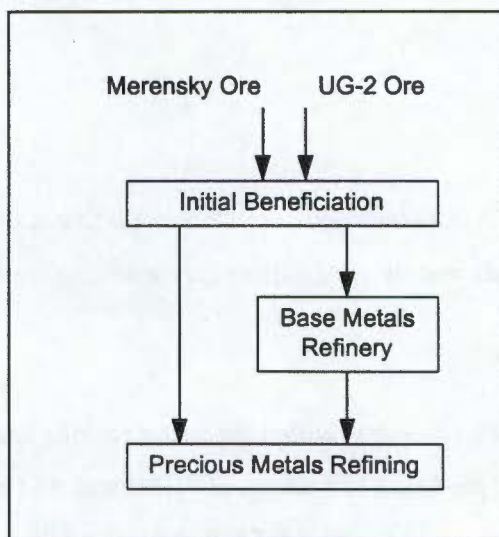


Figure A2.5-1: Generic overall Flow Diagram for Platinum Production

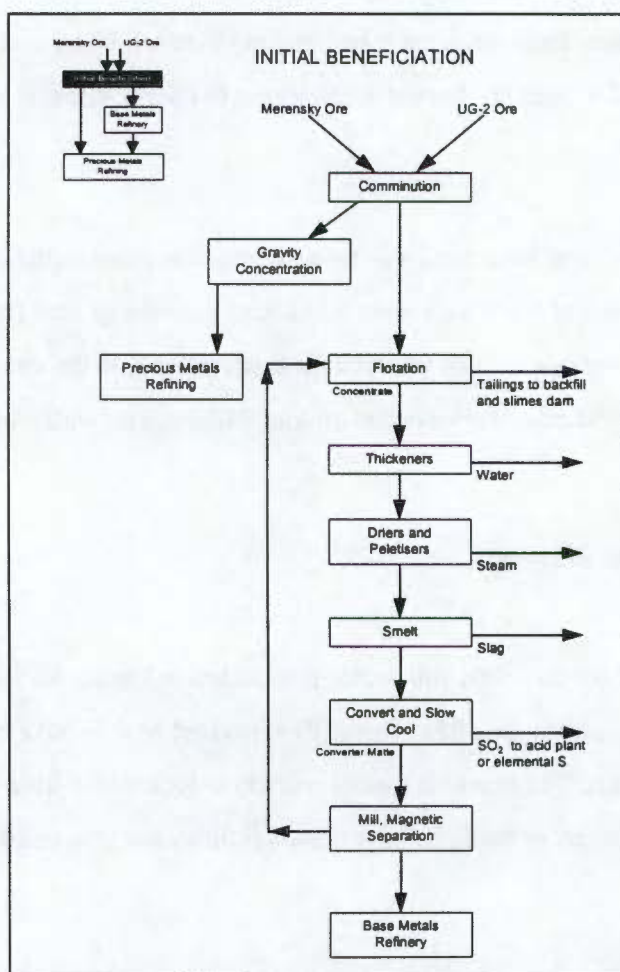


Figure A2.5-2: Initial Beneficiation in the Platinum Industry

The final stage of PGM processing, the precious metals refinery, has not been modelled in this work. There were two main reasons motivating this decision:

- Although environmentally hazardous streams leave the system, these streams are small.
- As PGMs are valuable, the preferred practice is to retain them within the process in recycle streams rather than to discharge them to the environment.

The other reason for omitting the precious metals refinery from the model is lack of information about the processes themselves. The efficiencies of the processes are of tantamount importance to the producers of PGMs. (The greater the efficiency of units is the smaller will be the recycle, and the less the valuable material locked up in the plant's recycle inventory). Companies are therefore loathe to divulge information about processes or operating procedures for fear of losing competitive advantage in the industry.

2.5.3 Waste Inventory

Initial concentration made a significant contribution to the 81 Mt of waste produced by the industry in 1996. Details of the total waste stream are presented below.

2.5.3.1 Predicted Values

Table A2.5-1 gives details of the waste stream by component for the two processes detailed. Totals have been rounded.

This table shows that water is a major constituent of the waste stream from the industry, as was the case for gold processing. Although several added reagents also find their way into the waste water stream, most other major contributors are metals found in the virgin ore. There is also a significant environmental impact associated with the iron and the sulphur in the waste from the base metals refinery.

Table A2.5-2 gives a break-down of the units generating this waste stream. Units from the two processes have been combined. Only waste-generating units were included. It is not surprising that the major contributors to the waste stream are the units in the initial beneficiation stage of the process. As unit operations within plants processing these ores behave in fundamentally different

ways, the UG-2 and Merensky ores were dealt with separately in the model, by changing stream splits and unit operation assumption sets.

Component	Initial Concentration (kt/annum)	Base Metals Refinery (kt/annum)	Totals (kt/annum)
H ₂ O	47 000	62	47 062
Merensky	15 000		15 000
UG-2	13 000		13 000
Cr ₂ O ₃	5 700		5 700
Fe	270	0.17	270
Air	59		59
SiO ₂	42		42
O ₂	24	18	42
S	24	6	30
Various Reagents	30	0.09	30
H ₂ SO ₄		19	19
MgO	16		16
CaO	13		13
Ni	12	1	13
Cu	12	1	13
Cr	6		6
Co		0.19	0.19
PGM	0.03		0.03
Total	81 200	107	81 307

Table A2.5-1: Waste generated by the Initial Concentrator

Unit Operation	Waste (kt/annum)	Generation To		
		Air	Water	Solid/ Liquid
Comminution	43 000			
Gravity Concentration	19 000			
Merensky Floatation	17 000			
UG-2 Floatation	1 200			
Thickener	710			
Dry and Pelletise	100			
Merensky Smelter	84			
UG-2 Smelter	81			
Converter	61			
Second Stage Leach	50			
Copper Electrowinning	16			
Evaporator	3.8			
(NH ₄) ₂ SO ₄ Prod	2.4			
Cobalt Removal	2.3			
Nickel Electrowinning	1.4			
Sulphur Removal	0.02			
Total	81 300	230	80	81 000

Table A2.5-2: Waste generation according to Unit Operation

2.5.3.2 Comparison with other information

The results from the FRD/CSIR study are tabulated on Table A2.5-3.

Category	Waste Stream (kt/annum)	Hazardous Waste Groups			Generation To		
		Haz Waste (Grp 1-3)	Potential Haz (Grp 4)	Non- Haz (Grp 5)	Air	Water	Solid/ Liquid
Initial Concentration							
Tailings	35000						
Ni & Cu							
Production:							
Furnace slag	1035959						
Acid Bleed	0.87						
Air emission	27						
Base metals							
Refinery							
Ni, Cu, Co							
Refining:							
Acid Effluent	0.62						
Alkali Effluent	0.09						
Boiler Effluent	0.21						
Process Water	13.32						
Surface Runoff	2.70						
Solid Waste	0.91						
Tailings	9100						
TOTAL	45182	18	28	45136	27	18	45137

Table A2.5-3: Waste Stream generated by the South African platinum producing industry
(References: FRD and CSIR)

The values for the initial concentration stage are compared in Table A2.5-4.

Comparison of the tailings amount from the referenced work and the amount of minerals (i.e., material originating from the ore itself), shows the predicted waste stream to be similar. However,

the FRD/CSIR value for slag is extremely different from that predicted. For the amount of slag to be so high in the model the production of base metals would have to be a great deal smaller.

Alternatively, there would have to be a great deal more feed to the system, or the platinum grade in the ore would have to be a lot lower. As the assumptions made governing these criteria in the model are thought to be accurate, the difference appears inexplicable.

Waste Component	Predicted Amount	CSIR/FRD Amount	CSIR/FRD : Predicted
Tailings:			
Total	810 000	35 000	0.04
Minerals only	34 000		1.02
Slag	160 000	1 035 959	6.50
Acid	Included in Tailings	877	
Air	23 000	27 000	1.20

Table A2.5-4: Comparison of model-predicted amounts with values from literature

2.5.3.3 Discussion

There is confidence in the values generated by the model. The assumptions made are thought to be reasonable. For the required amount of PGMs to be produced at a sensible ore grade and with acceptable process efficiency, the amount of material entering the process is realistic.

Confidence levels - Initial Concentration

Flow Sheet	95%
Assumptions	75%
Stream Splits	75%

The flowsheet for this part of the process is simple so a high level of confidence is placed in its accuracy. The assumptions and stream splits are also reasonable.

Confidence levels - Base Metals Refinery

Flow Sheet	85%
Assumptions	75%

Stream Splits

80%

Because the process was less well-defined in literature less confidence is place in this generalised flowsheet. Available information on some of the smaller unit operations was also scarce. For this reason the assumptions governing these units are called into doubt. The stream splits are fairly well-defined along company boundaries and there are few plants in the sub-sector, so the stream splits are thought to be reasonable.

2.5.4 Using the Model - Comparison of Feed Material

Table A2.5-2 contains details of the waste streams leaving different unit operations within the process. Separating the waste generated by the two floatation and two smelter processes from a waste minimisation point of view separately may be misleading. The total waste generated by the units is a function of their efficiency and of the amount of material entering the process. The differing nature of the ores also affects the nature of the waste stream. Figure A2.5-4 and Figure A2.5-5 are included to clarify the position. These figures reveal which parts of the waste stream can be attributed to the various units on a normalised basis. Values have been normalised with respect to PGMs leaving the unit. This is an elementary calculation as all streams leaving the unit operation are known.

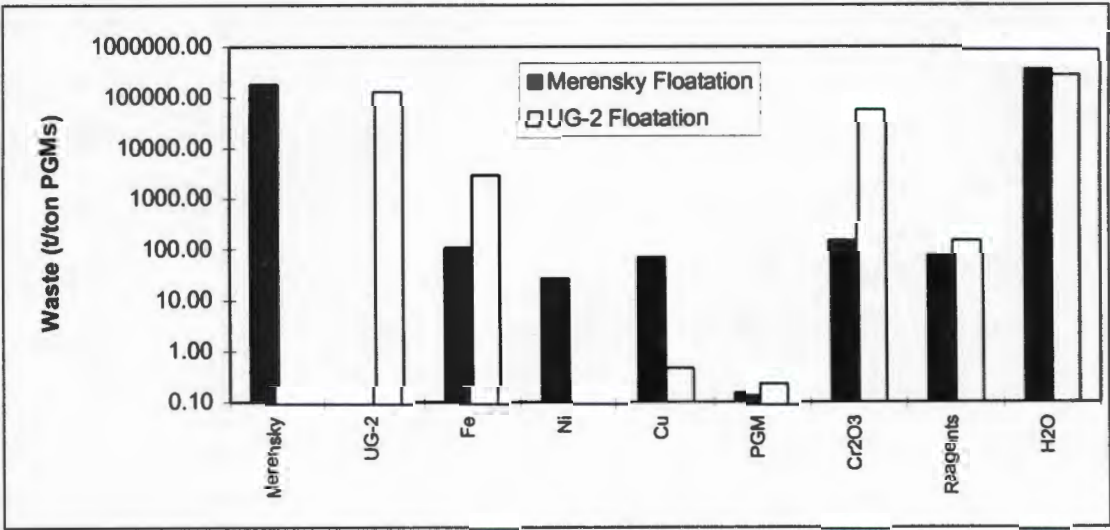


Figure A2.5-4: Comparison of Flotation processes (normalised with respect to PGM content in output stream)

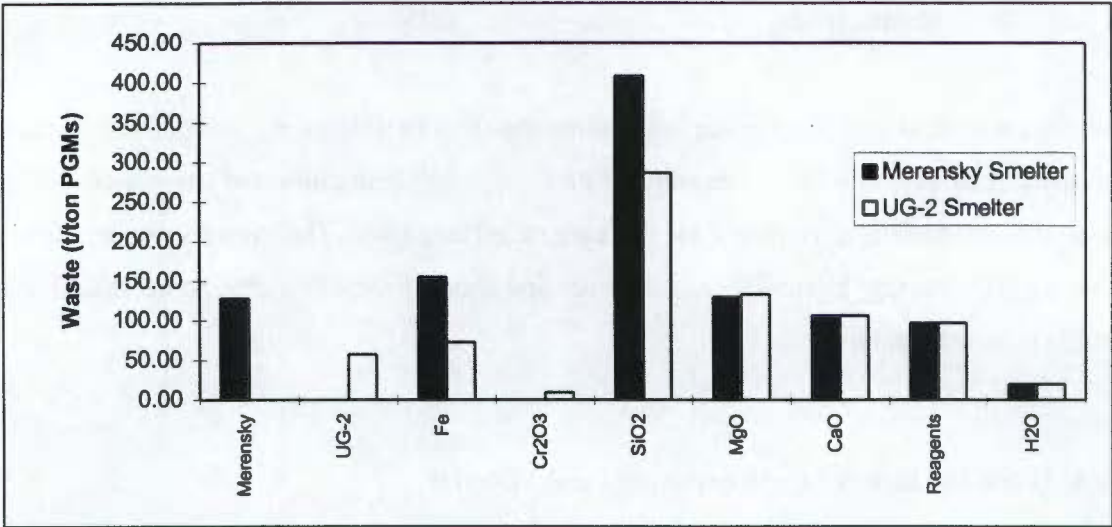


Figure A2.5-5: Comparison of Smelter processes (normalised with respect to PGM content in output stream)

The waste generation of these two processes is similar where water and ore are concerned. Differences are noticeable in the metals leave the processes. These metal contents are a function of the ore processed by the units and of the efficiency of the unit operation. The difference in the normalised amount of PGMs leaving the units is directly related to the efficiency of the units in recovery. There is obviously a disparity in the recoveries experienced with the two types of ores.

Reagent addition differs between these two processes as do the types of metals occurring in each of the two ores. That there is less ore in the waste for the UG-2 smelter would suggest that the processes upstream of these smelters operate more efficiently. These results suggest that, if Cr₂O₃ wastes present an environmental problem, processing of the UG-2 ores should be minimised.

2.5.5 Using the Model - Concentration effects

The three major stages in the platinum group industry are initial concentration, base metals refining and precious metals refining. Figure A2.5-6 diagrams the extent of the differences in bulk stream flow size between the three processes. This concentration of the process stream can, in many ways, remove the uncertainty about the precious metals refinery, and possibly, about the base metals refinery as well. (With respect to mass, the subsequent processes are fairly insignificant compared to the initial concentration stage.)

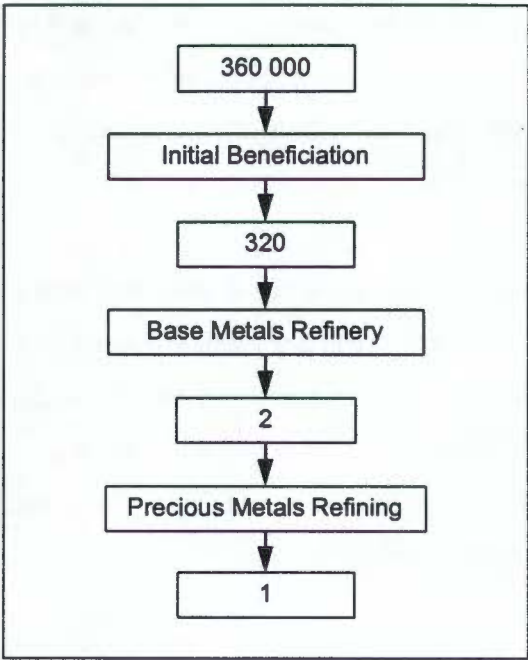


Figure A2.5-6: Concentration of Bulk stream flows in the Platinum Industry

There are, however, other factors to consider. As bulk stream flow decreases the materials become more concentrated. Wastes from these more concentrated stages have a far higher potential to disperse wastes into the environment as pollution. Figure A2.5-7 has been included to demonstrate this concept. Both bulk flow of the process stream (and its inverse effect on concentration of potential pollutants) and dispersion potential (of components in the waste stream) must be considered when evaluating whether a stream is sufficiently important to merit modelling.

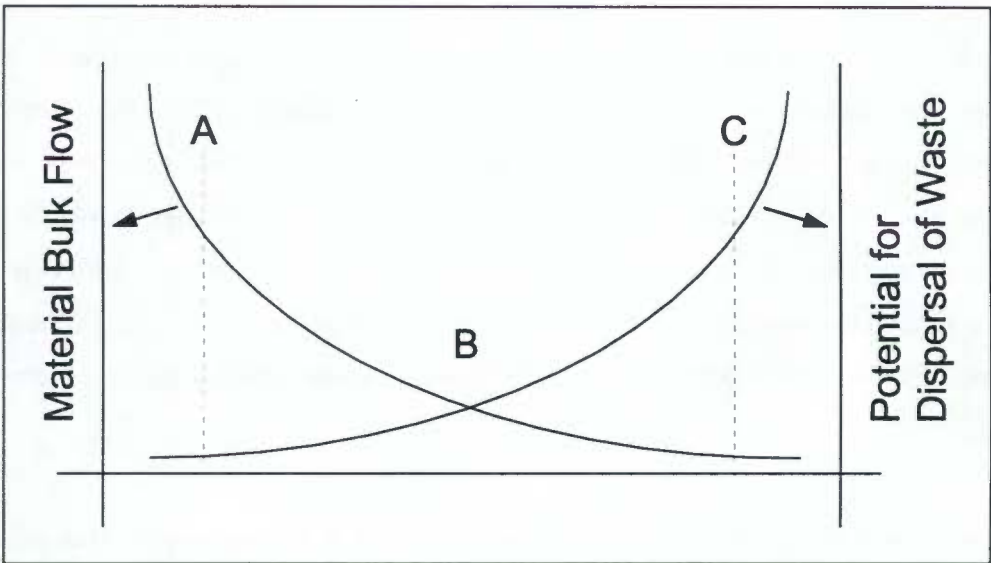


Figure A2.5-7: Bulk flow versus Potential Dispersion from Wastes

At point A on Figure A2.5-7 it would be essential to model the unit operation accurately as the effect of the mass of material is far greater than the potential of the waste to pollute. At point C more attention should be paid to the behaviour of the waste in the natural environment than to accurate modelling of unit operations.

When choosing technologies, those which operate at point B of such a diagram would have the best environmental performance. Different technology suites would have different gradients on the loci drawn on Figure A2.5-7, this too should be considered when choosing technologies. It is proposed that a risk assessment view of environmental impact and waste management regimes would inform this decision. An exhaustive description of such an approach is beyond the scope of this project. The work of Basson (1998) is addressing these issues.

2.5.6 Using the Model - Wastes covered by Legislation

The Environmental Management Programme Report (EMPR) is a legal document prescribed by the South African Department of Mineral and Energy Affairs. The Minerals Act of 1992 requires that all operating mines establish strict environmental management regimes for their processes. The reports are scrutinised by the following government departments :

- Department of Mineral and Energy Affairs
- Department of Water Affairs and Forestry
- Department of Environment and Tourism

Each has to ensure that all possible environmental impacts - air, water, land degradation, noise, etc., are laid out in the report. These impacts must be quantified and the quantifications supported by monitoring regimes. All impacts having been quantified, a financial allocation has to be made to ensure that these can be ameliorated even when the mining company no longer exists. Each government department has to be satisfied that the report includes all impacts subject to its authority, meeting legislated requirements, and that sufficient financial reserves exist to ameliorate the impacts. Only then are the documents approved as a legally binding agreement between the government and the mine.

The various EMPR reports for the sector have been drawn up by a vast spectrum of people across a span of varied specialisations. They range from environmental consultants, and environmental

officers for the various groups, to plant operators. Bias varies in accordance with each contributor's field of expertise so that the final pool of information available is extremely varied.

The reports comprise a rich source of information pertaining to the technology employed and to the impacts of processes. They require the establishment of a full mass balance for each process. This information is useful when such a methodology as that described in this thesis is adopted. However, they are not standardised at all. Adopting an approach to flowsheet development such as the one described in this section would ensure that the information in the reports is of the same standard (i.e., it has been verified to the same level) and comparable (it is presented according to the same criteria or with the same basis). This would improve the value of the data generated as a decision support tool.

Another point is that the legislation covers only "at-mine" processes so, in the case of the platinum group, and ferro-alloys industries, where much of the processing takes place away from the mine, the total impact of the process is not recorded. For the gold mines, where all of the primary processing takes place at the mine, the mass balance evaluations can be used to generate a waste inventory for the process, this is not the case for the platinum industry. There would be more value in standardising the boundaries drawn around processes so that competing sub-sectors can be compared. (Here sub-sectors could be seen as competing for a single energy or water supply.)

These reports also have the failing of attributing all service streams (electricity and water) to the mining processes. Insufficient distinction is made between the water used specifically in mining, water used in the processing plant, and water used in the various support structures. This problem of assigning services used is worse in the case of electricity. The main reason for this attitude is that domestic electricity and water have traditionally been provided free to mine personnel. This aggregation of services can obscure the effect of waste minimisation and clean technology initiatives. It also places no responsibility to improve efficient use of services on domestic users within the mining industry.

The methodology described in this section could be used as the basis for a consistent gathering of information within legislation. The use of this methodology would ensure that the data gathered are uniform and sufficient to support strategic planning by government.

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Appendix 2-6: Sub-sectoral study of the Base Metals Industry

2.6.1 Introduction

South Africa's mineral wealth carries through to the base metals sector. The major base metals produced are aluminium, copper, lead, nickel and zinc. Because it is closely linked to the production of copper, phosphate has been included in this section. Antimony, also produced in one plant, was not included because of the limited size, and thus the limited impact, of the process. Zirconia and various other minerals are by-products of other processes within the sector. Tin is no longer mined in significant quantities. Production levels of the major base metals are listed in Table A2.6-1 below.

Product	Concentrators	Purifiers	Production (kt 1996)	Reserves (Mt)	Rank	Production Rank
Copper	6	5	152	13	13	12
Lead	2		89	3	6	8
Nickel	1		34	12	5	8
Zinc	3	1	77	15	4	8
Phosphate	3		2 700	2 500	3	8
Aluminium		2	570			10

Table A2.6-1: Base Metals Production in South Africa 1996 (DMEA, 1997)

2.6.2 Generalised flow diagrams

Figure A2.6-1 is a generalised flow sheet for the base metals industry. It has been divided into two main sections:

- The initial beneficiation of ore to a metal-in-concentrate (MIC) product
- The downstream purification of this concentrate to a pure metal

Aluminium is not concentrated in South Africa. Raw material for the smelters is imported. The initial concentration stage for the rest of the base metals was modelled as a single process for two reasons. One is the similarity between these processes across the whole industry and the other is protection of company-specific information. At present nickel is not purified in South Africa. On this macro-scale

the interlinking between the various purification stages is not readily apparent but when one studies Figure A2.6-3 through Figure A2.6-6, the links between the purification stages become more obvious. This is especially notable when the relationship between copper and phosphate is examined. The flow diagram for the initial concentration stage can be found on Figure A2.6-2. The four purification process modelled are on Figure A2.6-3 to Figure A2.6-6.

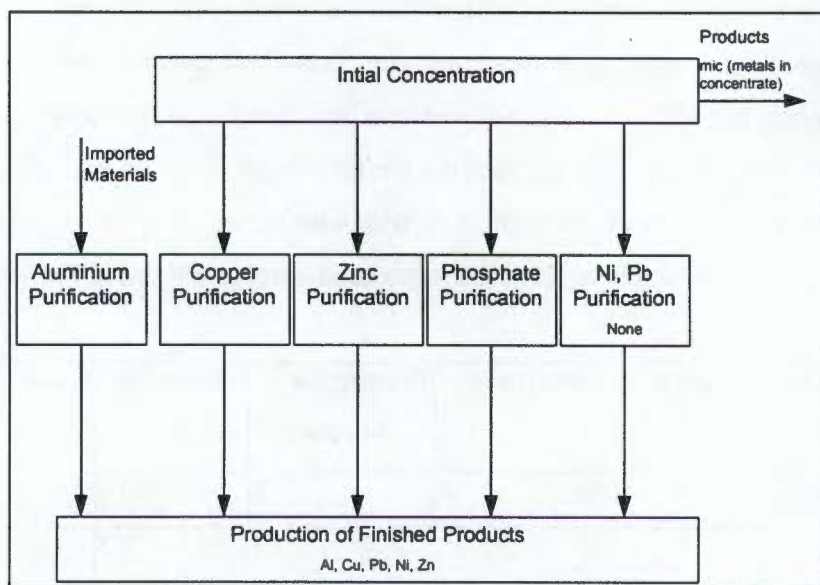


Figure A2.6-1: Overall Flow diagram for the Base Metals Industry

In the initial beneficiation processes Figure A2.6-2 the mineral-rich material within the ore is physically separated. For modelling purposes, in this section all of the material entering the process reported to the milling unit. However, some processes produce copper only, some copper and zinc only - and so on. Care was taken to weight all assumptions governing the unit operations so that they would reflect both the correct product split and accurate as unit by-passes within the flow sheets.

Aluminium production Figure A2.6-3 is, on face value, a fairly simple process. It is however, extremely energy intensive so it has an indirect environmental impact. Also, because this is a smelting operation, air emissions are inevitable. Time constraints make all gaseous emissions difficult to deal with; other wastes can be stored and ameliorated but for gases there is only a short space of time for this it is not possible to contain and deal with later.

In Figure A2.6-4 the interlinking of copper purification with the down-stream phosphate producer is made obvious. Not all of the copper purifiers have this link to a downstream producer. Care was

taken to account for this in the drawing up of the mass balances. Copper MIC is smelted as indicated on the figure. This too was modelled.

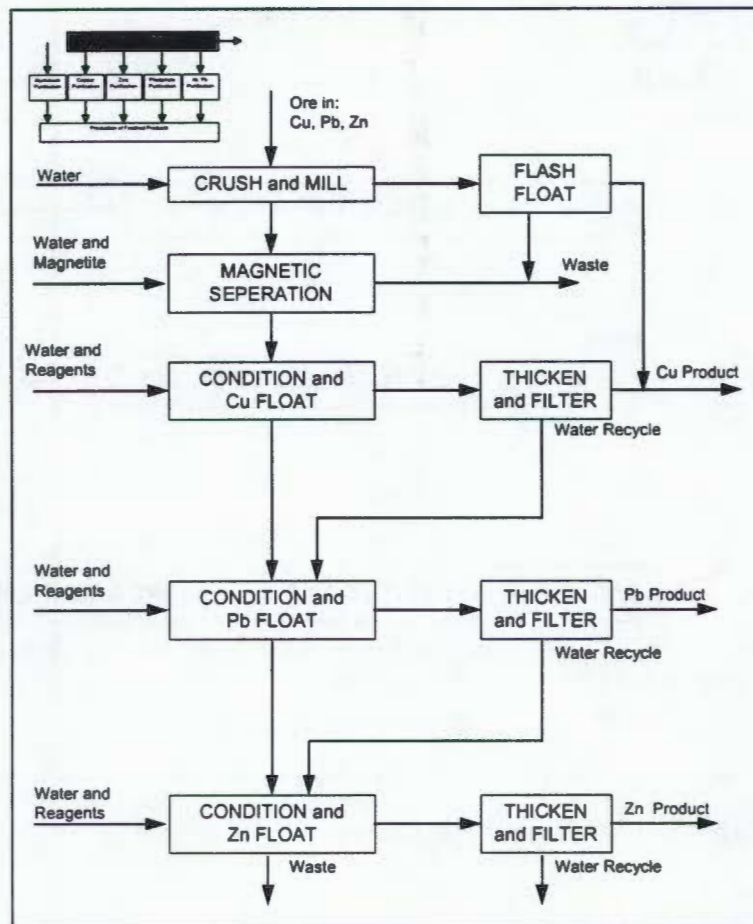


Figure A2.6-2: Flow sheet for Initial Beneficiation in the Base Metals Industry

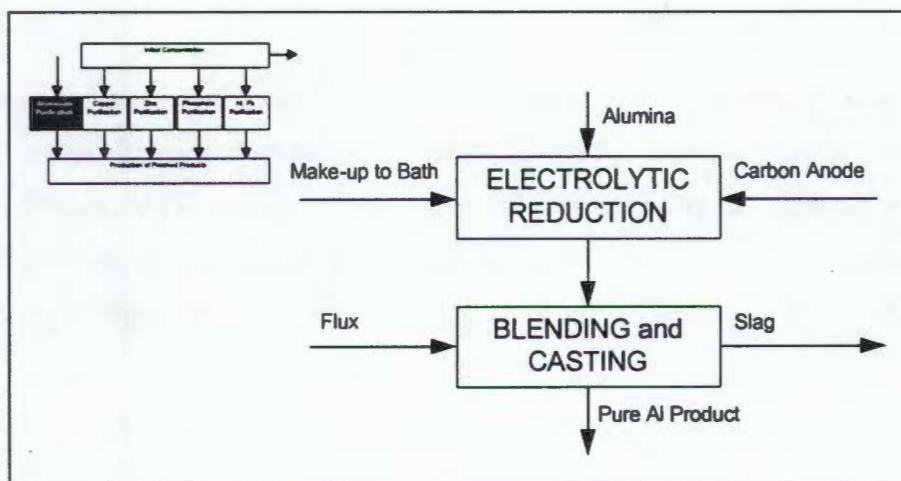


Figure A2.6-3: Flow sheet for Aluminium Purification

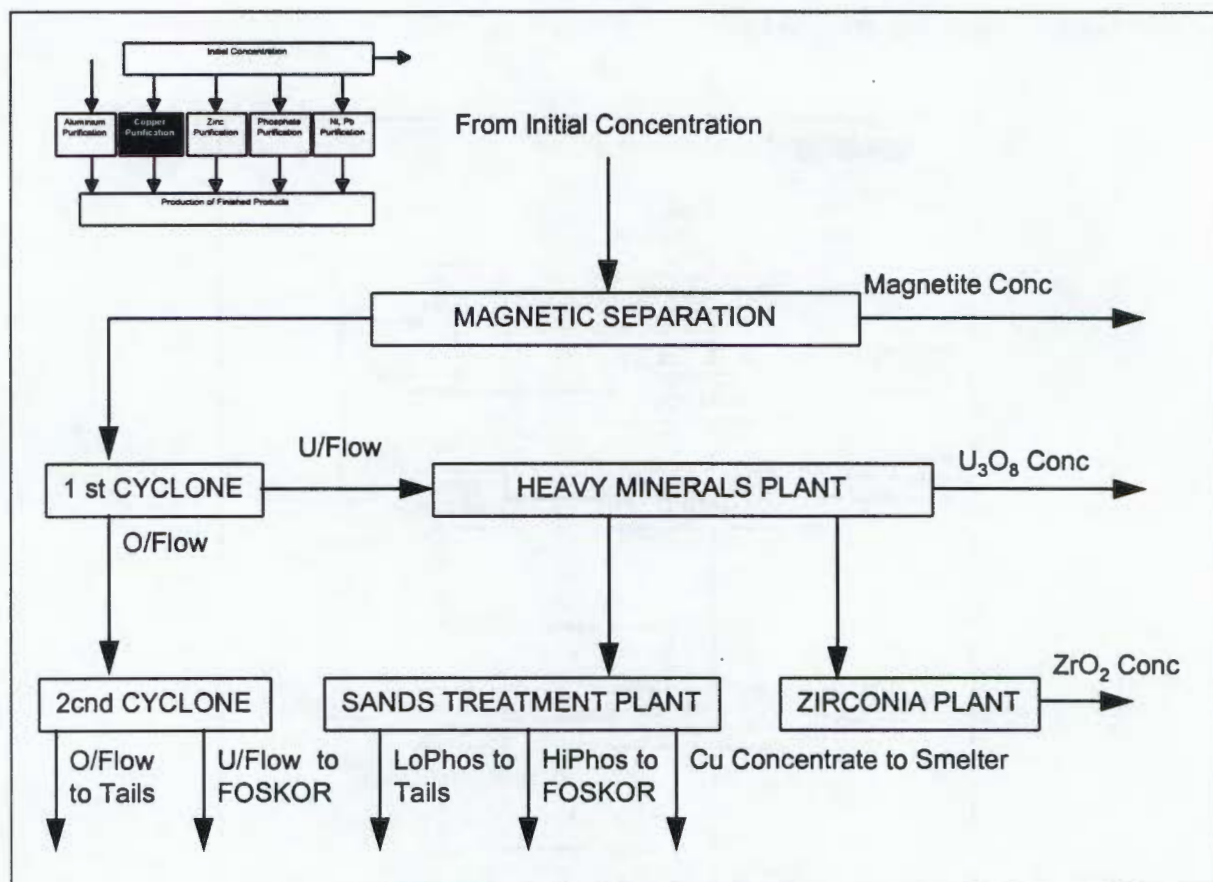


Figure A2.6-4: Flow sheet for Copper Purification

Zinc (Figure A2.6-5) metal in South Africa is produced in a leaching and electrowinning process. Such a process generally has higher impacts on the water system than on gases. The recycles were dealt with carefully for this process so that the correct amount of liquid leaving the system could be evaluated.

As, strictly speaking, phosphate is not a base metal, the initial concentration of the phosphate (Figure A2.6-6) process was not included in the initial concentration of the base metals sector. It was included here, however, because in the purification stages it has a significant link with copper. The initial concentration phase of the phosphate process has been modelled by duplicating the model developed for the initial concentration of base metals and altering the assumptions governing it.

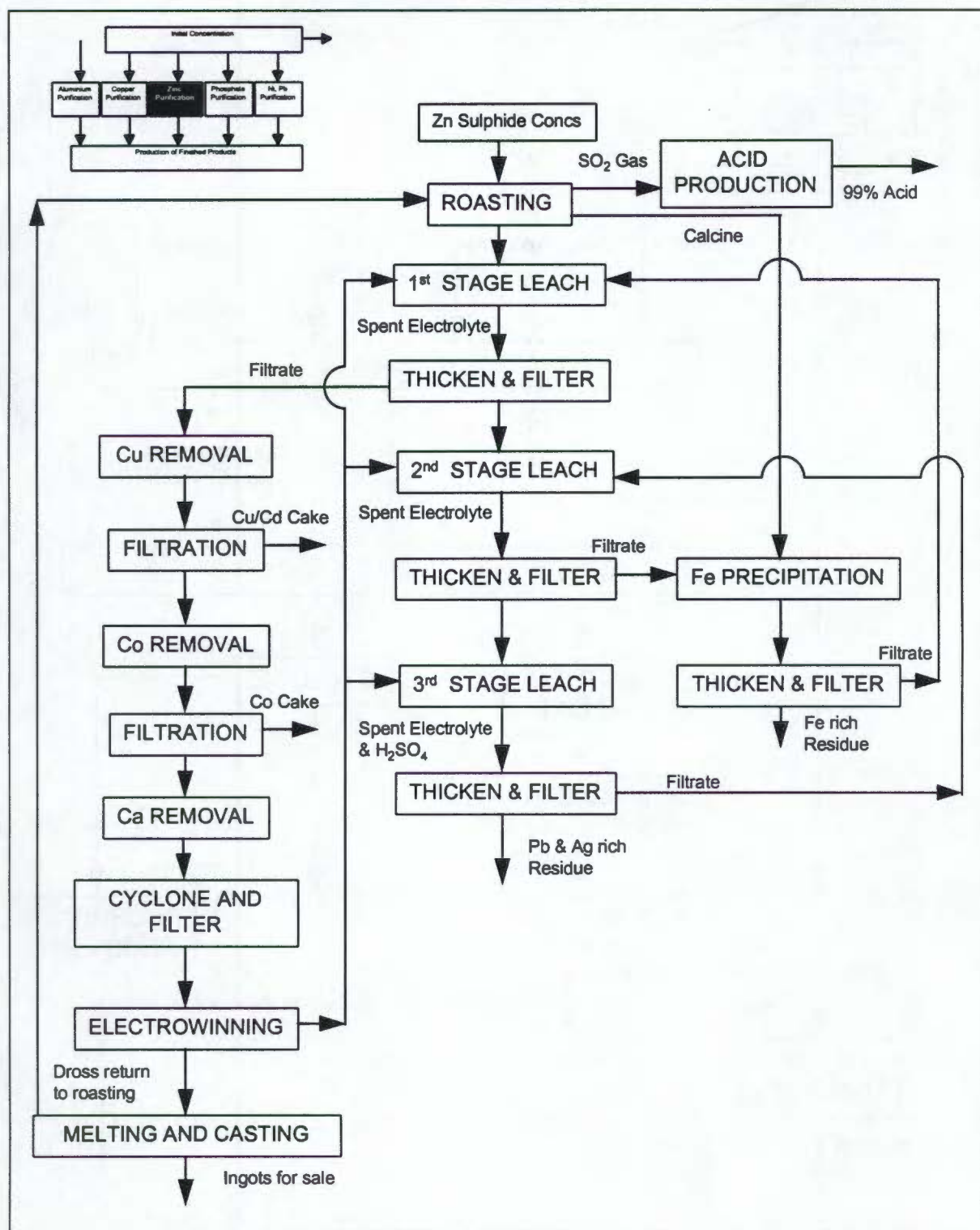


Figure A2.6-5: Flow Sheet for purification of Zinc

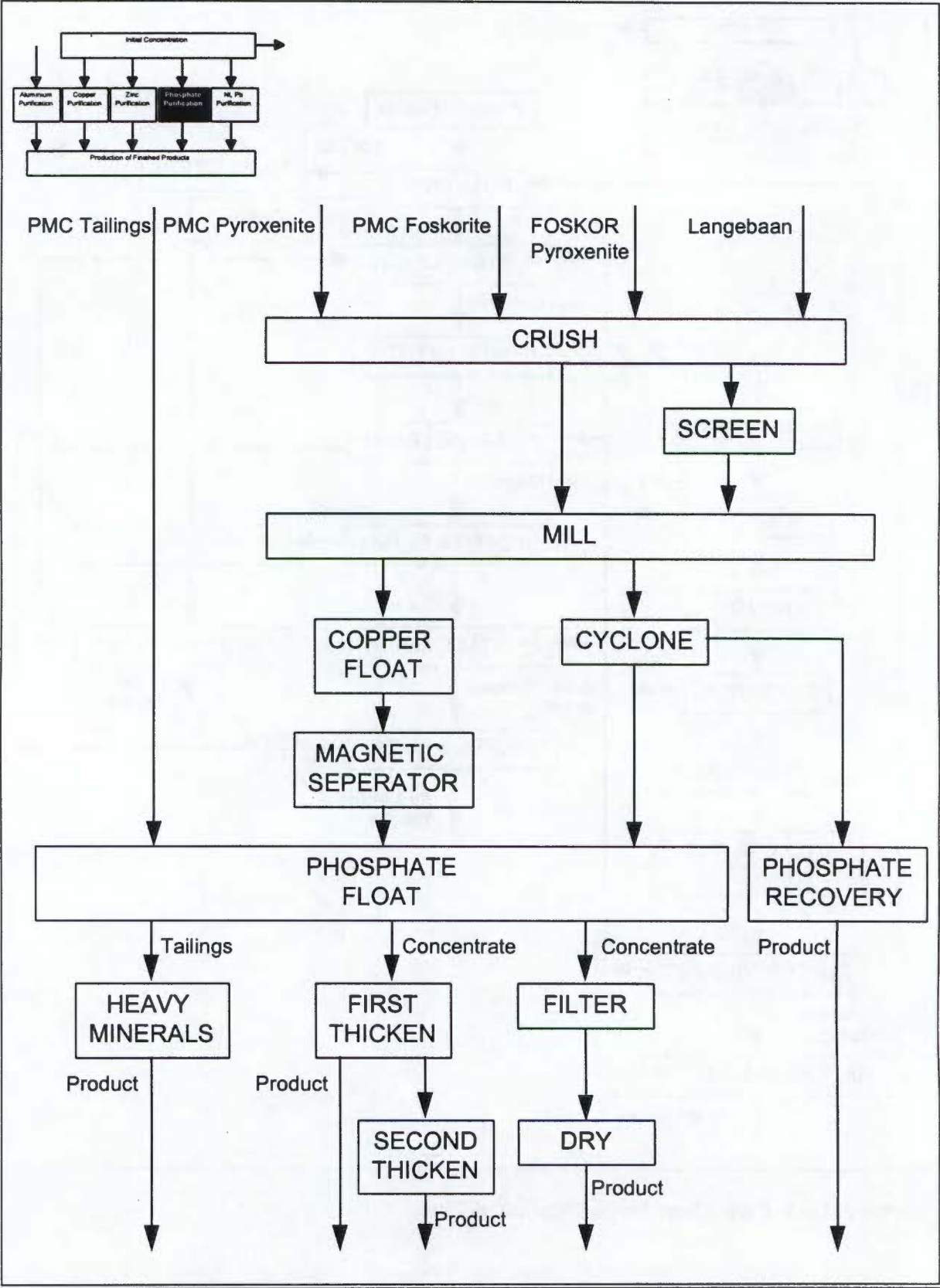


Figure A2.6-6: Flow sheet for Purification of Phosphate

2.6.3 Waste Inventory

Models of the generalised flow diagrams were used in calculating the waste inventories for the production of the various base metals during 1996. It must be noted that to ascertain the amount of metal produced by the base metals industry, the quantity of Nickel and Copper yielded as by-products of the PGMs industry were subtracted from the total Nickel and Copper production for that period. Table A2.6-2 gives the adjusted production values for which the waste inventories were calculated.

Product	Total Production (kt/annum)	Production from Base Metals (kt/annum)
Copper	152	144
Lead	89	89
Nickel	34	3
Zinc	77	77
Phosphate	2 700	2 700
Aluminium	570	570

Table A2.6-2: Base Metals Production from the Base Metals sub-sector

2.6.3.1 Predicted Values

The aggregated waste streams for all the processes in the base metals sector are presented in Table A2.6-3 in order of descending magnitude.

Product	Waste Generated (kt/annum)
Copper Purification	40 000
Phosphate Rock Production	8 500
Initial Concentration	8 000
Aluminium Smelting	920
Zinc Refining	230
Total	50 450

Table A2.6-3: Aggregated wastes for the Base Metals sub-sector

The components of the waste stream are detailed in Table A2.6-4.

Component	Waste Generated (kt/annum)	Generation to		
		Air	Water	Solid/Liquid
Water	30 000			
Ore	16 000			
Air	2 600			
CO ₂	570			
CO	320			
Oxygen	240			
Silica Flux	145			
Cu	100			
Coal (Carbon in slag)	90			
P ₂ O ₅	71			
Reagents	67			
H ₂ SO ₄	58			
SO ₂	54			
Steam	32			
Various Reagents	24			
Fluorine	18			
SO ₃	15			
Al ₂ O ₃	13			
Fe ₂ O ₃	9.4			
Fe	6.0			
Nitric Acid	6.0			
Al	3.3			
Zn	2.9			
Pb	2.1			
TiO ₂	1.6			
Zirconium	1.5			
Magnetite	0.65			
Total	50 450	4 450	2 900	43 010

Table A2.6-4: Components of the Base Metals waste stream

Two factors emerge from these figures: Firstly, water makes up the major part of the waste stream. Secondly, the nature of waste type differs between the processes. Some processes are purely hydrometallurgical in nature, so their waste is emitted mainly to the water stream. Other processes are pyrometallurgical and they have greater emissions to the atmosphere.

The figures again show the concentration effect remarked on in the PGMs section. Most of the waste material leaves the initial beneficiation stages. However, the wastes leaving the purification stages can be more hazardous to the environment, so they cannot be discounted.

It is also important to consider the waste streams separately. Aggregating them only masks the processes giving rise to them, and make suggestions focused on process improvement impossible. For this reason Table A2.6-5 has been included, presenting the break-down of the waste stream according to unit operation.

It is noteworthy that units dealing with initial beneficiation of ore tend to have high waste generation potential. The complete matrix of unit operation and waste stream components is available. This information can be used for pin-pointing the stream with the highest flow rate of hazardous materials and indicating action needed.

Process	Unit	Waste (kt/annum)
Initial Concentration 8 000 kt/annum	Loss of Recycle Water	4 900
	Tails Thickening	3 100
	Magnetic Separation	1.0
Copper 40 000 kt/annum	Tails	36 500
	Acid Plant	1 800
	Electrostatic Precipitator	900
	Reverb	420
	Gas Loss	380
Zinc 230 kt/annum	Leaching	120
	Recycle Water Cleaning	57
	Acid Plant	28
	Electrowinning	25
Phosphate 8 500 kt/annum	Heavy Minerals Plant	4 700
	Physical Separation	2 000
	Crush and Screen	1 300
	Thickening	500
Aluminium 920 kt/annum	Reduction	850
	Blend & Cast	70

Table A2.6-5: Waste generated according to unit of origin

2.6.3.2 Comparison with other information

Table A2.6-6 gives data obtained from the two CSIR managed studies for comparison with model predictions of waste generation. Again the main point of verification is the amount of ore in the waste stream. Not the entire industry has been presented here. The most notable omission is the copper purification industry which, as figures have shown, makes a significant contribution to the waste stream. However, all possible verification has been carried out.

Category	Waste Stream (kt/annum)	Hazardous Waste Groups			Generation To		
		Haz Waste (Grp 1-3)	Potential Haz (Grp 4)	Non- Haz (Grp 5)	Air	Water	Solid/ Liquid
Phosphate tailings	10920						
Base metal mining							
Overburden							
Tailings	30000						
	29600						
Zinc refining:-							
Copper cake	0.41						
Cobalt cake	14.80						
Cadmium cake	0.05						
Solids effluent	13.20						
Effluent	0.29						
Lead residue	3.28						
Iron residue	9.03						
As contain waste	0.08						
TOTAL	70561	276	135	70520	0	0	70561

Table A2.6-6: Waste Stream generated by the South African base metals industry
(References: FRD and CSIR)

The quoted value for the phosphate tailings is similar to the entire waste stream prediction for the phosphate industry (i.e., 8 500 kt/annum). This would imply that the water content of the stream has, in this instance, been included.

The overburden and tailings values from the base metals sector are difficult to verify as there is no indication as to which processes are included here. However, the total ore waste predicted for the industry is far higher than that quoted, the predicted value being in the region of 204 000 kt/annum which is more than three times greater than the quoted value.

The quoted total waste stream for the Zinc industry is 41 kt/annum compared with the 230 kt/annum predicted. When the water value is excluded from this stream the value falls to 93 kt/annum. For the value to be as low as that given in the reports, most of the gaseous waste would also have to be excluded.

2.6.3.3 Discussion

Levels of confidence placed in the various aspects of the models are presented and evaluated here. As in the case of the PGM industry, one model builds on the others and inaccuracies in one model will persist through all calculations. However, as the confidence levels show, it is still felt that the information is sufficiently defined to render values of use on the macro-scale at which modelling has been carried out.

Confidence levels - Initial Concentration

Flow Sheet	95%
Assumptions	80%
Stream Splits	85%

All of these confidence levels are high because plant-specific information was used. The assumptions have the lowest rating because operating conditions will fluctuate in all plants.

Confidence levels - Copper

Flow Sheet	95%
Assumptions	75%
Stream Splits	75%

Again plant-specific information was available but not in as much detail as for the Initial Concentration stage of the process.

Confidence levels - Zinc

Flow Sheet	80%
Assumptions	70%
Stream Splits	75%

Here most of the information was drawn from journal articles and technical texts. Their applicability to the South African situation was not always clear. Modelling some of the smaller units was difficult as little information on their operation was available.

Confidence levels - Phosphate

Flow Sheet	90%
Assumptions	65%
Stream Splits	70%

As already stated, this section was modelled more to demonstrate the cross-linking within this sector than for accurate waste stream calculation for the industry as a whole. No plant-specific information was available.

Confidence levels - Aluminium

Flow Sheet	95%
Assumptions	80%
Stream Splits	80%

Although generalised, the information generated by the Aluminium model is good because the process employed is well known and thus well defined.

2.6.4 Using the Model - Interlinking of Processes

There is an interesting interlinking of processes between the major copper producer and downstream production of phosphate rock. What would be considered a waste stream from the copper process is sold on to the producer of phosphate where the phosphate contained in the stream is removed. Thus the waste ore becomes a "product". The phosphate producer also sells a stream on to a fertiliser manufacturer. This is, in practice, a waste minimisation approach. Moreover, the water stream is returned to the copper producer once its content of saleable phosphate has been removed.

It was decided that this interlinking of processes should be evaluated. A comparison of the plants working on an interlinked basis was made with the processes working independently. It was assumed that the phosphate producer would not increase production to make up for lost capacity should the waste stream from the copper producer not be available. Table A2.6-7 is included to show the change in the waste stream when the processes are decoupled. The values in this table show the percentage change in the waste stream as a ratio of the base-line (or coupled) waste stream; i.e., a negative value denotes a decrease in the waste stream on uncoupling of the process.

Waste Component	Increase in waste stream (kt/month)
Ore	-167.32
Cu	-3.90
P ₂ O ₅	12.33
Water	791.86
Nitric Acid	-0.33
Reagents	0.44

Table A2.6-7: Increase in waste stream when the copper stream is sold on to the phosphate producer

As expected, the amount of ore solid waste decreases as does the quantity of copper being lost. However quite a large increase in the water waste stream occurs because a significant amount of water is used in the phosphate process. Figure A2.6-7 has been included to give an indication of the masses of material involved. Note: This figure is plotted on a log scale.

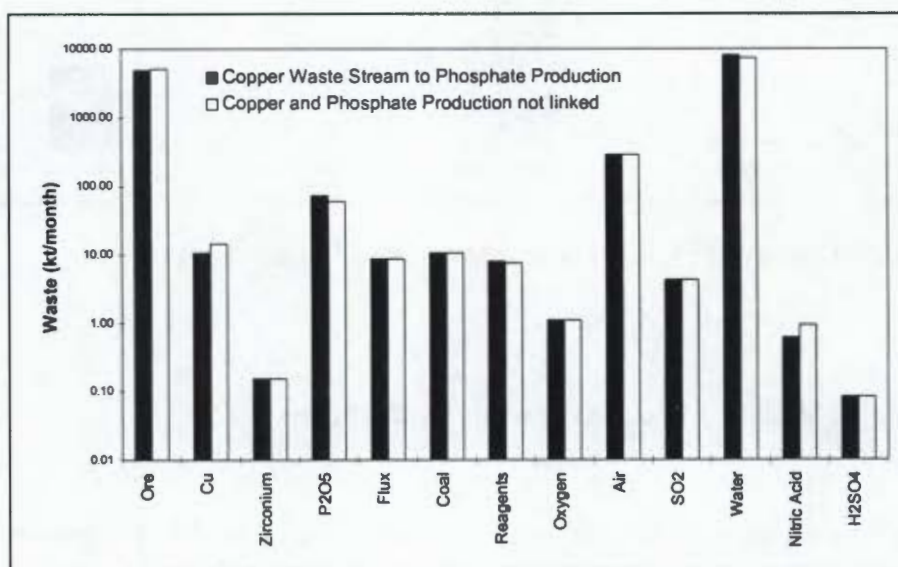


Figure A2.6-7: Change in Waste stream with decoupling of Processes

What this figure shows is the relatively low mass increase that the values in Table A2.6-7 represent. In studying the waste stream components it becomes obvious that the nature of the waste will not change radically. The coupling of the two processes does not introduce a new component into the waste stream.

It has been acknowledged that the magnitude of the waste stream increases with the coupling of the processes. The magnitude of this change is 600 kt/month. In conjunction with this increase in waste there is an increase in product. This increase in the product stream is reflected in Figure A2.6-8 which shows an increase in the phosphate stream and in the stream available to the fertiliser producer. If a phosphate product value of R86.00 per ton is assumed (DMEA, 1993), and a phosphate value of R40.00 per ton in the stream sold to the fertiliser producer is estimated, the resulting increase in income for the phosphate producer would be R4 500 000 per month. The increase in income per increased kiloton of waste is R7200.

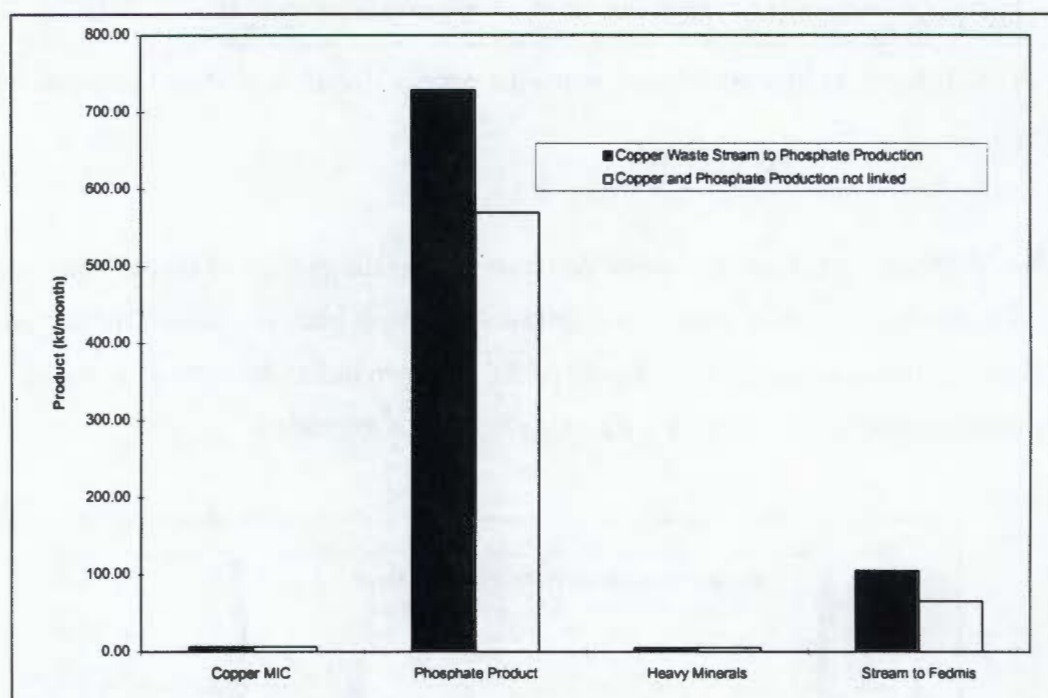


Figure A2.6-8: Change in Products with decoupling of Processes

2.6.5 Using the Model - Allocation of waste stream to product

In order to determine the environmental implications of a specific product it is necessary to allocate the waste stream created in producing the output to the product. Complications arise when there is more than one product produced by a single process or plant. Such is the case in the base metals

sectors. If a process gives rise to more than one product and has more than one point of output from the process then the waste stream can be allocated to each product in accordance with its point of exit from the process.

Allocating the waste stream to a product is important when one performs the LCA on a specific product. A metal in the hands of a dealer in Europe may be drawn in part from metal purified in the land of origin, and the balance drawn from sources in other countries. The approach developed makes it possible to allocate the waste stream to that metal according to all the processes involved in its purification.

In this case study the initial beneficiation stage of the base metals industry and its linking to the downstream processes of copper purification and zinc purification was investigated. The initial beneficiation stage yields three products - a lead concentrate, a copper concentrate and a zinc concentrate. These products leave the operation from three easily definable stages of the process. The waste stream from each stage can thus be apportioned according to a set of determined ratios, to the product leaving it. Waste streams can also be allocated to the products from the two purification stages generating them.

For simplicity the waste stream generated by the initial concentration stage on its own is considered. This is depicted in Figure A2.6-9.

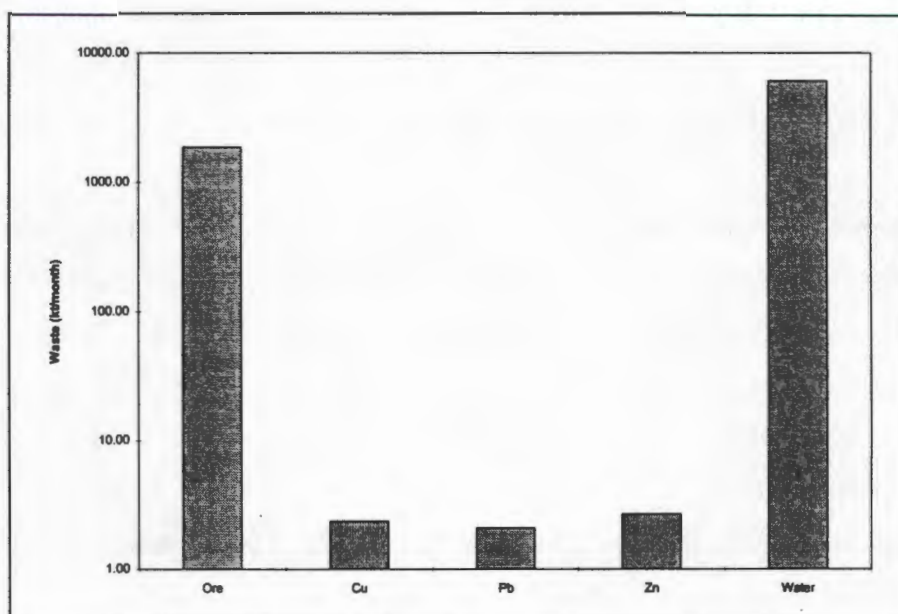


Figure A2.6-9: Total waste stream from Initial Concentration

The waste stream leaves the process at three distinct points which coincide with points at which the products leave the process. It is thus possible to allocate a waste stream to its point of exit from the process and link it to a product stream. These waste streams are illustrated in Figure A2.6-10.

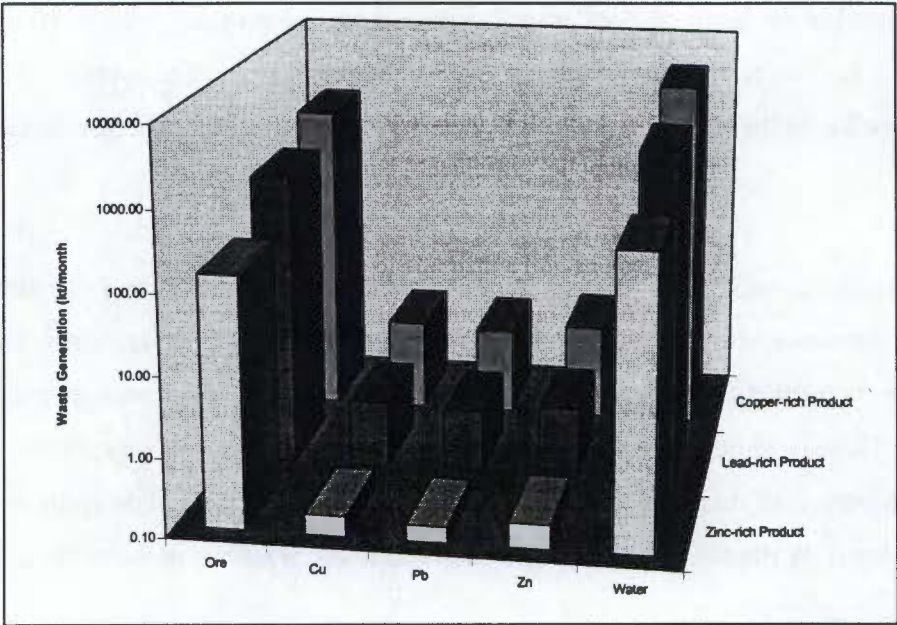


Figure A2.6-10: Breakdown of Waste Stream by Product

However, the product streams each contain in differing proportions, three saleable components - copper, lead and zinc. It is possible to assign the waste stream to a specific component. This allocation can be carried out in two ways, firstly on a mass weighted basis. An illustration of the mass break-down of the product stream is included in Figure A2.6-11.

Allocating the waste on a value weighted basis is included because the ability of a process to pay for the amelioration of the wastes which it generates is determined by the income which it earns. In calculating the values on Figure A2.6-12 the following constants were used:

Copper MIC	R 3730/tonne
Lead MIC	R 1100/tonne
Zinc MIC	R 1110/tonne

The weightings represented by Figure A2.6-11 and Figure A2.6-12 were then used in allocating the waste stream to the specific products.

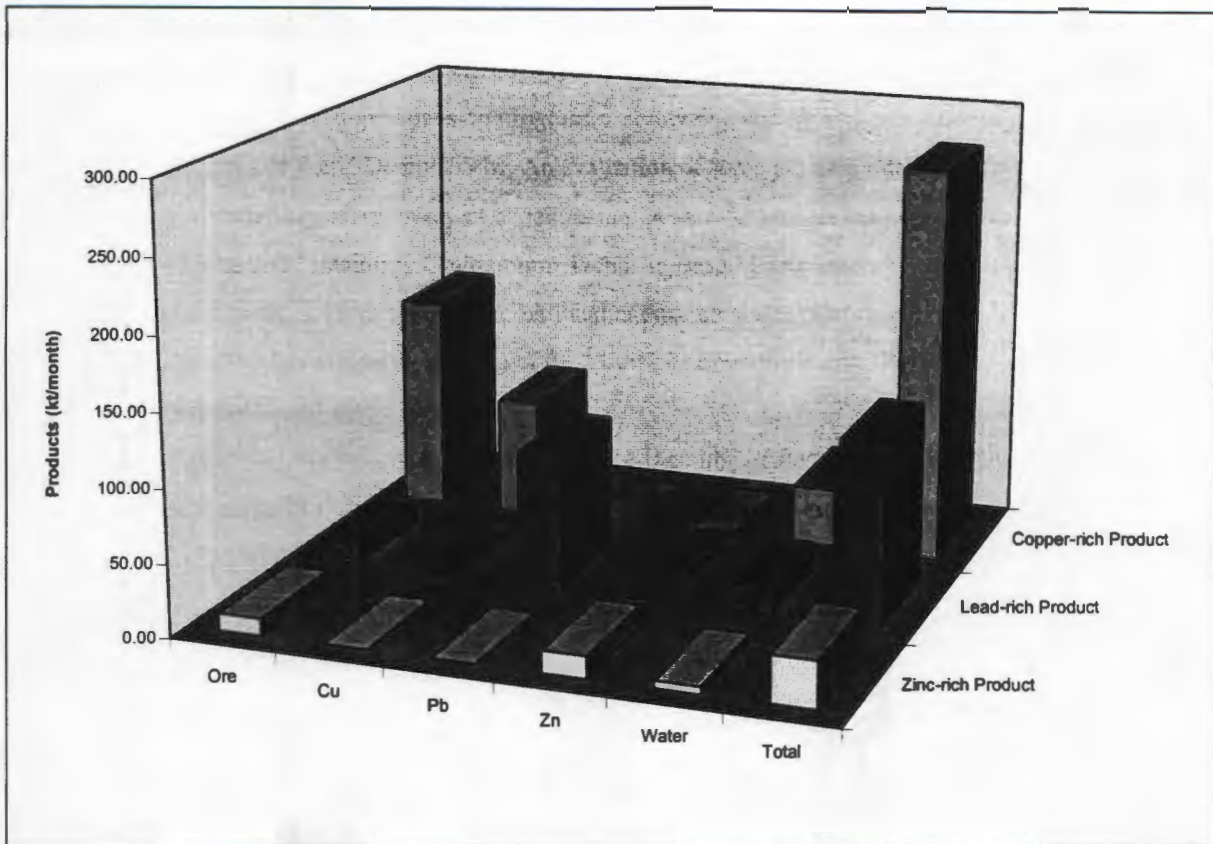


Figure A2.6-11: Break down of Products

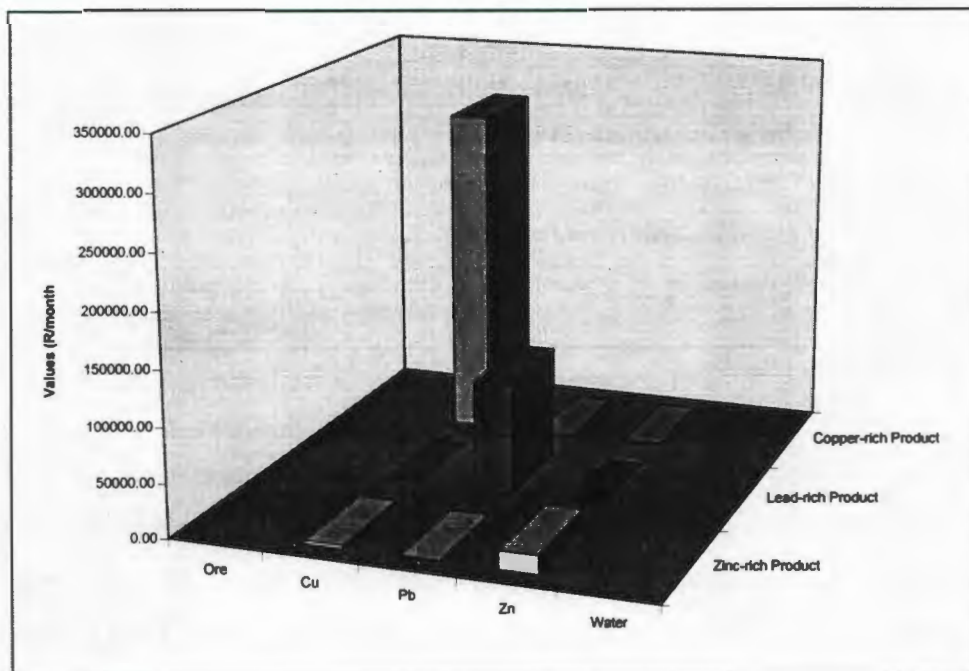


Figure A2.6-12: Value of products

Having completed the allocation of the waste stream for the initial beneficiation process, one can consider copper smelting and zinc electrowinning, the other two processes involved in South African copper and zinc production. In allocating these waste streams it was assumed that these operations produced only one saleable product - the metal itself. The overall waste stream for the products - copper, lead and zinc - is presented in Figure A2.6-13.

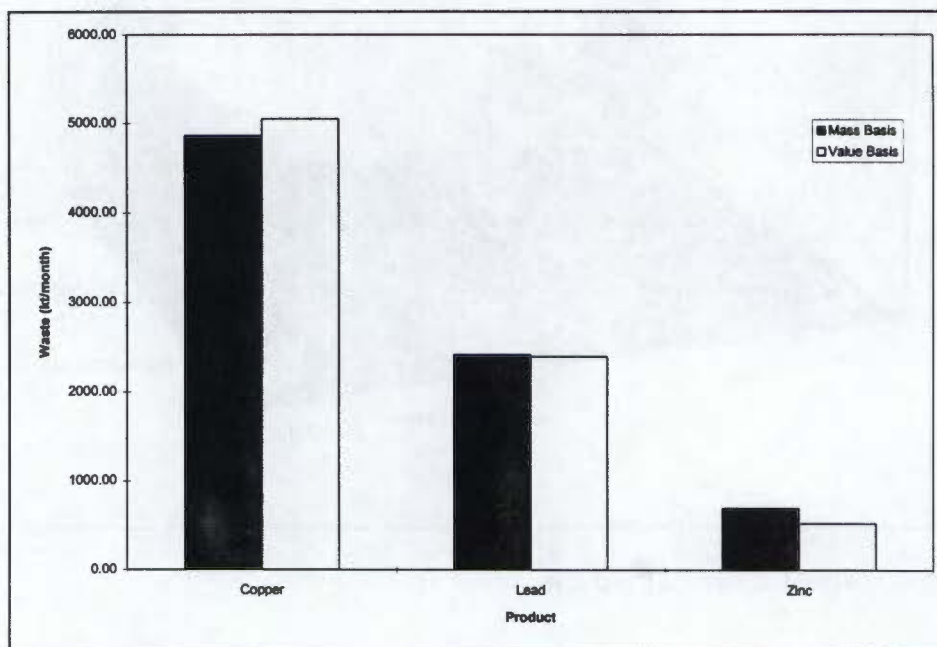


Figure A2.6-13: Waste Stream Allocation

It is obvious that the waste streams are greatly affected by this inclusion of the refining processes.

The amount of waste generated depends on the quantity of material entering the process. Therefore, a normalisation of the waste stream is necessary if the waste generation linked to each different product is to be compared. The wastes generated have been normalised with respect to the total value of product arising from the process.

Our main reason for doing this is the present government initiative to increase the value added to South Africa's primary products - to increase the level to which metals are beneficiated. This is sensible when the values of, for example, copper as metal in concentrate (MIC) and copper in metallic form are compared. The average export value of MIC copper was R3 700 per ton for the year 1996. That of metallic copper was R10 200.

However, increased beneficiation levels would lead to an increase in the waste stream, the purification steps, as observed, having fairly high waste generation potential. The type of waste generated would also be different. Use of LCA will make it possible to anticipate the increase in waste generated relative to the increase in product value.

Figure A2.6-14 and Figure A2.6-15 show the differences in the waste generation potential of the various metals once waste has been normalised with respect to the income generated by the product. Referring to Figure A2.6-14 one can see that for each unit of income for the copper process, less waste is generated in the production of concentrates, than in the production of the refined metal. Cost, when the overall amount of waste generated is normalised with respect to income, is R26/tonne.

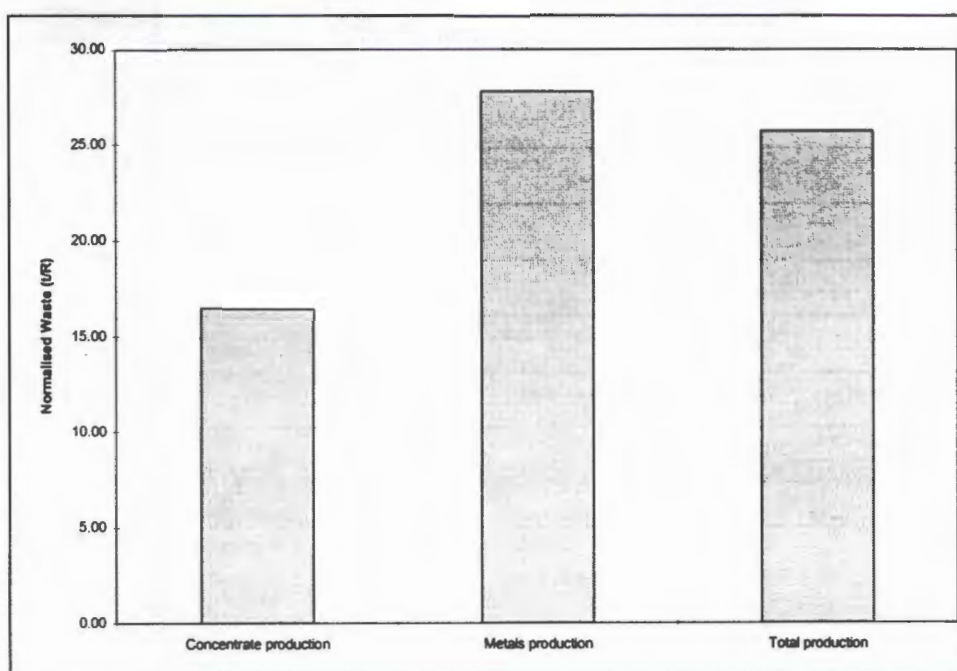


Figure A2.6-14: Waste Generation of the Copper Industry Normalised with respect to value of Copper produced

The picture for Zinc is different (Figure A2.6-15). The refining process of this product produces far less waste per unit of income than does the initial concentration phase. The overall waste generation cost for Zinc is R1/tonne.

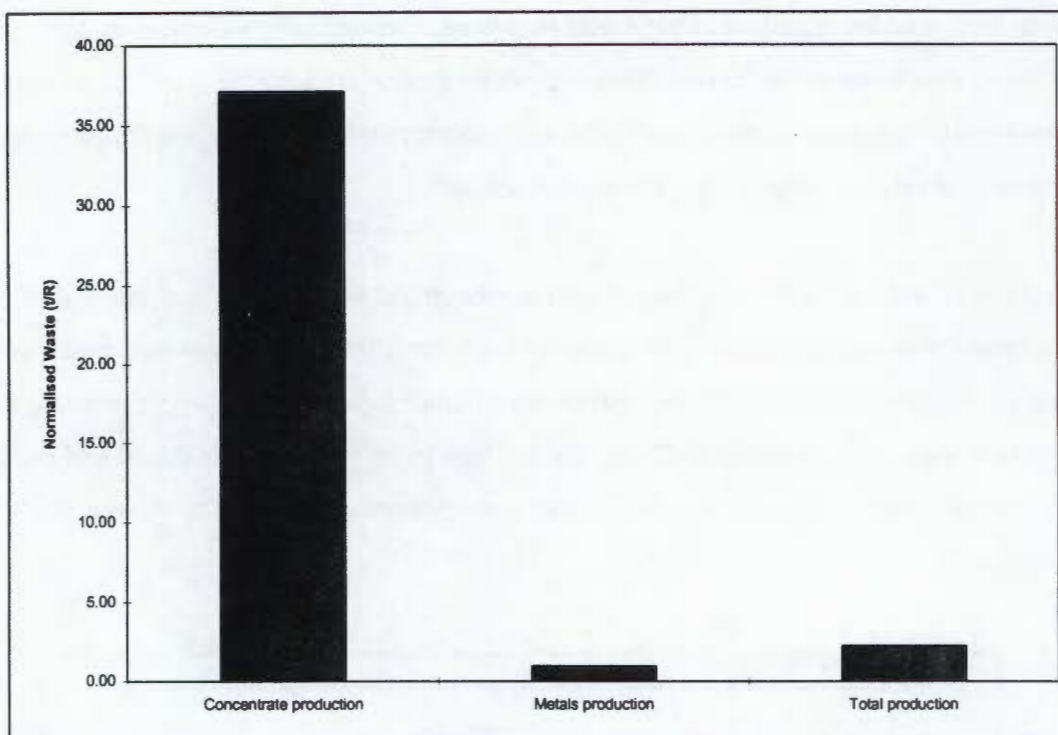


Figure A2.6-15: Waste Generation of the Zinc Industry Normalised with respect to value of Zinc produced

Thus, when value-added processes are contemplated, if the only factor to be considered is their waste generation potential in relation to income derived, then zinc processing should be chosen in preference to copper processing.

2.6.6 Using the Model - Relative environmental performance of Technologies

Another factor needs to be taken into account, that of the hazardousness of the waste being generated by the processes. Hazard is a subjective concept and different definitions are applied in different countries. In this case study the hazard rating applied to wastes has been consistent with the South African Water Act. Within this act there are five groups of hazard ranging from highly toxic (Group 1) to non-hazardous (Group 5). All wastes which fall in groups 1-4 according to this rating have been included as hazardous. Table A2.6-8 contains ratios of

- Product to total waste produced
- Income to the process to total waste produced by the process
- Income to the process to hazardous waste produced

Including income in these ratios is important as this gives an indication of the ability of the process to pay for its environmental liability (where environmental liability is seen as an indication of the environmental risk associated with a waste; this liability will be greater for more hazardous wastes).

	Product:Total Waste (kt/kt)	Income:Total (R 000/kt)	Income:Hazardous Waste (R 000/kt)
Initial Concentration			
Cu	0.002	30	
Zn	0.03	130	
Zn:Cu	14	4	
Purification			
Cu	0.02	820	900
Zn	0.30	3 800	10 000
Zn:Cu	16	5	11
Overall			
Cu	0.003	90	1 300
Zn	0.15	1 800	10 500
Zn:Cu	47	21	8

Table A2.6-8: Comparison of Total Waste and Hazardous Waste generated in the Production of Copper and Zinc

Studying the first column in this table shows that, in producing the same amount of waste, 47 times more Zinc than Copper is produced. Copper has a higher sales value than Zinc and this too can play a part. When this is factored in it can be seen that, in producing the same amount of waste, Zinc generates 21 times the income than the Copper process. The Zinc process produces far more hazardous waste however, and the amount of income from Zinc is only eight times that of copper for the same amount of hazardous waste.

This would imply that, overall, the zinc process would be in a better position to pay for remediation of its environmental impact than would be the case with copper. There is an initiative in South Africa at present to promote “value-added” processes, i.e., the purification of MIC. Should this be the case, the income generated by the Zinc industry is eleven times greater than that of the copper industry in producing the same quantity of hazardous waste and thus more attention should be paid to the Zinc industry than the Copper industry in this respect.

Although somewhat cursory, this case study illustrates the concept of comparing disparate technologies and products. This is made possible by the consistent boundary drawn when establishing the various flow sheets.

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Appendix 2-7: Sub-sectoral study on the Ferro-alloys Industry

2.7.1 Introduction

Two notably different types of process are considered here. The first is the initial beneficiation of manganese, vanadium, chrome, iron and silicon ores. This is then extended to include the production of any chemical compound forms of the metals sold as salts or other compounds. The second group of processes covers the production of various metallic products, ranging from pig iron through various combinations of ferro-alloys to stainless steel.

South Africa's ranking in this sub-sector, with respect to reserves, production and export is shown in Table A2.7-1. These figures make it plain that this is a sector with a long projected life, and one with immense room for growth.

Product	Reserve Base Mt	Rank (% of world reserves)	Production kt/annum	Rank (% of world production)	Export kt/annum	Rank (% of world exports)
Chrome	3 100	1 (68%)	4 810	1 (44%)	1 300	1 (44%)
Iron	5 900	8 (6%)	29 200	8 (3%)	19 000	6 (4%)
Manganese	4 000	1 (80%)	3 200	3 (14%)	1 600	1 (27%)
Silicon			29	8 (4%)	25	4 (5%)
Vanadium	12	1 (45%)	15	1 (57%)	14	1 (80%)

Table A2.7-1: Reserves, production and exports of the South African Ferro-Alloys sector (DMEA, 1997)

The quantities of the various products in this sector produced for the year 1996 are listed in Table A2.7-2. Note that because actual production figures for steel are not available, the quantity given for steel produced is the installed furnace capacity.

Chrome Producers : There are 17 chrome mines in South Africa. Their product range includes metallurgical material, various concentrator products (fines, sands, etc.) and chemical grade chromite. Most of the chrome produced is used locally as an alloy in the production of steels.

Iron Producers : Iron is produced as either a primary or a secondary product, in eight processes.

Products are usually hematite and magnetite although some titaniferous magnetite is produced. About 40% of the mined iron ore is exported.

Manganese Producers : There are nine manganese producers in South Africa. Their products range from manganese ore, through chemical grade manganese and manganese sinter to battery grade manganese which is produced from a dump reclamation programme. Included in this section is the production of ores, chemical products and manganese alloys.

The income from the sector in 1996 was R 9.2 billion.

Product	Breakdown	Production (kt/annum)
CONCENTRATED ORES		
Chromite	< 44-%	3 200
	44-48%	1 600
	>48%	10
	Overall	4 810
Iron	Hematite	27 000
	Magnetite	2 200
	Overall	29 200
Manganese	30-40%	1 100
	40-45%	220
	45-48%	140
	>48%	1 700
	Chemical	40
	Overall	3 200
Silicon Metal		29
Vanadium		15
ALLOYS		
Charge Chrome		2 800
Ferro-Manganese	Fe-Mn and Fe-Si-Mn	500
	Other Fe-Mn	240
	Overall	740
Ferro-Silicon		90
STEEL		
Carbon Steel		200
Stainless Steel		7 800

Table A2.7-2: Products from the Ferro-Alloys sector - 1996

Silicon Producers : Silicon metal is the product of only one process in South Africa. Although some silicon metal is sold, most is used in manufacturing various silicon alloys.

Vanadium Producers : There are four vanadium ore producers in the ferro-alloys sector. Various vanadium chemicals are produced. Most vanadium finds its way into ferro-alloys production.

Steel Producers : These producers rely on the production of the metals mentioned above. There are eight steel processes in the country. This however, is one of the industrial sectors showing the greatest growth at present and various process upgradings and new process developments are taking place.

2.7.2 Generalised flow diagram

The ferro-alloys sector is extremely complex, with many interconnected processes. Figure A2.7-1 contains the macro-scale flow diagram for the industry and indicates the number of products as well as the number of processes.

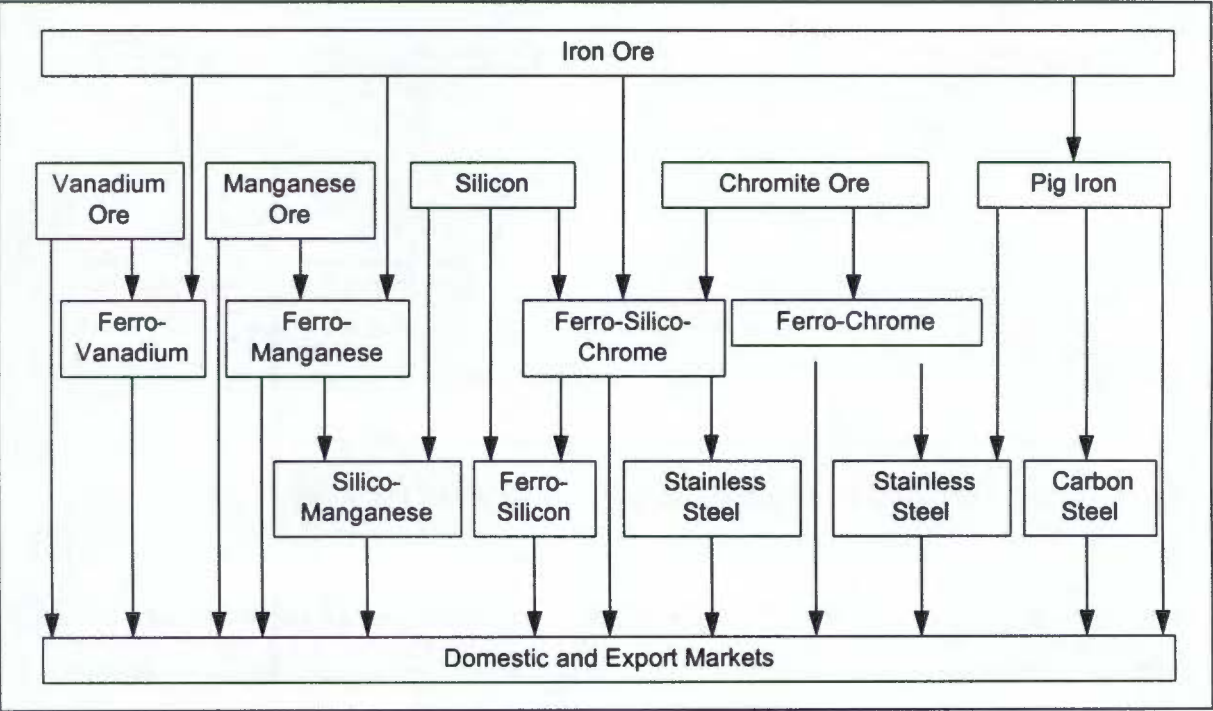


Figure A2.7-1: Overall flow diagram for the Ferro-Alloys industry

Processes within the sector have significant overlaps with similar process routes being adopted in many cases. For this reason a generic initial concentration model was used. This model is diagrammed on Figure A2.7-2. However, to maintain detail within the waste inventory and link wastes to specific products, the models for the various alloy processes (which contain many similar processes), were developed independently. Common processes were duplicated for each product.

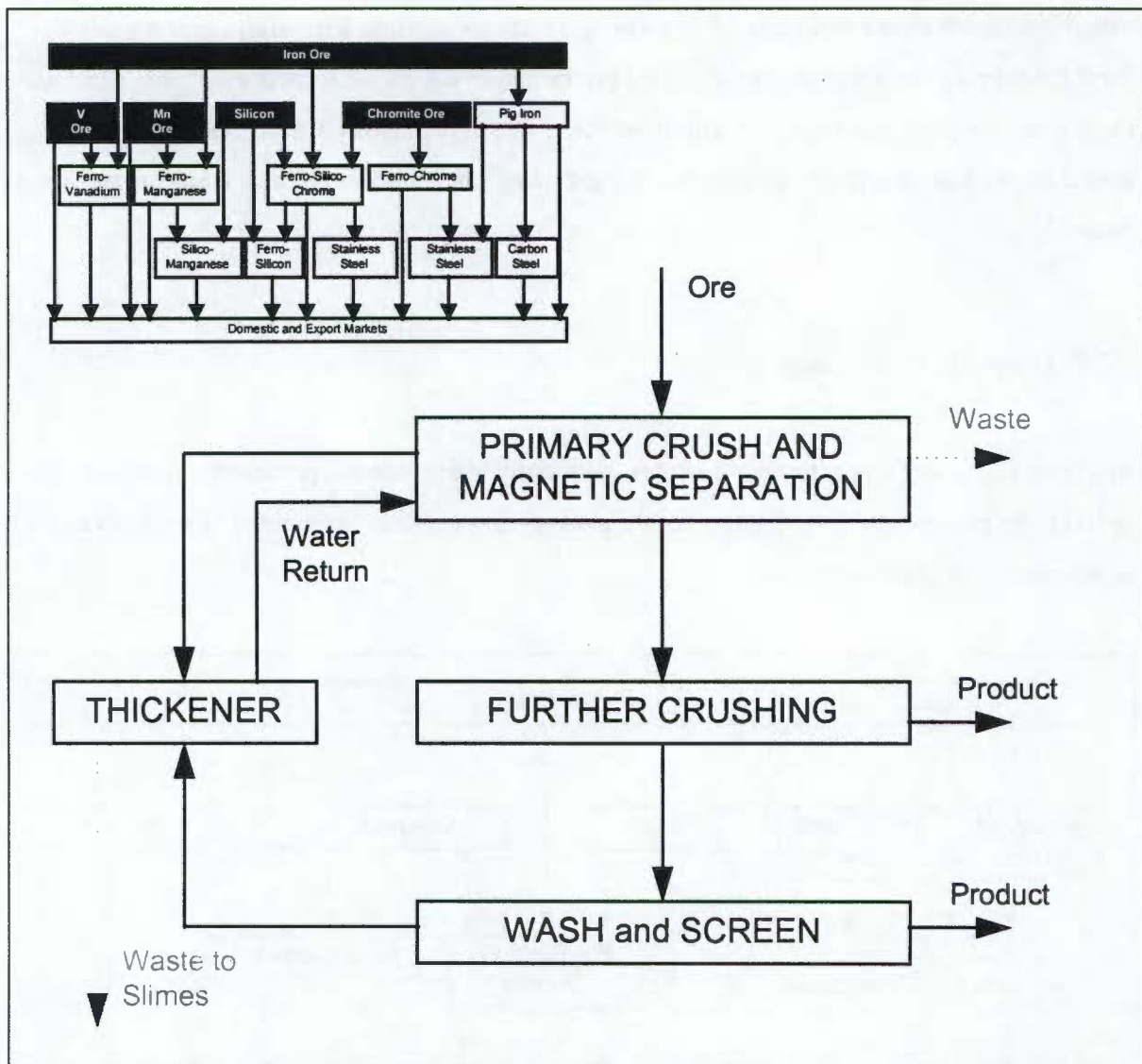


Figure A2.7-2: Generic flow diagram for ore dressing in the Ferro-alloys industry

Because the industry is so complex the flow diagrams were developed on two levels. The first details the routes followed by specific metals within the sector, while the second shows unit operation details within the processes themselves. These two levels are included in Appendix 1-2. Appendix 1-2a gives the metal-specific flow routes, while Appendix 1-2b shows unit operations.

Two of these flowsheets have been included to illustrate this concept. The flow route for vanadium has been included in Figure A2.7-3. This figure shows the route that vanadium flows through the overall diagram. The operation of the ferro-vanadium plant is described by Figure A2.7-4. This shows the operation of one of the process blocks in Figure A2.7-1. These figures have a “route map” which places the diagram within the operation of the overall diagram.

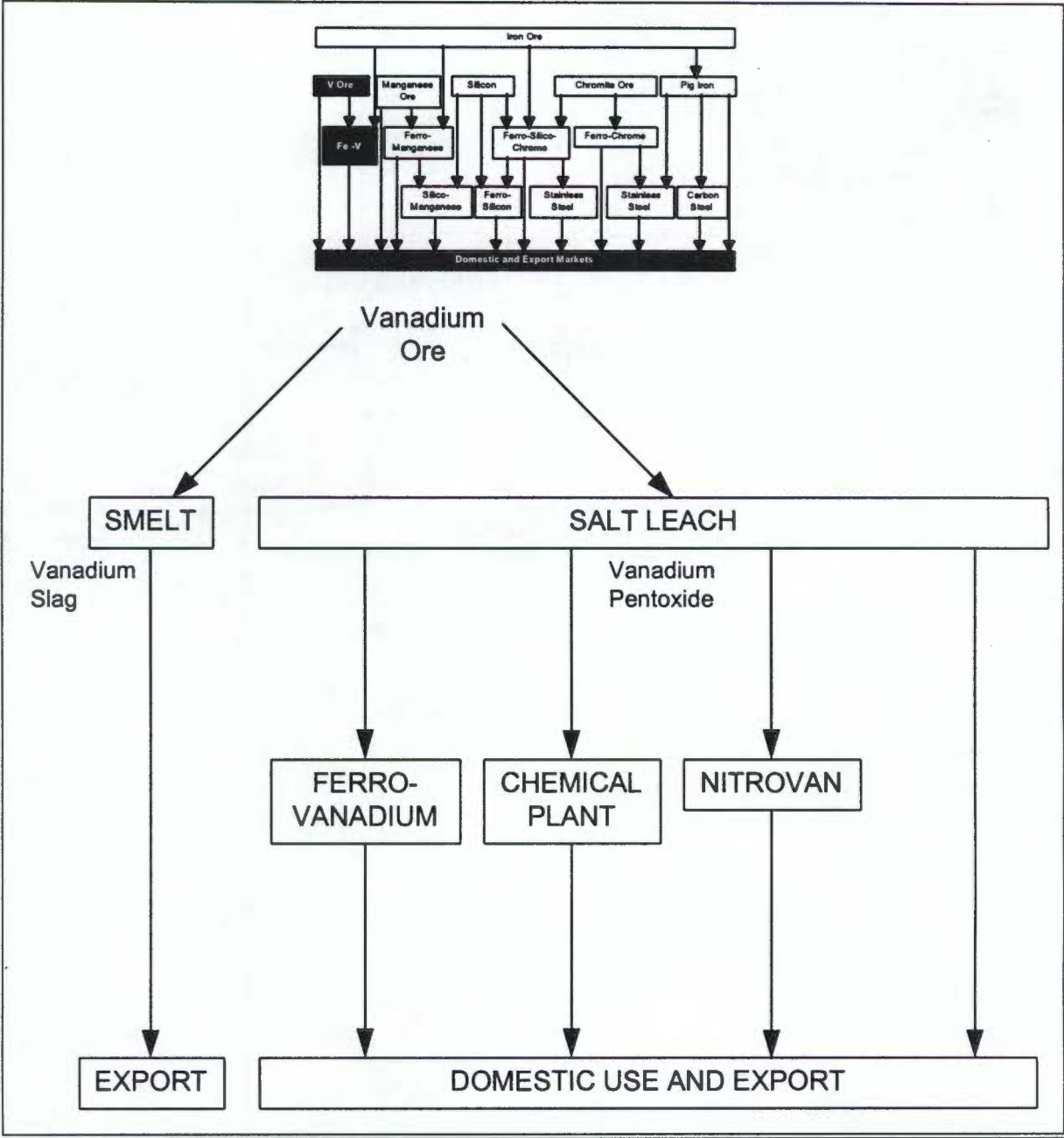


Figure A2.7-3: Overall Flowsheet for Vanadium

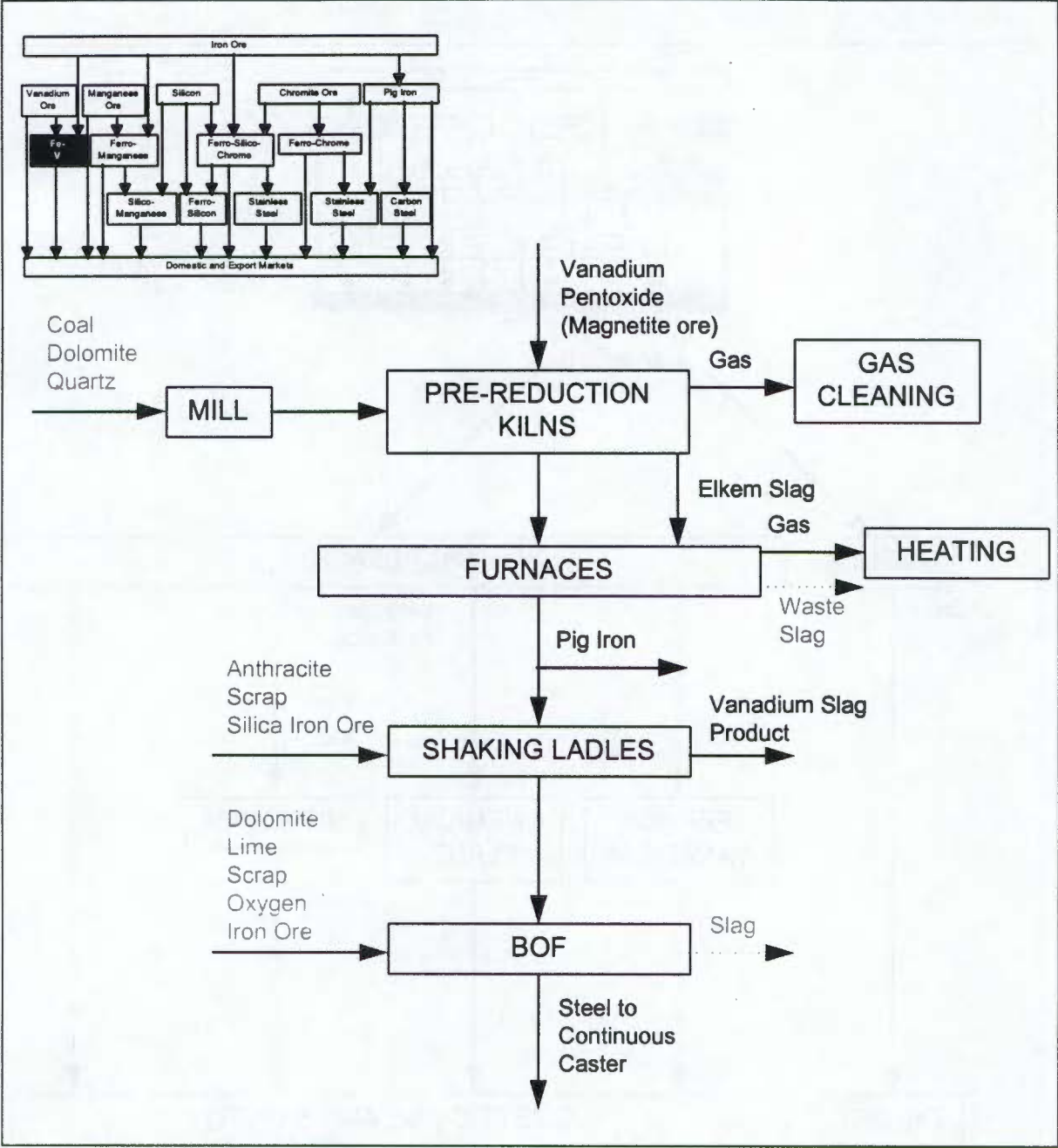


Figure A2.7-4: Ferro-Vanadium Plant

2.7.3 Waste Inventory

The total waste inventory for this sub-sector was modelled for the production figures quoted in Table A2.7-2. The waste inventory includes a significant volume of information as detailed as component waste generated by each unit operation, within each process, for the sub-sector.. For ease of interpretation, the inventory was aggregated to get total waste values for the industry.

2.7.3.1 Predicted Values

Table A2.7-3 contains the component break-down of the total waste stream generated by the ferro-alloys sector. This total was just over 41 Mt for the year under consideration. As noted in the other sub-sectors, most components in the waste stream originate in the initial concentration phase of the industry.

Component	Waste Generated (kt/annum)	Generation to		
		Air	Water	Solid/Liquid
Water	9 600			
Ore	9 200			
Coal	6 300			
SiO ₂	3 800			
Other	2 500			
Fe	1 800			
CaO	1 700			
CO	1 300			
MgO	1 000			
Al ₂ O ₃	730			
MnO	570			
Mn	550			
TiO ₂	360			
CO ₂	320			
H ₂	310			
Ash	270			
O ₂	270			
Wash Solution	69			
FeO	55			
Cr ₂ O ₃	11			
V ₂ O ₅	10			
Cl ⁻	10			
Na ⁺	7.9			
SO ₂	7.0			
Air	5.8			
HCl	5.4			
VO ₃ ⁻	5.2			
NH ₃	5.0			
NaVO ₃	2.2			
S	0.16			
Total	41 070	17 000	70	24 000

Table A2.7-3: Component break-down of waste generated by the South African Ferro-alloys sector

Figure A2.7-5 shows the spread of this waste over the various processes within the sub-sector. It contains information about the total amount of waste originating from each process. The component distribution of this waste stream is given in Figure A2.7-6. It includes the ten components most significant on a mass generation basis and shows how much waste each specific process generates.

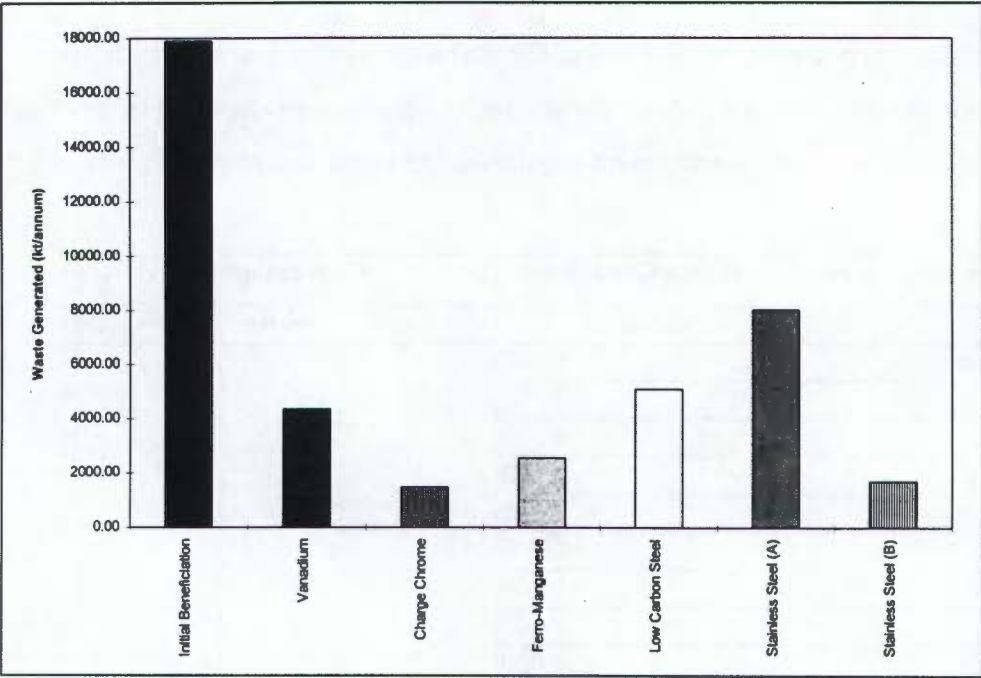


Figure A2.7-5: Waste Inventory for the Ferro-alloys Industry

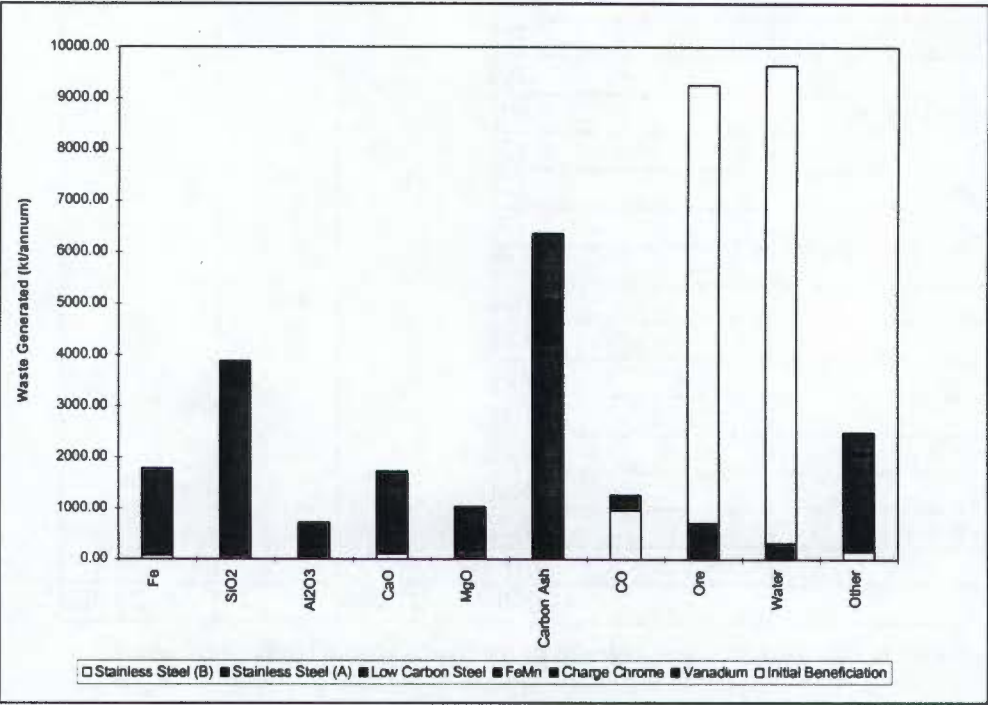


Figure A2.7-6: Component Detail of Waste Generated - Ten components with highest mass generation

This figure also illustrates the predominance of ore and water waste originating in the initial beneficiation stages of the sector. The various reagents used in the pyrometallurgical production of the different alloys are distributed over the various processes. Most of the coal-based waste is generated by the conventional electric arc furnace process used in the production of stainless steel. This can be explained by the size of the installed capacity of this process. As might be expected the most SiO₂ waste is generated in the production of carbon steel.

Some smaller components in the waste stream (Table A2.7-3) are considered hazardous in nature. Despite the small mass generated, their environmental impact may be significant. Other waste stream components are possible feedstocks and should be kept separate. Figure A2.7-7 show the distribution of the most significant of these components for the various process routes modelled.

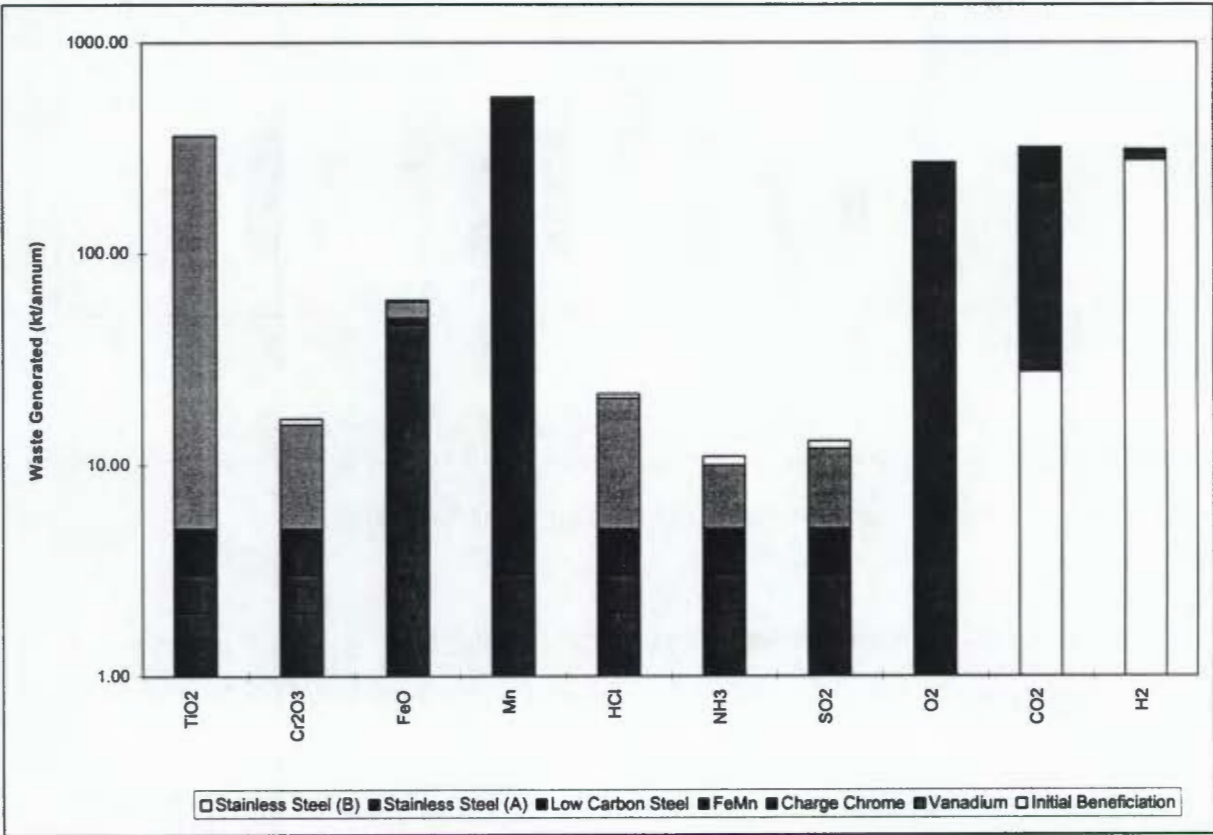


Figure A2.7-7: Component Detail of Waste Generated - Ten most hazardous or valuable components

Amongst toxic emissions the amount of Cr₂O₃ in the waste for the sector is of concern as is the quantity of acidic waste. On a global scale the production of SO₂ is of general concern.

The waste inventories having been detailed to process unit operation level and are available on request. These figures show the component waste distribution across the various processes within the sector. Figure A2.7-8 has been included for illustrative purposes. Initial concentration has not been included as it really only generates two main waste components, namely water and gangue.

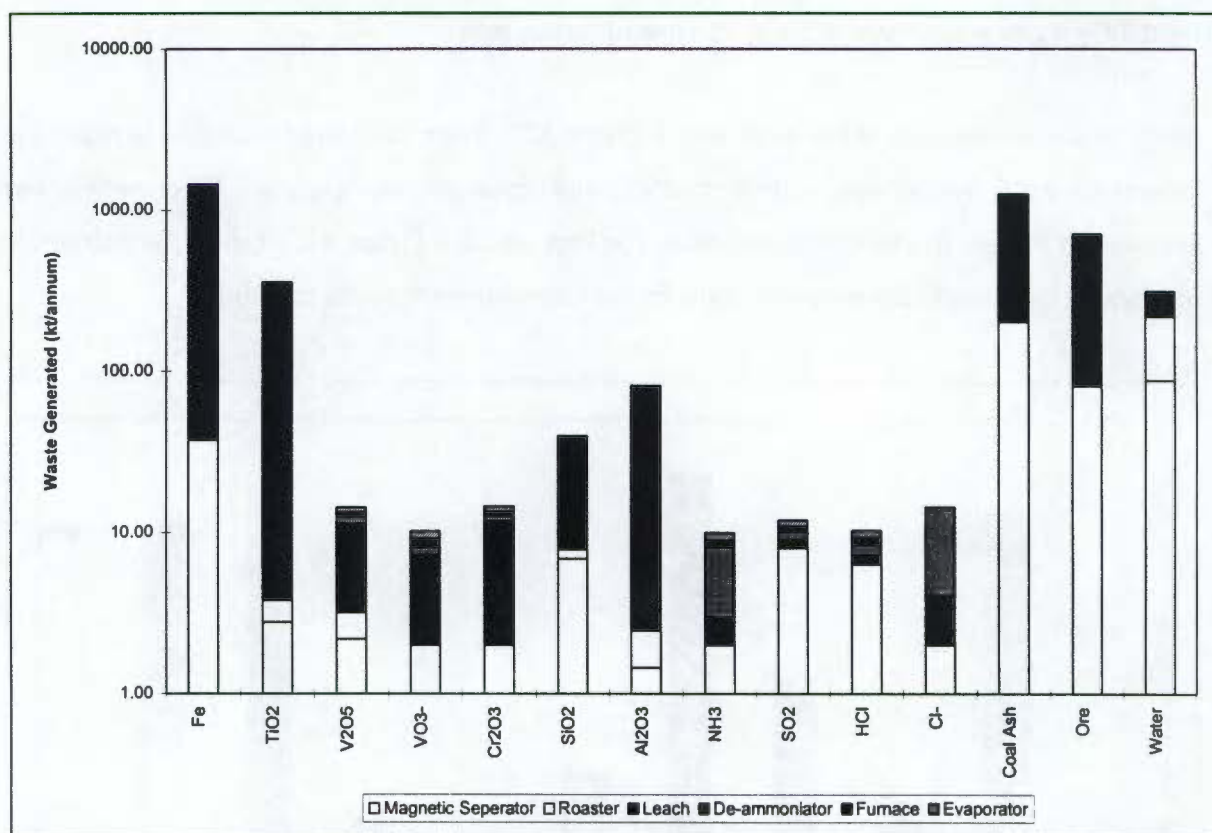


Figure A2.7-8: Waste Distribution for the production of Vanadium

The suite of figures illustrate two mechanisms within the model:

- It is possible to determine the waste associated with a specific product flow within an overall process
- It is possible to separate the waste inventories into different stages (*viz.*, concentration and purification)

These concepts were also explored in the appendices relating to PGMs and Base Metals.

2.7.3.2 Comparison with other information

The information available for verification of the models is detailed in Table A2.7-4.

A comparison with this information is included on Table B5.5. This table shows significant differences between the modelled values and those quoted in literature.

Category	Waste Stream (kt/annum)	Hazardous Waste Groups			Generation To		
		Haz Waste (Grp 1-3)	Potentia l Haz (Grp 4)	Non- Haz (Grp 5)	Air	Water	Solid/ Liquid
Steel making							
Slags	2242						
Dusts & sludges	289						
Raw material or product fines	80						
Refractory waste	214						
Fe-Silicon							
Dust	40						
Slags	1600						
Filter residues	18						
FeMn slag	370						
Gas emissions	13						
Waste water	16						
TOTAL	4883	324	4559	0	13	16	4854

Table A2.7-4: Waste Stream generated by the South African ferro-alloys industry

Process	Modelled	CSIR Data	Ratio (Modelled/CSIR)
Steel Making	8 500	2826	3.0
FeCr and FeSi	13 000	1658	7.8
FeMn	980	370	2.6
Gas emissions	2 300	13	170
Waste Water	9 700	16	610
Water excluding Initial Beneficiation	380	16	24
Contaminants in Water	20	16	1.3

Table A2.7-5: Comparison of modelled waste inventory with data from literature

The differences in the waste inventories for the Steel and FeCr/FeSi processes are significant. There are two possible explanations:

- An underestimation of the values in the literature survey or an overestimation in the modelling assumptions.
- New processes have been commissioned since the evaluation of the CSIR/FRD data; because of the static nature of the data-base it is not possible to incorporate these changes.

The extreme disparity in the gaseous emissions may be ascribed to the quantification of the amount of air in the stream as well as the waste components within the stream in the modelled results.

The difference in the amount of waste water leaving the industry probably results from our inclusion in the inventory, of waste water from the initial beneficiation. Excluding this value renders a far better (albeit not necessarily acceptable) comparison. The values are quite similar when only the chemical water contaminants are aggregated.

2.7.3.3 Discussion

There are vast differences between the modelled waste inventory and that obtained from literature data. These disparities have not been fully explained but some could be ascribed to differences in the definition of waste streams and the fact that modelled values included dilutants such as water and air.

The confidence levels for the information used to model the entire sub-sector remain constant and are indicated below :

Confidence levels - Ferro-alloys

Flow Sheets	90%
Assumptions	75%
Stream Splits	65%

The confidence placed in the flowsheets is high as the processes are fairly standard and well-defined. The assumptions on the other hand are not as easy to source because the operation of the processes changes continuously in accordance to product specification changes, and because of the variety of products from the sector. The confidence placed in the stream splits is low it is possible that there is duplication in some of the data used (i.e., that some double accounting is taking place, material accounted for in one process also being included in others).

2.7.4 Using the Model - Improvement in Technology

The steel manufacturing sector has gone through many stages of development. The waste generation potential of these technologies is compared here. The Corex process is used to demonstrate the waste generation potential of modern technologies. Figure A2.7-9 shows how the waste stream differs for four different technologies. First is a blast furnace, the oldest technology currently employed. This is followed by the next step in technological development, the basic oxygen furnace (BOF), then by the Corex process and, finally, by the coupling of a Corex reactor with a downstream direct reduction process (DRI) using the CO- and CO₂-rich off-gas from the Corex process in the reduction of iron.

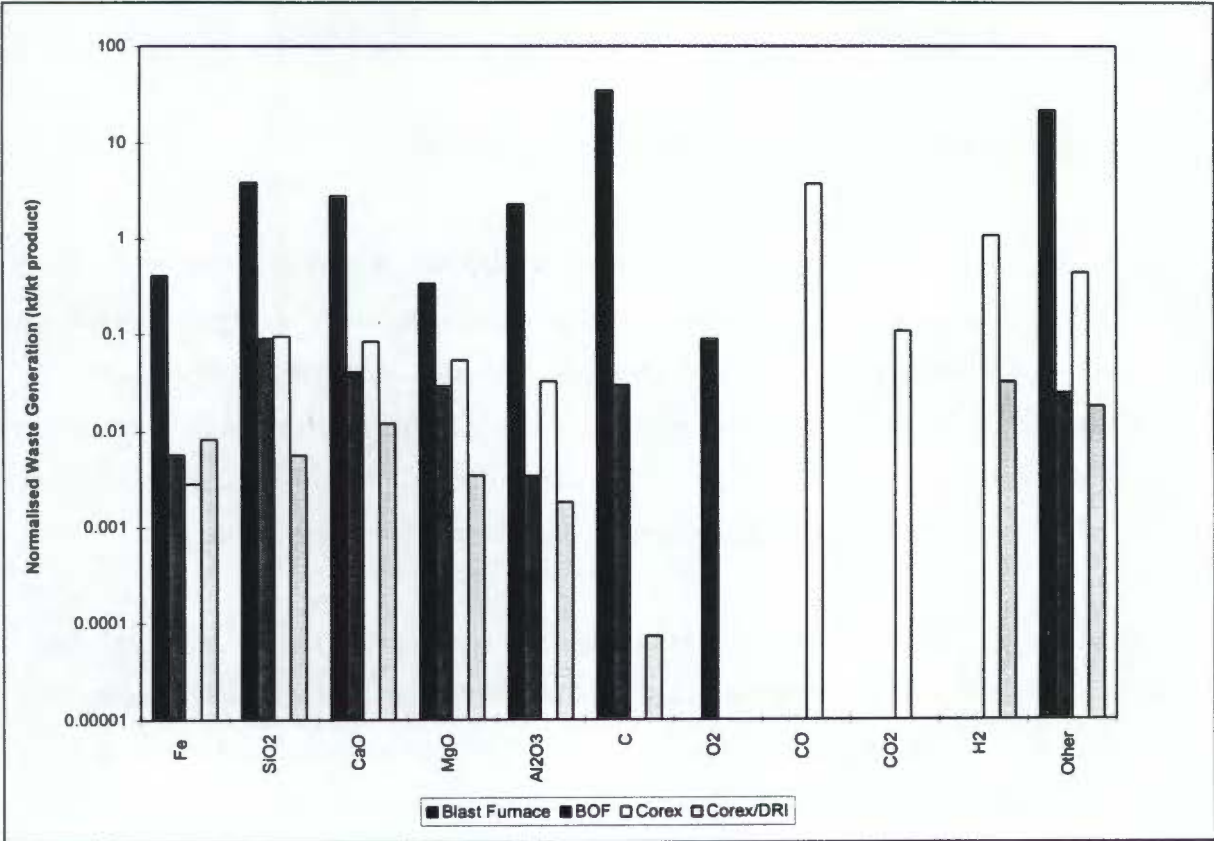


Figure A2.7-9: Change in Waste Generation with Development of Technology

The environmental performance of the blast furnace is the worst, generating the most waste per tonne of product. When the Corex process is run without a downstream DRI section, the environmental performance is also poor. Of the four technologies compared here, coupling a DRI and a Corex process gives the best environmental performance.

The total waste generated per ton of product by the four processes is included in Table A2.7-6. These figures bear out the observations made about Figure A2.7-9 and indicate the magnitude of the difference between the waste streams.

Technology	Waste Generation (kt/kt product)
Blast Furnace	66
Basic Oxygen Furnace	0.31
Corex	5.6
Corex/DRI	0.08

Table A2.7-6: Normalised waste generation potential of Steel production technologies

2.7.5 Using the Model - Building Industrial Developments

An interesting case study to investigate is that of the development of industrial nodes within South Africa. There are a number of large scale resource refining projects underway at present which are a result of the government's emphasis on "value-added" processes. A notable feature of these developments is that the projects do not operate in isolation. The process is commissioned and then further processes added on to increase the value of the product still more. An example here is the production of steel followed by the construction of a galvanising works to plate the steel.

Experience has shown that society reacts extremely negatively to these "add-on" industries. They express the opinion that the companies should put all their cards on the table when proposing the initial project. Essentially they are calling for a Strategic Impact Assessment (SIA) of the project. An SIA develops potential scenarios for industrial development associated with a major project and evaluates the potential environmental impact associated with these scenarios.

This case study shows how the waste stream changes should a galvanising plant be added to the controversial steel production plant at Saldhana on the West coast of South Africa. It is proposed that there is insufficient zinc in the South African market at present and that a new zinc electrowinning process would need to be constructed as well.

In developing this scenario the model for zinc as described in the base metals sector was linked to the Corex steel production model. Figure A2.7-10 demonstrates the change in the waste stream as the

Corex plant is linked to a DRI process (as described in section 2.7.4) and then further linked to a zinc plant for galvanising of the steel.

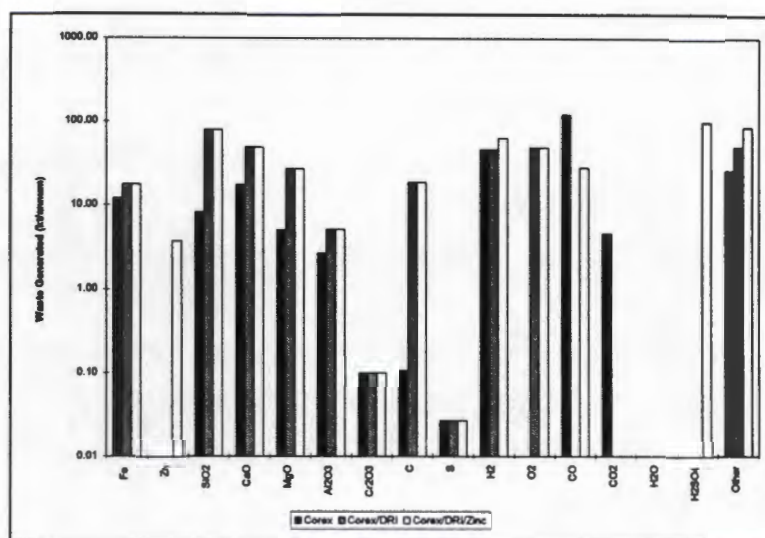


Figure A2.7-10: Change in Waste stream with Industrial Development

It is proposed that this generalised approach to flowsheet modelling can be used in developing SIAs.

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Appendix 2-8: Sub-sectoral study on the Mineral Sands Industry

2.8.1 Introduction

At present the concentration of rutile and ilmenite involves two processes, physical concentration of the titanium-rich material and pyrometallurgical processing of this concentrate. Of several obstacles to quantifying the waste stream from this industry, the largest is the confidentiality surrounding sales data for the industry. Even the values quoted by the South African Department of Mineral and Energy Affairs for the South African industry are obtained from other countries. The most recent information available on this sector is for 1991. It has been included on Table A2.8-1 to indicate South Africa's position in the world titanium market.

Country	Reserve Base		Production	
	Mt	(Rank)	kt	(Rank)
Norway	90	(1)	317	(4)
Canada	73	(2)	522	(3)
Australia	66	(3)	1155	(1)
South Africa	45	(4)	758	(2)

Table A2.8-1: Titanium reserves and production data (DMEA, 1995)

There are two main processes producing titanium in South Africa. Both have titanium slag products. For the one process there are no production figures available. However, a full Environmental Impact Assessment (EIA) was carried out for the newer process and this information has been used to infer this sector's performance. The projected production rate of the new process is 195 kt/annum.

2.8.2 Generalised flow diagram

Figure A2.8-1 shows the generalised flow diagram used to model the industry.

This figure shows only the initial concentration stages of the process. It was drawn up from a process which concentrates and then exports titanium-rich products, thus no information about smelter operations is available.

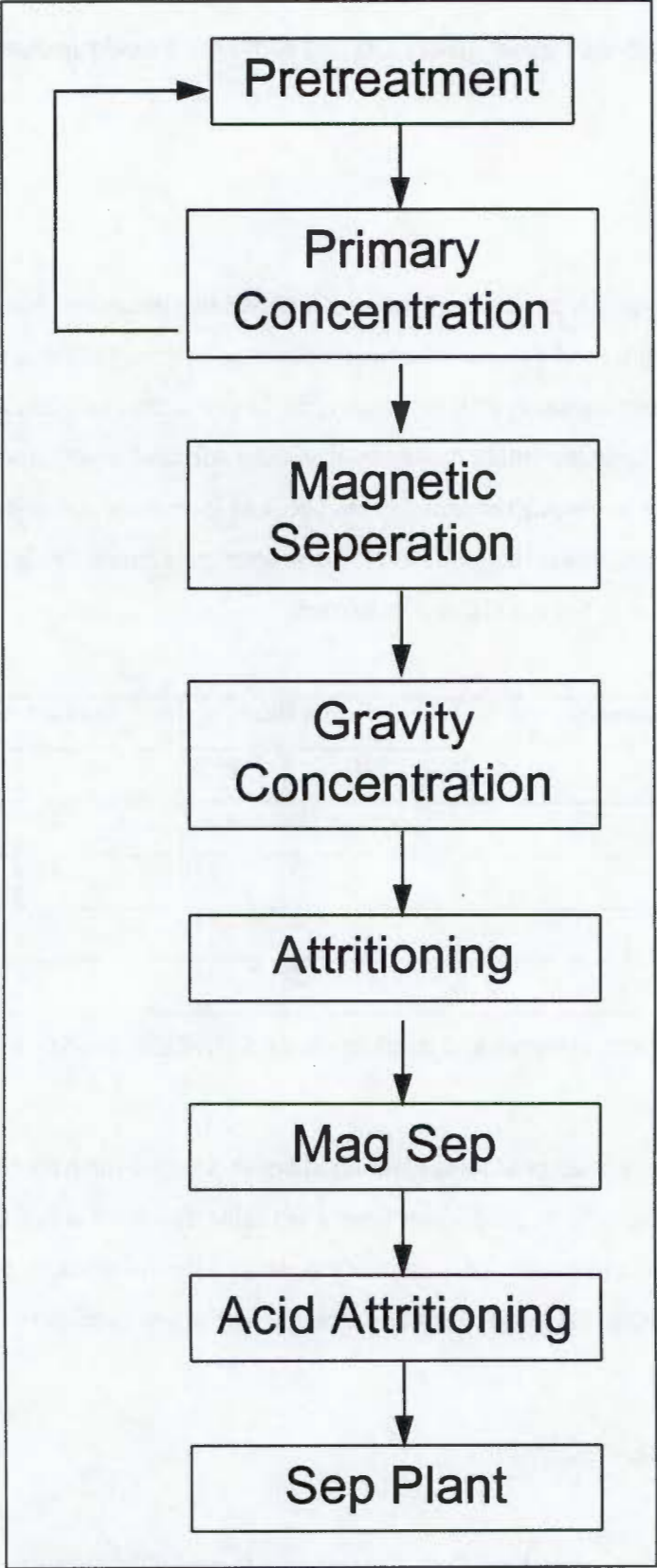


Figure A2.8-1: Generalised flow diagram for washing of Beach Sands

2.8.3 Waste Inventory

The process described by Figure A2.8-1 was modelled to obtain the waste inventory for the sub-sector. Information on the operation of the process was taken from the EIA.

2.8.3.1 Predicted Values

Table A2.8-2 shows the predicted waste stream for the South African beach sands industry. The total waste stream, most of it sand, is projected to be 20 Mt/annum.

Component	Waste Generated (kt/annum)	Generated to		
		Air	Water	Solid/Liquid
Sands	12 000			
Sea Water	7 000			
River Water	1 500			
Acid	4.0			
Total	20 500			20 500

Table A2.8-2: Waste generated by the proposed Beach Sands industry

This information is as detailed as possible within the limits of available information.

2.8.3.2 Comparison with other information

No information for the South African situation exists for comparison with this data.

2.8.3.3 Discussion

This information is as detailed as available data allows. Confidence levels placed in the various facets of the models are indicated below.

Confidence levels - Beach sands washing

Flow Sheet	90%
Assumptions	90%
Stream Splits	90%

Confidence placed in the models is high as they are contained in the EIA for the process. However, for reasons given above, modelling of the entire industry has not been possible, so these assumptions are not for the whole South African industry, but for merely a section of it.

2.8.4 Using the Model - Adaptability of Methodology

Although it is derived from plant-specific information, of all the inventories presented in this report, the inventory for this sector has the least published information for its basis. From the limited information obtainable a usable waste stream was generated. With the addition of further data, this inventory could be useful in determining the performance of the sub-sector as a whole.

This demonstrates the adaptability of the methodology employed. It can be used to model vast sectors, complex processes and processes which have multiple products. It also generates useful information from a limited set of assumptions. A further application is that by merely changing the assumptions governing the models to plant-specific, one can also change those models which are generalised to plant-specific in nature.

2.8.5 Using the Model - Application of the Methodology in the EIA process

All the information used in generating the mass balance, and thus the waste inventory, for the beach sands sub-sector of the South African minerals beneficiation industry, was taken from an EIA. For this reason a section on the level of technical information available from EIAs as a whole is included. The level of technical detail relating to process included in EIAs is, generally, either insufficient or obscure. In many cases it is difficult to interpret the process-specific information. The mass balances on the processes are often given in inconsistent units and are thus difficult to interpret.

The EIAs executed to date have extensive reports on the perceived impacts of the process. It is, however, often difficult, or well nigh impossible, to link these impacts to the points where they arise in the process. This considerably impedes the ability of interested and affected parties (I&APs), the people usually included in the scoping for the EIA, to make sensible decisions relating to the mitigation of these impacts.

It is also difficult to evaluate whether the process chosen in the first place is the best as far as environmental impact is concerned. LCA methodology is therefore proposed as a useful tool with

which to link wastes back to the unit operations generating them. Coupling of wastes to impacts by means of Life Cycle Assessment (LCA) tools would clarify which processes give rise to particular impacts and again would focus attention on the process. This would transparently reveal to I&APs which unit operations within the process are of concern and where further attention might best be focused.

As processes are the main cause of pollution, the process is where attention should be directed, especially in the initial design (and EIA) phases of a project. In the EIA process in South Africa more attention needs to be paid to the validity of process choice. The process chosen, and its applicability to the site on which it is to be placed, are among the main factors to be considered in an EIA.. This is often ignored.

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Appendix 4: Audit Trail Case Study

Appendix 4-1: Audit Trail Case Study

This appendix contains the information used to establish the mass balance used in the case study of the audit trail contained in Section 4.3.1.

The breakdown of the feed was assumed to be:

Percent	Component
50.00	% Zn (min)
30.00	% S (min)
10.00	% Fe (max)
10.00	% Balance

Other assumptions made were:

- 100 mass units per year (upa) of feed were processed
- there is 8 mass percent water in the feed

Thus the feed stream to the process is:

Component	Mass	Units
Zn	46.00	upa
S	27.60	upa
Fe	9.20	upa
H ₂ O	8.00	upa
Balance	9.20	upa
Total	100.00	upa

Three unit operations were modelled, namely Roast, Leach and Electrowin. The assumption sets governing the operation of these units are:

Roaster
Assumptions
0.15 mass units of coke are added per mass unit of concentrate
Coke composition:
80% Carbon
20% Ash
All Carbon is consumed
Stoichiometric amount of oxygen are required
Air is 20% oxygen and 80 % Nitrogen
All water is vaporised
90% of Sulphur is oxidised
Assumptions
Leach
95% of Zinc leaches
5% of Iron leaches
All the Sulphur leaches
5% of the other material entering the process leaches
No ash leaches
Solid waste is perfectly dry
Electrowin
All contaminants removed before Zn is removed
All Zn removed

As was observed in Section 4.3.1, the form of the flowsheet means that no stream split information is required.

The mass balances established for each unit operations are:

Roaster		Assumptions							
		Coke added		0.15 u coke per u Concentrate					
		Coke composition		80.00 % Carbon					
				20.00 % Ash					
				90.00 %					
		Sulphur oxidised							
		All Carbon is consumed							
		Stoichiometric amount of oxygen are required							
		Air is 20% oxygen and 80 % Nitrogen							
		All water is vaporised							
				Zn	Feed	Coke In	Air In	Gaseous Waste	To Leach
Mass In	398.96				46.00				46.00
Mass Out	398.96			S	27.60				2.76
Balance	0.00			Fe	9.20				9.20
				H ₂ O	8.00			8.00	
				C		12.00			
				Ash		3.00			3.00
				O ₂			56.79		
				N ₂			227.17	227.17	
				SO ₂				49.63	
				CO ₂				44.00	
				Balance	9.20				9.20
				Total	100.00	15.00	283.96	328.80	70.16
Mol Mass									
C	12.00	u/mol							
S	32.06	u/mol							
O	16.00	u/mol							
N	14.01	u/mol							
Zn	65.38	u/mol							
Fe	55.85	u/mol							
upa		u/mol							
S in	24.84	0.77							
C in	12.00	1.00							
thus SO ₂	49.63	0.77							
and CO ₂	44.00	1.00							
thus O ₂	56.79								
and N ₂	227.17								

Leach		Assumptions					
		Zn leached		95.00 %			
		Fe leached		5.00 %			
		S leached		100.00 %			
		Balance leached		5.00 %			
		No ash leaches					
		Solid waste is perfectly dry					
				Zn	From Roaster	Solid Waste	To Electrowin
Mass In	70.16				46.00	2.30	43.70
Mass Out	70.16			S	2.76	0.00	2.76
Balance	0.00			Fe	9.20	8.74	0.46
				H ₂ O			
				C			
				Ash	3.00	3.00	
				O ₂			
				N ₂			
				SO ₂			
				CO ₂			
				Balance	9.20	8.74	0.46
				Total	70.16	22.78	47.38

Electrowin	Assumptions			
	All contaminants removed before Zn is removed			
	All Zn removed			
Mass in	47.38			
Mass Out	47.38	From Leach	Solid Waste	Product
Balance	0.00	Zn	43.70	43.70
		S	2.76	2.76
		Fe	0.46	0.46
		H ₂ O		
		C		
		Ash		
		O ₂		
		N ₂		
		SO ₂		
		CO ₂		
		Balance	0.46	0.46
		Total	47.38	3.68
				43.70

These mass balances were used to establish the input-output matrices included in Section 4.3.1. of the thesis.

Appendix 5: Information in support of the Gold Case Study

Appendix 5-1: Environmental and Economic Information used in the design process

5.1.1 Physical and Chemical Processes for the Recovery of Gold

The recovery of gold from ores can be divided into two main categories, those which use the physical difference in properties between gold and the mineral matrix into which the gold is bound, and those that use chemical reactions to remove the gold from the ore.

5.1.1.1 Chemical Processes for the recovery of Gold

Most gold is recovered using a chemical process. This is achieved by milling the ore to a fine powder and then dissolving the gold with sodium cyanide. Most developments in the industry have been in the downstream separation of the cyanide liquid that contains the gold, from the solid material which is then sent to waste dumps. There are two main processes being used at the moment

- **Merrill-Crowe:** In this process the liquid is separated from the solid by filtration and the gold is then deposited by adding Zinc powder to the liquid, the so-called “cementation” step. The gold is then further purified.
- **Carbon-in-Pulp (CIP) or Carbon-in-Leach (CIL):** Both of these processes recover gold from solution by adsorbing it onto small carbon particles (usually burnt coco-nut shells). The gold is then washed off the particles in a process called elution, the gold is still in solution. It is removed from solution electrowinning.

Note: The difference between CIP and CIL is the point in the process at which the carbon is added to the system; in CIP carbon is added to the pulp (or mineral-water stream) after cyanide dissolution has taken place; the cyanide leach and carbon contacting cells are a single piece of equipment (or train of cells) in the case of CIL.

It is not always possible to leach (dissolve) the gold from the ore directly with cyanide - these ores are called refractory ores. What is required here is for the ore to be oxidised before it can be leached. Once the refractory ore has been oxidised it is possible to treat it using one of the

cyanide-based process routes already described. In other words, oxidation is a pre-cursor to cyanidation; it cannot replace it.

Different combinations of technologies for the cyanide-based extraction of gold are discussed in Table 5.1-1 below.

Table 5.1-1 Cyanide Dissolution Processes

Technology	Brief Description	Application
Merrill-Crowe Precipitation (Filtration)	The dissolution of gold by cyanide is followed by the filtration and clarification of the gold-rich liquid. The gold is then precipitated from the solution with zinc	Referred to as the conventional or filtration method. Has been applied to all kinds of gold ores. CIP/CIL are more efficient in recovering gold. Large scale operations.
Carbon-in-Pulp (CIP) ¹	The gold is dissolved using cyanide. This is followed by the adsorption of the gold direct from the cyanide pulp onto activated carbon. The gold is eluted or washed off the carbon particles and then electrowon from solution	Low grade ore and recovery of gold value in old tailings dams. Ores with silver to gold ratio greater than 10:1. Small and large scale operations. Where solutions are high grade (above 10-15 g/t equivalent).
Heap Leach	Gold bearing ore is heaped onto an impervious pad. It is then sprinkled with a cyanide solution to dissolve the gold. The solution is recycled to increase gold concentration. Gold is removed from solution by precipitation.	Treatment of low-grade, shallow gold deposits that are mined by low-cost, opencast methods.
In-situ Leaching	Cyanide solutions are injected into the ore-bearing strata. The gold bearing solution is then pumped to the surface, the gold is precipitated from solution using zinc.	In-situ leaching of gold has few applications as it is unreliable, this is due to monitoring and control difficulties.

5.1.1.2 Physical Process for the recovery of Gold

These processes rely on the difference in physical properties between the gold and the ore. The most common here is the mercury amalgamation of gold. In amalgamation the ore is milled and then mercury is added to the process, the gold particles attach to the mercury and, because both mercury and gold are heavy, fall to the bottom of the amalgamation vessel. It is thus possible to separate the amalgam from the ore. There are other processes such as gravity concentration and flotation, which exploit other physical properties of the gold.

5.1.2 System Parameters

The assumption sets used to establish mass balance closure for the gold processes in the generic gold flowsheet as discussed in section 2 and detailed in Appendix 2-3 are used as a starting point for the process model in this design. These values are included in 5.1.2.1. The energy input to each unit operation is calculated using design equations for the equipment, these are included in 5.1.2.2. The only input of significance is electricity, this was determined by analysing the impacts associated with the inputs to the gold industry in South Africa as presented in Appendix 2-3. The impacts associated with this energy provision are included in 5.1.2.6 below. Labour required for the plant is calculated using an heuristic equation, this is described in section 5.1.2.4. The operating costs are calculated as the sum of the costs of labour, energy and water costs for each unit operation. These are listed in 5.1.2.5. The functions used for estimating capital costs for each unit operation are included in section 5.1.2.2.

Equivalency factors have been taken from the PEMS User Manual (PEMS, 1998), these are included in section 5.1.2.7. The impact associated with the gold industry in 1996 was determined using these equivalency values. This impact profile is detailed in 5.1.2.8 along with the economic performance of the industry. These economic indicators are used in the normalisation of the objective functions derived according to the equation sets detailed in Section 4.

5.1.2.1 Mass Balance Assumption Sets

- *Comminution*

Table 5.1-2 *Comminution Assumption Set*

Crushing		
Bypassing Crusher	10.00	% of feed
Water Usage	0.10	m ³ per ton
Energy consumption	2.5	kWh/t ore
Mill		
Mass percent to mill	70	%
Iron consumed	1.5	kg/ton for ball milling
Density from 3 ^o cyclone	1.5	
Energy required	25	kWh/ton ore milled
Mass percent leaving mill	40	%

- *Cyanide Dissolution*

Table 5.1-3 *Cyanide Dissolution Assumption Set*

Cyanide Dissolution		
NaCN addition	0.24	kg/ton ore
Require lime	1.1	kg/ton ore
Undissolved Au	0.5	% undissolved
Solids Dissolving	0.5	%

- *Intensive Cyanidation*

Table 5.1-4 *Intensive Cyanidation Assumption Set*

Intensive Cyanidation		
Water to solid ratio	3	
Grade	5	g/ton
Dissolution	99.99	%
Cyanide conc	5	mass % NaCN

- *CIP and Electrowinning*

Table 5.1-5 *CIP and Electrowinning Assumption Set*

CIP, Elution and Electrowin		
Au absorbed max	99.96	%
C loss	0.22	%/day
	6.6	%/month
C	0.01	t/ton ore
Other metals on C	0.1	%
Au recov in elution and electrowin	100	%
Elute Water required	100	m ³ /ton C
Elution NaCN	18.3	kg/ton
Elution HCL	52.8	kg/ton
Elution NaOH	42.2	kg/ton
Regeneration requires	1	t steam/t C

- *Filtration, Clarification and Cementation*

Table 5.1-6 Filtration, Clarification and Cementation Assumption set

Filtration and Cement		
Wash ratio	1.15	ton per ton solid
Water in solid product	30	%
Zn	25	g/m3 soln
Nitrate	50	g/m3 soln
Au precipitating	100	%
Au in Product	35	%

- *Amalgamation*

Table 5.1-7 Amalgamation and Retorting Assumption Set

Amalgamation and Retort		
Fe removed	100	%
Amalgam contains	40	% Au by mass, rest Hg
Mass Hg lost	0.3	%
Recovery of Au in Hg	72	%
Recovery Au in Retort	100	%

5.1.2.2 Capital Cost and Energy Requirement Estimations

The functions listed here have been taken from the source *South African Costs of Equipment for the Metallurgical Industry* (Mintek, 1983) unless indicated otherwise. Figure 5.1-1 below contains an indication of the fluctuation of the inflation rate in South Africa for the years 1985 to 1996. The cumulative effect of these inflation rates renders a price multiplier of 4.7 to bring the 1983 rand costs up to 1996 values. This inflation rate is the average inflation rate for the time period illustrated.

It is assumed that the capital will be paid back over 10 years and that the interest rate charged for the capital is 15%, this is relative to a South African long bond interest rate of 14.6% in 1997 (Editors Inc, 1997). The calculations for the repayment of this interest use the equation (Brennan, 1998):

$$A = P \left[\frac{i(1+i)^n}{(1+i)^n - 1} \right]$$

where: A is the amount to be repaid per time period

P is the present value of the capital invested

i is the interest rate charged

n is the number of time periods over which repayments are made

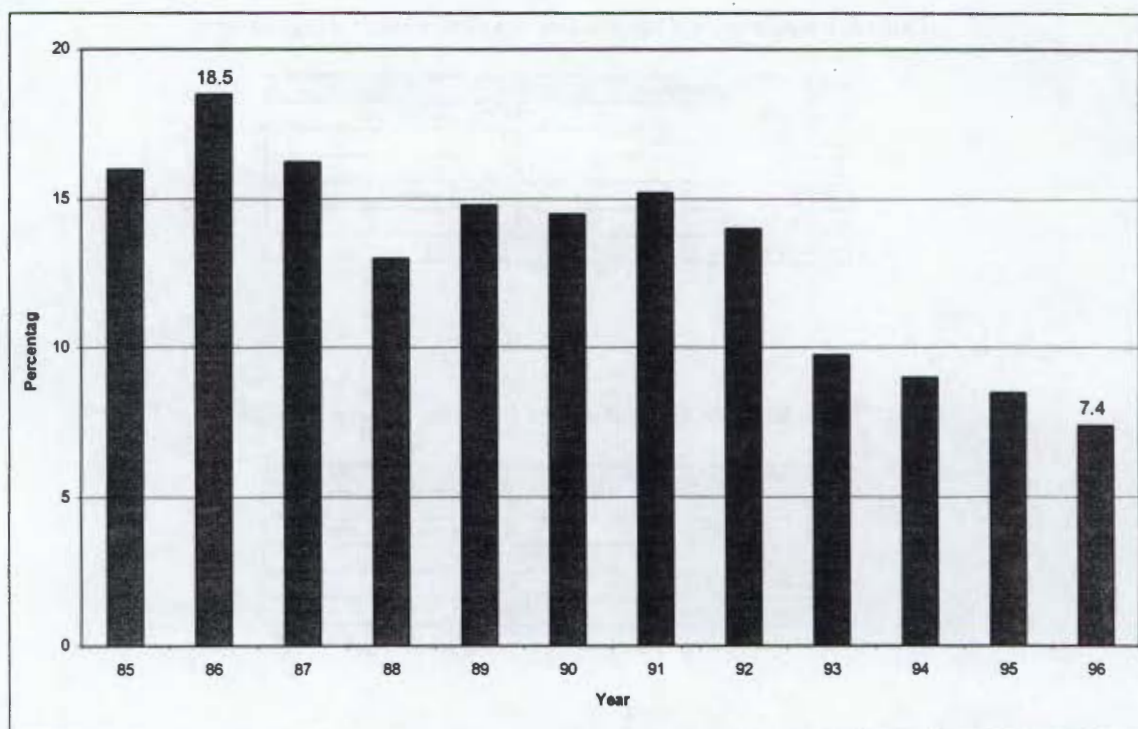


Figure 5.1-1 South African Inflation Rates - 1985 to 1996

The total profit from the process for each year is calculated from:

$$\text{Profit} = \text{Income from Sales} - \text{Costs (operating and capital)}$$

This profit then discounted to a present value relative to a "hurdle rate" which represents the required rate of return for the project. This uses the equation (after Brennan, 1998):

$$F = P[1 + h]^m$$

where:

F is the value of the profit in the future

P is the discounted value in the present

h is the discount or hurdle rate

m is the number of years in the future at which the calculation takes place

The cumulative profit is calculated for the process relative to the chosen discount rate. A positive cumulative profit implies that the process as designed will deliver at least the chosen hurdle rate of return on investment, a negative cumulative profit means that the process as designed does not meet the required return on investment. The process that returns the highest cumulative profit relative to a chosen hurdle rate is the project that will deliver the rate of return on investment.

- *Comminution*

Crushing - Impact crushers processing medium-hard rock:

$$\text{Price (R)} = 10\,737 + 342.4 \times \text{capacity (t/h)}$$

$$\text{Power (kW)} = 0.4993 \times \text{capacity (t/h)}^{1.077}$$

$$\text{Motor Price (R)} = 20\,528 + 36.33 \times \text{power (kW)}$$

Milling - Rod mill

$$\text{Price (R)} = 27\,524(D^{2.5} \times L)^{0.7160}$$

$$\text{Power (kW)} = 8.3(D^{2.5} \times L)^{1.001}$$

$$\text{Motor Price (R)} = 20\,528 + 36.33 \times \text{power (kW)}$$

$$\text{Feed for Max Power (t/day)} = 5.23(D^{2.5} \times L) + 0.312S$$

last equation from (Stanley, 1987), where S is daily feed of steel (kg). Thus assuming that the mill is operating at maximum power $D^{2.5} \times L$ can be solved using the last equation and thus the other equations can be solved.

- *Gravity Concentration*

Assume that these are Plane Tables, this equipment is usually 4 m long and can process 60 t/h per m of width (Stanley, 1987).

$$\text{Price (R)} = 2\,935 + 6301 \times \text{area (m}^2\text{)}$$

- *Cyanide Dissolution*

Mild steel pachuca with a 60° cone

$$\text{Price (R)} = 25\,492 \times e^{0.00179 \times \text{capacity (m}^3\text{)}}$$

Rule of thumb is eight tanks in series with nominal 4 hours residence time in each (Stanley, 1987).

- *Intensive Cyanidation*

Mild steel pachuca with a 60° cone

$$\text{Price (R)} = 25\,492 \times e^{0.00179 \times \text{capacity (m}^3\text{)}}$$

Residence time of 20 hours (Stanley, 1987).

- *CIP, Elution and Electrowinning*

CIP - Mild steel, cylindrical, flat-bottomed tank

$$\text{Price (R)} = 261.9 \times \text{volume}^{0.7448}$$

The following equilibrium equation has been derived (Stanley, 1987):

$$C_c - C_c(I) = k \times C_s \times t^n$$

where

C_c is the concentration of gold on carbon at time t (g/t)

$C_c(I)$ is the concentration of gold on incoming carbon (g/t)

k is the rate constant (h^{-1}); simplifying assumption of using an average rate constant derived from processing a similar ore body - 69.8 h^{-1} (Nicol *et al*, 1984).

C_s is the concentration of gold in solution at time t (g/t)

t is residence time

n is specific to the ore being processed, the simplifying assumption that $n = 0.5$ has been made for this calculation (Fleming *et al*, 1979).

Number of tanks used ranges from 4 to 10 with the most common being 6 (Bailey, 1985).

Electrowinning - Stainless Steel tank

$$\text{Price (R)} = 9639 \times e^{0.0269 \times \text{capacity (m}^3\text{)}}$$

Typical installation: through flow of 0.5 m/min with 75 cm between plates for the worst case scenario (Paul *et al*, 1983).

Elution takes place in one vessel with three cycles of reagents flowing through it.

$$\text{Price (R)} = 19\,384 + 3075 \times \text{volume}$$

Typically elution columns have a radius of 1m and a height of 1.5m to handle 90 tonnes of carbon in 50 minutes (Bailey, 1985).

- *Filtration, Clarification and Cementation*

Filtration - Complete drum filter

$$\text{Price (R)} = 23\,385 \times \text{area}^{0.4601}$$

Duty should not exceed $5 \text{ t m}^{-2} \text{ day}^{-1}$ for effective operation.

Clarification - Floc bed clarifier

$$\text{Price (R)} = 1346 \times \text{volume}^{0.5104}$$

Constant rising velocity of 0.3 m/s (Young, 1987). The height of the clarifier was taken to be 6m as there was no indication of the height and changing the height from 2m to 6m makes a R150 difference in the price of the unit when the greatest possible flowrate is sent through it. The economic objective is thus virtually unchanged by this value and thus assuming it to be 6m was deemed sufficient.

Cementation - Stainless Steel tank

$$\text{Price (R)} = 9639 \times e^{0.0269 \times \text{capacity(m}^3\text{)}}$$

Residence time is 4 hours (Stanley, 1987).

5.1.2.3 Amalgamation

1 ton in a 0.6m by 0.9m amalgam barrel for 18 hours residence time (Stanley, 1987), then a cyclone classifier.

Amalgam tank - mild steel tank

$$\text{Price (R)} = 11\,376 + 2097 \times \text{diameter (m)}$$

Classifier

$$\text{Price (R)} = 3550 \times \text{capacity(t/h)}^{0.5540}$$

- *Retorting*

Electric furnace

$$\text{Price (R)} = -126\,379 + 34\,262 \ln(\text{power(kW)})$$

Power draw 200 kW (Stanley, 1987)

5.1.2.4 Labour Requirements

Wessel made a study of the labour requirements for the American chemical processing industry (1952) which gives an indication of the number of working hours required to run each piece of equipment. Although this information is relatively old, it can still be applied in this case as operating practices in the minerals industry are fairly standard. As a feature his equation includes a "scaling factor" that relates to the operation of the unit (batch or continuous) as well as the use of multiple units.

$$\log_{10} y = 0.783 \log_{10} x + 1.252 + B$$

where y is the operating labour hours per short ton per processing step
 x is the capacity in short tons per day
 B is a constant; $B = 0.132$ for multiple units or complete batch processes; $B = 0$ for an average chemical plant; $B = -0.167$ for highly automated plants
 (1 short ton equals 907.2 kg)

5.1.2.5 Operating Costs Estimations

Operating costs are based on the total spent on water, electricity and labour for the unit operation. The following assumptions are made:

- Labour costs R24 000 per person per year (DME, 1997).
- Electricity costs 12.3 c/kWh (Eskom, 1998)
- Water costs R4.80 per m³ (R4.80 per ton) (RWB, 1999)
- Carbon costs 50 R/ton (DME, 1997)
- Zinc costs 2500 R/ton (DME, 1997)
- NaCN costs 5000 R/ton (ChemQuest, 1999)
- Lime costs 200 R/ton (DME, 1997)
- Mercury costs are negligible
- Iron costs 2000 R/ton (DME, 1997)

5.1.2.6 Impacts associated with Electricity Inputs

The environmental impacts associated with the provision of electricity from coal-based stations in South Africa has been studied by Notten (1998). The contributions of this electricity provision to the impact categories selected for this study are listed in the table below. These values are relative to a functional unit of 1MW.

Table 5.1-8 *Contribution of Electricity generation to environmental impacts*

Impact Category	Amount	Units
Greenhouse	660	ton CO ₂
Acidification	4.0	ton SO ₂
Human toxicity	0.77	ton toxic substances
Eutrophication	0.20	ton Phosphate
Landfill Volume	6000	m ³
Water Consumption	6000	litres

5.1.2.7 Equivalency Factors

The table below includes the equivalency values which will be used to link wastes to potential impacts (PEMS, 1998).

Component	Greenhouse ton CO ₂ per ton	Acidification ton SO ₂ per ton	Human Toxicity ton / ton	Eutrophication ton Phosphate/ton	Landfill Volume m ³ per ton	Water Consumption l/ton
Ore	0	0	0	0	1.5	0
Water	0	0	0	0	1	1000
Fe	0	0	0.48	0	7	0
S	0	64.01	0	0	2	0
Hg	0	0	120	0	13.6	0
Au	0	0	0	0	17.2	0
NaCN	0	0	0.22	0	0	0
CaO	0	0	0	0	0	0
Diss Salts	0	0	0.003	0	0	0
Zn	0	0	0.003	0	7	0
Nitrate	0	0	0.0008	0.1	0	0
NaOH	0	0	0	0	0	0
H ₂ SO ₄	0	64.01	0.038	0	0	0
Carbon	0	0	0	0	2	0
HCl	0	0.88	0.038	0	0	0
Steam	0	0	0	0	0	0
Ag	0	0	0	0	10	0
Other Metals	0	0	0.003	0	7	0

* calculated relative to the potential number of H⁺ ions which would be associated with the component

5.1.2.8 Baseline Gold Industry Profile

The waste, input and product inventories for the gold industry for 1996 are included in Appendix 2-3 of this thesis. The performance of the industry will form the basis for normalisation for this case study as discussed in section 4 of the thesis. Thus the total contribution of the gold industry to the impact categories chosen must be evaluated. Annual contributions have been evaluated, these calculations include impacts associated with energy provision:

- Greenhouse Impact - 165700 ton CO₂
- Acidification effect - 7452000 ton SO₂
- Human toxicity - 54960 ton/ ton relative to total daily intake
- Eutrophication - 50 ton phosphate
- Landfill volume - 221600000 m³
- Water usage - 267600000000 liters
- "Profit" for the industry - 3 000 R million from:
 - Sales value of products for the year - 27 000 R million (COM, 1997)
 - Capital expenditure less recoupments - 3000 R million (COM, 1997)
 - Operating costs - 21 000 R million (COM, 1997)
- Working profit per ton milled is 31 R/ton (COM, 1997)

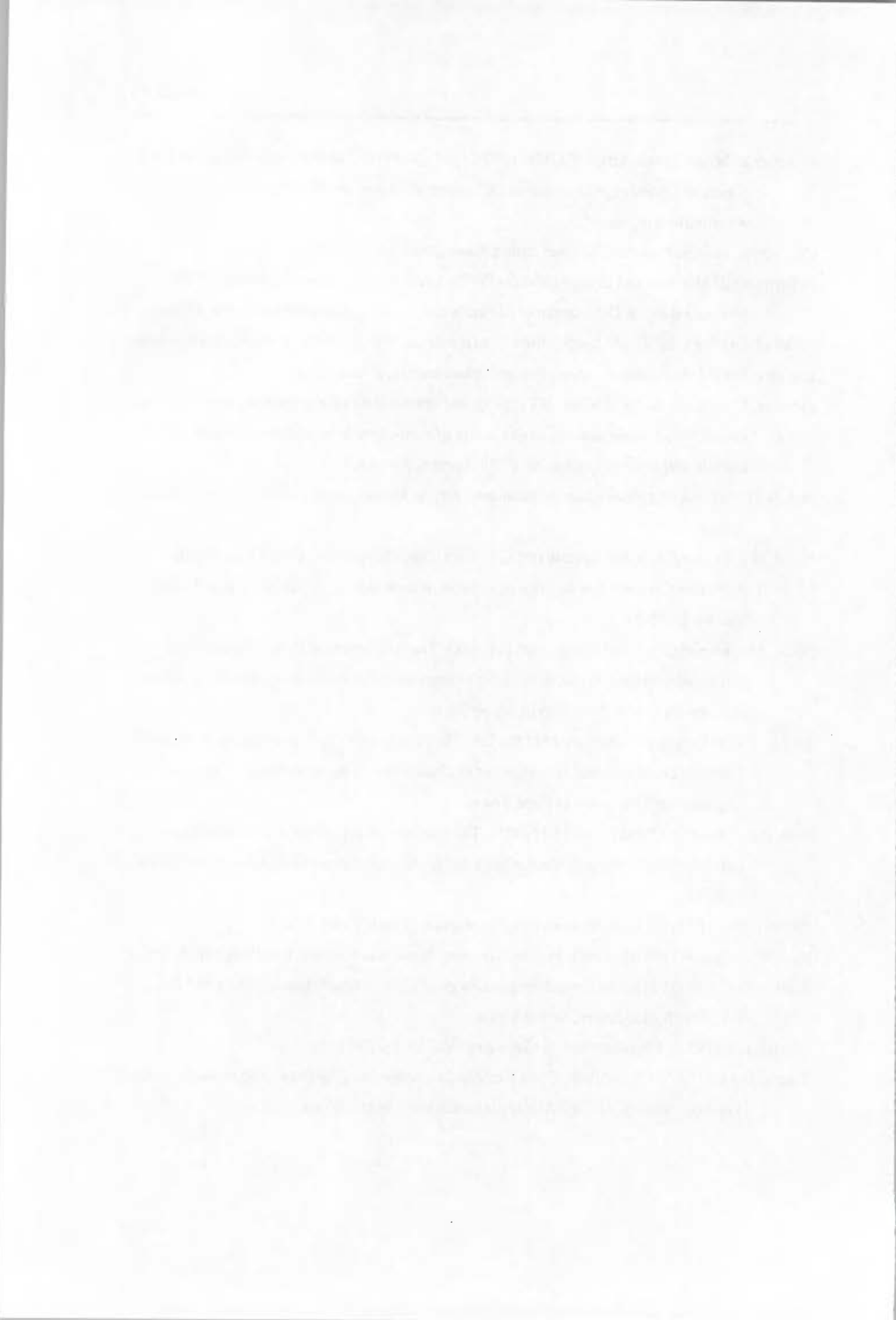
5.1.2.9 Assumptions for burning carbon to provide energy for retorting

All of the carbon is burned to CO₂. The only equivalency factors for CO₂ that is non-zero is its contribution to greenhouse gases where it is the reference material and thus the equivalency factor is 1. Carbon used in the process is sourced from a renewable resource. The cost of the amalgamation equipment required remains the same. Burning carbon to recover gold eliminates the need for elution and electrowinning processes.

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Appendix 5-2: Impact Profile of the South African Minerals Industry

This appendix contains details of the total waste stream from the South African minerals industry. This is an aggregation of the information contained in Appendices 2-3 to 2-9. These wastes have been linked to impacts using the equivalency table included in this appendix. The total impact profile for the industry is also included in this appendix.

5.2.1 Waste streams from the South African Minerals Industry

Details of the waste stream from the gold has not been included here as it has already been evaluated during the case study.

5.2.1.1 Coal waste stream

Coal	kt/annum
Coal	48000
Water	5700
Magnetite	45

5.2.1.2 Platinum Group Metals waste stream

PGMs	kt/annum
H ₂ O	47000
Merensky	15000
UG-2	13000
Cr ₂ O ₃	5700
Fe	270
Air	59
SiO ₂	42
O ₂	42
S	30
Various Reagents	30
H ₂ SO ₄	19
MgO	16
CaO	13
Ni	13
Cu	13
Cr	6.0
Co	0.19
PGMs	0.03

5.2.1.3 Base Metals waste stream

Base Metals	kt/annum
Water	30000
Ore	16000
Air	2600
CO ₂	570
CO	320
Oxygen	240
Silica Flux	145
Cu	100
Coal (Carbon in slag)	90
P ₂ O ₅	71
Reagents	67
H ₂ SO ₄	58
SO ₂	54
Steam	32
Various Reagents	24
Fluorine	18
SO ₃	15
Al ₂ O ₃	13
Fe ₂ O ₃	9.4
Fe	6.0
Nitric Acid	6.0
Al	3.3
Zn	2.9
Pb	2.1
TiO ₂	1.6
Zirconium	1.5
Magnetite	0.65

5.2.1.4 Mineral Sands waste stream

Sands	kt/annum
Sands	12000
Sea Water	7000
River Water	1500
Acid	4.0

5.2.1.5 Ferro-Alloys waste stream

FeAlloy	kt/annum
Water	9600
Ore	9200
Coal	6300
SiO ₂	3800
Other	2500
Fe	1800
CaO	1700
CO	1300
MgO	1000
Al ₂ O ₃	730
MnO	570
Mn	550
TiO ₂	360
CO ₂	320
H ₂	310
Ash	270
O ₂	270
Wash Solution	69
FeO	55
Cr ₂ O ₃	11
V ₂ O ₅	10
Cl ⁻	10
Na ⁺	7.9
SO ₂	7.0
Air	5.8
HCl	5.4
VO ₃ ⁻	5.2
NH ₃	5.0
NaVO ₃	2.2
S	0.16

5.2.2 Equivalencies

Details of the equivalency values for the gold sub-sector have not been included here as they are included in Appendix 5-1. All equivalency values are according to PEMS (1998).

5.2.2.1 Coal

Component	Greenhouse ton/ton CO ₂	Acidification ton/ton SO ₂	Human Toxicity ton/ton	Eutrophication ton/ton Phosphate	Landfill Volume m ³ /ton	Water Consumption l/ton
Coal	0	0	0	0	0.6	0
Water	0	0	0	0	0	1000
Magnetite	0	0	0	0	2.6	0

5.2.2.2 PGMs

Component	Greenhouse ton/ton CO ₂	Acidification ton/ton SO ₂	Human Toxicity ton/ton	Eutrophication ton/ton Phosphate	Landfill Volume m ³ /ton	Water Consumption l/ton
H ₂ O	0	0	0	0	0	1000
Merensky	0	0	0	0	1.5	0
UG-2	0	0	0	0	1.5	0
Cr ₂ O ₃	0	0	6.7	0	7	0
Fe	0	0	0.48	0	7	0
Air	0	0	0	0	0	0
SiO ₂	0	0	0	0	0	0
O ₂	0	0	0	0	0	0
S	0	64.01	0	0	2	0
Various Reagents	0	0	0	0	0	0
H ₂ SO ₄	0	64.01	0.038	0	0	0
MgO	0	0	0	0	0	0
CaO	0	0	0	0	0	0
Ni	0	0	470	0	7	0
Cu	0	0	0.24	0	7	0
Cr	0	0	6.7	0	7	0
Co	0	0	24	0	7	0
PGM	0	0	0.79	0	7	0

5.2.2.3 Base Metals

Component	Greenhouse ton/ton CO ₂	Acidification ton/ton SO ₂	Human Toxicity ton/ton	Eutrophication ton/ton Phosphate	Landfill Volume m ³ /ton	Water Consumption l/ton
Water	0	0	0	0	0	1000
Ore	0	0	0	0	1.5	0
Air	0	0	0	0	0	0
CO ₂	1	0	0	0	0	0
CO	0	0	0.012	0	0	0
Oxygen	0	0	0	0	0	0
Silica Flux	0	0	0	0	0	0
Cu	0	0	0.24	0	7	0
Coal (Carbon in sla	0	0	0	0	0.6	0
P ₂ O ₅	0	0	0.0004	3.06	0	0
Reagents	0	0	0	0	0	0
H ₂ SO ₄	0	64.01	0.038	0	0	0
SO ₂	0	1	1.2	0	0	0
Steam	0	0	0	0	0	0
Various Reagents	0	0	0	0	0	0
Fluorine	0	1.6	0.041	0	1	0
SO ₃	0	1	0.038	0	0	0
Al ₂ O ₃	0	0	0	0	3	0
Fe ₂ O ₃	0	0	0.48	0	7	0
Fe	0	0	0.48	0	7	0
Nitric Acid	0	1	0.022	0.1	0	0
Al	0	0	0	0	3	0
Zn	0	0	0.033	0	3	0
Pb	0	0	160	0	3	0
TiO ₂	0	0	0	0	2.6	0
Zirconium	0	0	0	0	2.6	0
Magnetite	0	0	0	0	2.6	0

5.2.2.4 Ferro-Alloys

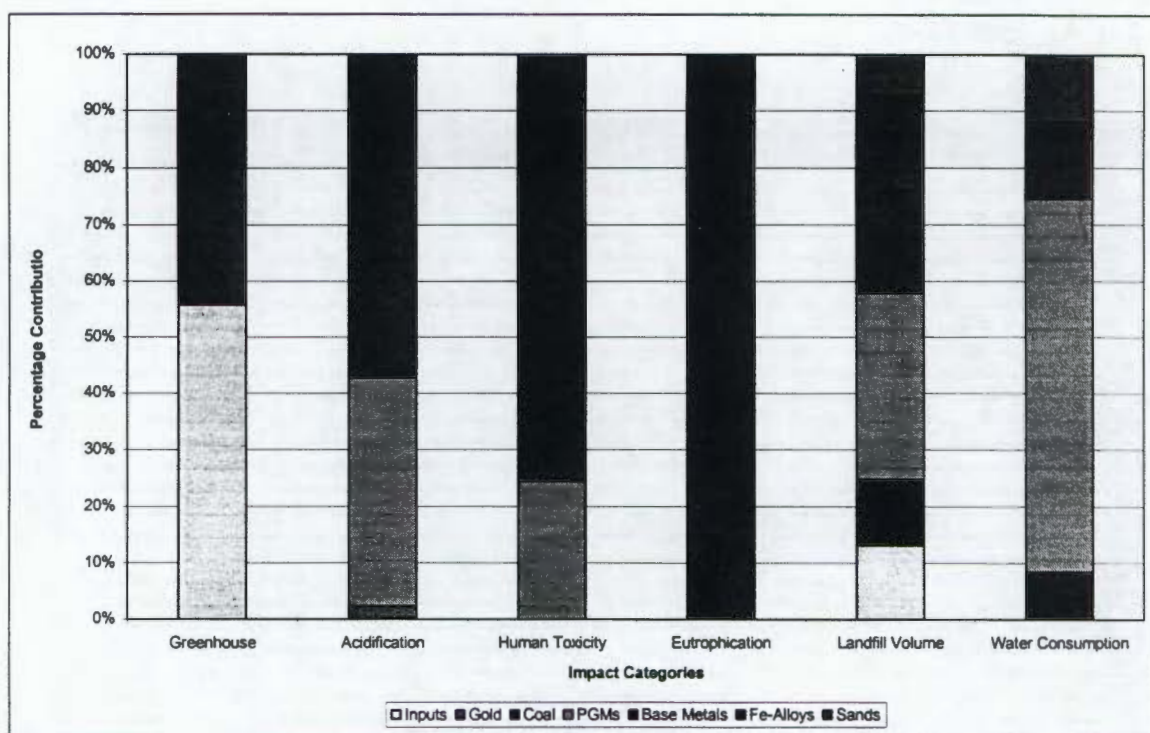
Component	Greenhouse ton/ton CO ₂	Acidification ton/ton SO ₂	Human Toxicity ton/ton	Eutrophication ton/ton Phosphate	Landfill Volume m ³ /ton	Water Consumption l/ton
Water	0	0	0	0	0	1000
Ore	0	0	0	0	1.5	0
Coal	0	0	0	0	0.6	0
SiO ₂	0	0	0	0	0	0
Other	0	0	0	0	1.5	0
Fe	0	0	0.48	0	7	0
CaO	0	0	0	0	0	0
CO	0	0	0.012	0	0	0
MgO	0	0	0	0	0	0
Al ₂ O ₃	0	0	0	0	3	0
MnO	0	0	120	0	3	0
Mn	0	0	120	0	6	0
TiO ₂	0	0	0	0	2.6	0
CO ₂	1	0	0	0	0	0
H ₂	0	0	0	0	0	0
Ash	0	0	0	0	0.6	0
O ₂	0	0	0	0	0	0
Wash Solution	0	0	0	0	0	0
FeO	0	0	0.48	0	7	0
Cr ₂ O ₃	0	0	6.7	0	7	0
V ₂ O ₅	0	0	120	0	6	0
Cl ⁻	0	0.88	0	0	0	0
Na ⁺	0	0	0	0	6	0
SO ₂	0	1	1.2	0	0	0
Air	0	0	0	0	0	0
HCl	0	0.88	0.038	0	0	0
VO ₃ ⁻	0	0	120	0	6	0
NH ₃	0	1.88	0.02	0.35	0	0
NaVO ₃	0	0	0	0	6	0
S	0	64.01	0	0	2	0

5.2.2.5 Mineral Sands

Component	Greenhouse ton/ton CO ₂	Acidification ton/ton SO ₂	Human Toxicity ton/ton	Eutrophication ton/ton Phosphate	Landfill Volume m ³ /ton	Water Consumption l/ton
Sands	0	0	0	0	1.5	0
Sea Water	0	0	0	0	0	1000
River Water	0	0	0	0	0	1000
Acid	0	0.88	0.038	0	0	0

5.2.3 Impacts Associated with the Minerals Industry

The contribution of each sub-sector of the South African Minerals Industry to the total impacts from the industry are detailed in the figure below. These total contributions follow in a table.



	Greenhouse ton CO ₂	Acidification ton SO ₂	Human Toxicity ton	Eutrophication ton Phosphate	Landfill Volume m ³	Water Consumption l
Inputs	3600000	22000	4200	1100	33000000	33000000
Gold		160000	420	1.5	610000	270000000
Coal					28000000	5700000000
PGMs		3100000	44000000		84000000	47000000000
Base Metals	2600000	4400000	590000	280000	46000000	
Fe-Alloys	320000	40000	14000000	1800	43000000	9600000000
Sands					18000000	8500000000
Total	6520000	7722000	184594620	282901.5	252610000	71103000000

5.2.4 Total Sales from the Industry

The table below gives an indication of the total sales from the industry, both mass and value.

	Sales Value (R 000 000/annum)	Sales Mass (kt/annum)
Gold	27 000	0.5
Coal	15 000	200 000
PGMs	7 600	0.2
Base Metals	1 900	3 600
Ferro-Alloys	9 300	92 000
Mineral Sands	n/a	760
Totals	60 800	296 361

n/a = not available

References

PEMS (1998); *PEMS.4 User Manual*; Pira International; Leatherhead, UK.



Appendix 5-3: Scores achieved by technology permutations for different weighting regimes

This appendix contains details of the scores achieved by each of the recycle technology permutations for different weighting sets.

5.3.1 Changing relative environmental and economic weightings

Number	Description	Initial Weighting Set	All Impact Categories x2	Green house x2	Human Toxicity x2	Economic Objective x2
A	Small-scale Intensive Cyanidation with central re-processing of tails through CIP	197.67	215.76	198.72	202.34	379.65
B	Small-scale Intensive Cyanidation with central re-processing of tails through	336.02	401.35	337.66	341.62	610.78
C	Large-scale Intensive Cyanidation and recycling of Tails, CIP separation	11.40	35.67	15.14	15.33	-1.35
D	Large-scale Intensive Cyanidation and recycling of Tails, Filtration separation	124.68	197.03	129.38	129.44	178.51
E	Small-scale Amalgamation with central re-processing of tails through CIP	252.42	270.24	253.69	257.29	490.09
F	Small-scale Amalgamation with central re-processing of tails through Filtration	311.22	358.81	312.73	316.36	578.62
G	Large-scale Amalgamation with reprocessing of tails through CIP	-0.52	23.16	3.15	3.34	-24.74
H	Large-scale Amalgamation with reprocessing of tails through Filtration	46.49	100.23	50.52	50.58	39.81
I	Small-scale Cyanide Dissolution and CIP	474.72	493.49	476.88	480.50	936.42
J	Small-scale Cyanide Dissolution and	598.93	654.26	601.65	605.79	1149.82
K	Large-scale Cyanide Dissolution and CIP	-11.31	12.42	-7.67	-7.49	-46.50
L	Large-scale Cyanide Dissolution and	56.18	117.54	60.79	60.85	51.67

5.3.2 Changing relative environmental and economic weightings to zero - equivalent to adjusting the objective function

Number	Description	Initial Objective	Environmental Objective only	Economic Objective only	Labour Objective only	Combined Objectives
A	Small-scale Intensive Cyanidation with central re-processing of tails through CIP	197.67	17.29	181.18	-0.48	197.99
B	Small-scale Intensive Cyanidation with central re-processing of tails through	336.02	63.97	273.40	-0.43	336.94
C	Large-scale Intensive Cyanidation and recycling of Tails, CIP separation	11.40	24.23	-12.79	-0.49	10.95
D	Large-scale Intensive Cyanidation and recycling of Tails, Filtration separation	124.68	71.85	53.33	-0.45	124.74
E	Small-scale Amalgamation with central re processing of tails through CIP	252.42	16.79	236.65	-0.39	253.05
F	Small-scale Amalgamation with central re processing of tails through Filtration	311.22	46.33	266.14	-0.38	312.10
G	Large-scale Amalgamation with reprocessing of tails through CIP	-0.52	23.69	-24.22	-0.40	-0.93
H	Large-scale Amalgamation with reprocessing of tails through Filtration	46.49	53.55	-6.87	-0.39	46.29
I	Small-scale Cyanide Dissolution and CIP	474.72	16.86	459.78	-0.25	476.39
J	Small-scale Cyanide Dissolution and	598.93	52.90	548.46	-0.26	601.10
K	Large-scale Cyanide Dissolution and CIP	-11.31	23.78	-35.14	-0.26	-11.62
L	Large-scale Cyanide Dissolution and	56.18	61.14	-4.74	-0.27	56.13

5.3.3 Changing relative environmental and economic weightings to reflect changes in priority relative to global, regional and local impacts

Number	Description	Initial Weighting Set	Greenhouse x5	Greenhouse only Environmental Objective	Acidification x5	Acidification only Environmental Objective	Toxicity and Eutrophication x5	Toxicity and Eutrophication only Environmental Objectives
A	Small-scale Intensive Cyanidation with central re-processing of tails through CIP	197.67	199.46	181.43	241.46	101.93	214.95	185.30
B	Small-scale Intensive Cyanidation with central re-processing of tails through Filtration	336.02	338.50	273.68	386.12	285.59	533.29	322.38
C	Large-scale Intensive Cyanidation and recycling of Tails, CIP separation	11.40	26.24	-9.09	54.44	-2.04	41.79	-5.20
D	Large-scale Intensive Cyanidation and recycling of Tails, Filtration separation	124.68	141.98	57.53	173.93	65.52	336.82	106.24
E	Small-scale Amalgamation with central re-processing of tails through CIP	252.42	254.42	236.89	296.14	247.32	269.82	240.74
F	Small-scale Amalgamation with central re-processing of tails through Filtration	311.22	313.50	266.40	357.13	277.31	444.43	299.13
G	Large-scale Amalgamation with reprocessing of tails through CIP	-0.52	14.17	-20.54	42.17	-13.54	29.62	-16.66
H	Large-scale Amalgamation with reprocessing of tails through Filtration	46.49	62.06	-3.02	91.33	4.29	193.04	29.72
I	Small-scale Cyanide Dissolution and CIP	474.72	477.62	480.03	519.50	470.50	493.06	463.89
J	Small-scale Cyanide Dissolution and Filtration	598.93	602.53	548.75	652.33	561.20	752.04	586.13
K	Large-scale Cyanide Dissolution and CIP	-11.31	3.40	-31.45	31.51	-24.42	18.89	-27.58
L	Large-scale Cyanide Dissolution and Filtration	56.18	73.97	-0.35	107.36	8.01	223.53	37.04

5.3.4 Scaling Criteria Scores for Technology Permutations

5.3.4.1 Range of Criteria Scores

Number	Greenhouse	Acidification	Human Toxicity	Eutrophication	Landfill Volume	Water Consumption	Labour	Cummulative Profit
A	0.25	10.75	3.87	0.25	1.53	0.64	0.48	-7.25
B	0.28	12.19	4.24	44.74	1.78	0.75	0.43	-10.94
C	3.70	10.75	3.89	3.70	1.56	0.64	0.49	0.51
D	4.20	12.19	4.26	48.65	1.80	0.75	0.45	-2.13
E	0.25	10.67	3.85	0.25	1.32	0.46	0.39	-9.47
F	0.26	11.16	3.89	29.10	1.41	0.51	0.38	-10.65
G	3.68	10.67	3.86	3.68	1.34	0.46	0.40	0.97
H	3.85	11.16	3.90	32.69	1.44	0.51	0.39	0.27
I	0.25	10.72	3.86	0.25	1.32	0.47	0.25	-18.39
J	0.29	12.74	4.43	33.24	1.61	0.59	0.26	-21.94
K	3.69	10.72	3.87	3.69	1.35	0.47	0.26	1.41
L	4.39	12.74	4.45	37.33	1.64	0.59	0.27	0.19

5.3.4.2 Best and Worst Criteria Scores

Number	Greenhouse	Acidification	Human Toxicity	Eutrophication	Landfill Volume	Water Consumption	Labour	Cummulative Profit
Best	0.25	10.67	3.85	0.25	1.32	0.46	0.25	1.41
Worst	4.39	12.74	4.45	48.65	1.80	0.75	0.49	-21.94
Range	4.15	2.07	0.60	48.41	0.48	0.29	0.24	23.34

5.3.4.3 Scaled Scores for Technology permutations

Number	Description	Initial Weighting Set	All Impact Categories x2	Green house x2	Human Toxicity x2	Economic Objective x2
A	Small-scale Intensive Cyanidation with central re-processing of tails through CIP	1.49	2.62	1.50	1.54	1.87
B	Small-scale Intensive Cyanidation with central re-processing of tails through Filtration	5.31	9.57	5.32	5.97	5.31
C	Large-scale Intensive Cyanidation and recycling of Tails, CIP separation	2.14	4.23	2.97	2.20	2.17
D	Large-scale Intensive Cyanidation and recycling of Tails, Filtration separation	5.51	10.88	6.47	6.19	5.67
E	Small-scale Amalgamation with central re-processing of tails through CIP	0.47	0.47	0.47	0.47	0.93
F	Small-scale Amalgamation with central re-processing of tails through Filtration	1.78	3.04	1.78	1.84	2.29
G	Large-scale Amalgamation with reprocessing of tails through CIP	0.99	1.95	1.81	1.01	1.00
H	Large-scale Amalgamation with reprocessing of tails through Filtration	2.32	4.59	3.19	2.41	2.37
I	Small-scale Cyanide Dissolution and CIP	0.90	0.96	0.90	0.92	1.75
J	Small-scale Cyanide Dissolution and Filtration	4.69	8.39	4.70	5.67	5.69
K	Large-scale Cyanide Dissolution and CIP	1.03	2.05	1.86	1.07	1.03
L	Large-scale Cyanide Dissolution and Filtration	4.90	9.75	5.90	5.90	4.95

Appendix 6: Information in support of the Zinc Case Study

Appendix 6-1: Environmental and Economic Information used in the case study

6.1.1 Processes for the refining of Zinc

The recovery of zinc from concentrates is based on the oxidation of the sulphide minerals followed by reduction of the zinc to metallic form. Historically this was done by pyrometallurgical means in retort furnaces. These were replaced around the middle of this century by roasting, sulphuric acid leaching and electrowinning of the zinc metal. The main reason for switching to the roast-leach process is the higher recovery of zinc in these processes (96% compared to 85%) and the environmental and health related problems associated with the old retort technology. However, there are also environmental issues associated with the leaching processes. Primary amongst these is the need to store large quantities of iron residues (jarosite, goethite and hematite). These are the solid residues from the leach processes. They can present an environmental problem because of the trace quantities of heavy metals they contain (in particular zinc, cadmium, copper, nickel and arsenic).

This sections gives background information on the different technologies available for the refining of zinc. Section 6 of this thesis contains a summary of the information presented below.

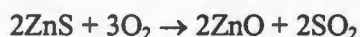
6.1.1.1 Roast-Leach-Electrowin (RLE)

This section is drawn from work that was carried out for African Environmental Solutions by the author of this thesis (AES, 1997). In this chapter the proposed zinc refinery and phosphoric acid plant is described. The description of the process, activities, dimensions, phasing and employment opportunities, is required to enable a proper assessment of likely impacts. The RLE process is illustrated on Figure 6.1-1.

Roaster

- The blended zinc concentrate is fed to the Roaster. The zinc in the concentrate is predominantly zinc sulphide. In this form it is not leachable with sulphuric acid. It is

therefore air-roasted (at $\pm 950^{\circ}\text{C}$) to produce a calcine. Zinc is predominantly present in the calcine as zinc oxide. The sulphur, from the zinc sulphide, reacts with the oxygen in the air to form sulphur dioxide (SO_2). The roasting reaction is exothermic, *i.e.* generates heat. This heat is sufficient to maintain the roasting reaction but fuel is required to start up the plant. The heat from the reaction can be used to generate steam for use in other sections of the plant. Dominant reactions in the roaster are (van Niekerk and Begley, 1991):



There are a number of side reactions that also occur, most common of these is the reaction of Zinc oxide with SO_2 :

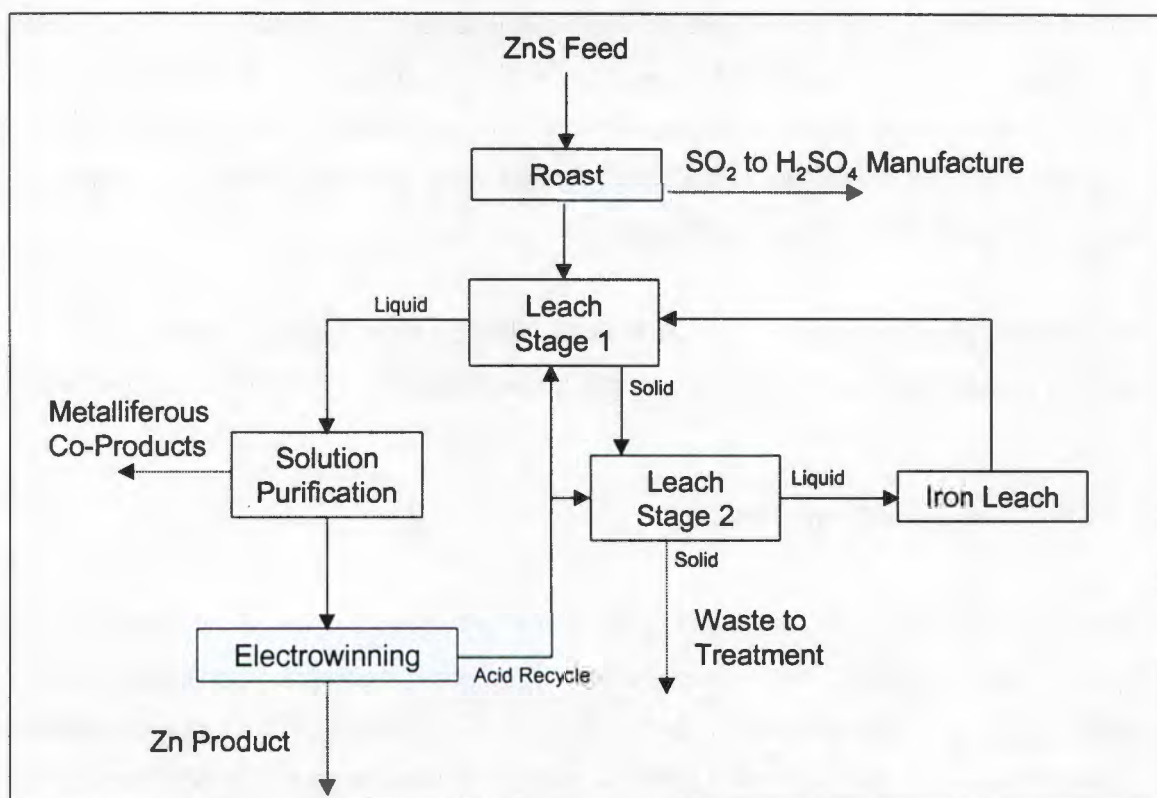
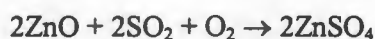


Figure 6.1-1 Simplified Roast-Leach-Electrowin Process

The off-gas from the roaster can contain dust which is usually recovered and added back into the process at the first stage of leaching. There is the potential for roasting to liberate the

more volatile metals in the feed material - most significant of these are arsenic and selenium. In addition, other volatile metals (cadmium, silver and lead) precipitate out on the dust.

Leach

- The calcine produced in the roaster plant is leached using sulphuric acid. The metals dissolve into solution in a three-stage leach process. These are leach stage 1 (also referred to as neutral leaching), leach stage 2 (neutral leach 2) and the iron leach. In leach stage 1 the calcine is reacted with acid for the first time. The solution from this stage is the raw zinc sulphate solution which reports to the electrowinning circuit. The solid residue from leach stage one reports to leach stage 2 where more acid is added. In this stage more of the zinc is dissolved, but so is more of the iron. The solid residue from this second leach is the major solid waste stream from the process. The nature of this waste depends, to a significant extent, on the leach stage which follows the second leach, viz., the iron leach. In the iron leach stage the pH of the solution is altered in order to oxidise iron and precipitate it in such a form that it cannot be dissolved when the stream is recirculated back to the first leaching stage. Typically the iron is precipitated as a jarosite (Woollacott and Eric, 1984; Habashi, 1993). However, the iron can also be precipitated as goethite or hematite.

Solution Purification

- The solution from the leaching circuit is purified in order to maintain the efficiency of the zinc electrowinning process. In solution purification the following components are removed:
 - Copper
 - Cobalt
 - Cadmium
 - Nickel

Other contaminants may be present.

The first process developed for the purification of the leach solution was installed at the Kokkola Zinc Refinery in Finland and ran from 1970 to 1974 (Fugleberg *et al*, 1993). This was a two-stage purification process which produced a copper precipitate and a cadmium precipitate. In 1975 the process was extended to a three-stage process producing a copper cake, a cobalt cake and a cadmium cake which can be further processed to produce cadmium metal. Within the design process undertaken a three-stage purification section will be used.

Electrowinning

- The cooled pure zinc sulphate solution is subjected to an electrical process called electrowinning. This results in zinc metal being deposited on cathodes and sulphuric acid being regenerated. The sulphuric acid is recycled to the leach processes. The process is thus self-sustaining with respect to acid requirements.

Casting of Zinc into Ingots

- The deposited zinc metal, which is 99,995% pure, is stripped from the cathodes, melted in induction furnaces and then cast into ingots.

6.1.1.2 Pressure Leach

Pressure leaching of Zinc has been used with two main applications:

- As a stand alone process
- In combination with roast-leach-electrowin (RLE) plants

There are three industrial examples of the former, i.e., Cominco at Trail in British Columbia (commissioned in 1981); Kidd Creek mine in Timmins Ontario (1983) and Ruhr-Zink in Datteln Germany (1991). The only industrial application of the latter is at Hudson Bay in Canada (1993) which processes 520 t per day (or 190 000 tpa). A schematic of the process at Hudson Bay is included in Figure 6.1-2. The start-up period for this plant was 10 weeks in total (Collins *et al*, 1994). Commissioning the pressure leach process decreased the output of SO₂ to the atmosphere by 89 000 tpa.

The main reason for installing the pressure leach in combination with RLE processes is so that production could be increased without either:

- Increasing SO₂ effluents
- Requiring increased capacity on a linked sulphuric acid plant

There are different iron rejection characteristics at each plant which requires customisation of the facilities to the ores being processed (Collins *et al*, 1994). Hudson Bay is the first plant which accepts 100% zinc concentrate in the feed to the autoclave, the other plants listed above accept a

combination of concentrate and tails from the atmospheric leach of ZnO. The reaction taking place in the pressure leach is:

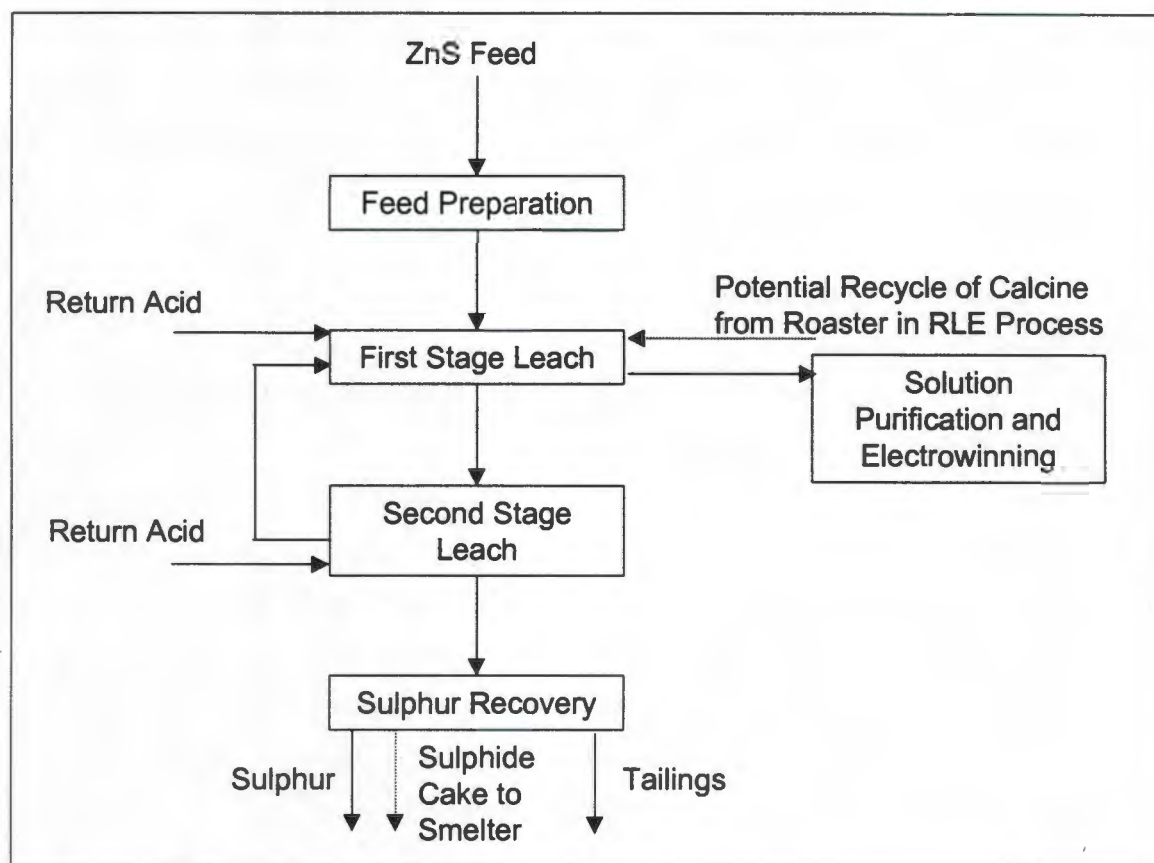
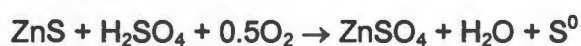
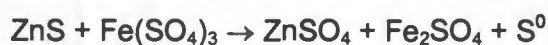
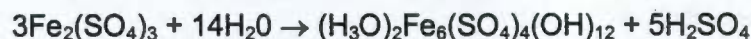


Figure 6.1-2 Flow sheet for 2-stage pressure leach process (after Collins et al, 1994)

This reaction is extremely slow in the absence of a species to facilitate oxygen transfer. One species that can facilitate this transfer is dissolved iron. The reaction is then the sum of:



There is usually sufficient acid soluble Fe in the concentrate to supply the requirements of the leach reactions listed above. Once this reaction has proceeded to the point where there is insufficient free acid to maintain Fe in solution, Fe will precipitate as a mixture of hydrated iron oxide and jarosite:



Iron solubility is limited relative to acid concentration, thus a minimum of 20g/l free H_2SO_4 is required to achieve high Zn extraction. When the Pressure leach is integrated with the RLE calcine may be used to neutralise the residual acid in the zinc pressure leach solution. In a stand-alone plant this free acid is neutralised using fresh zinc concentrate in a two-stage counter-current process.

To further explain the unit processes in Figure 6.1-2:

Feed Preparation

- The feed to the process is ground to render a product of 98% passing 45 micron as this particle size facilitates a 98% zinc extraction.

2-Stage Pressure Leach

- The temperature in the first autoclave is kept at 150°C. Oxygen is sparged into the autoclave. There is an excess of zinc concentrate in this first autoclave to ensure a solution which is low in acid and iron. The residence time in the first autoclave is one hour and the extraction of Zinc is 75%. An average acid to zinc mole ratio in the first autoclave is 0.75:1 with the acid concentration being 10g/l. Concentration of iron in the solution is 2.5 g/l. The solution from this first autoclave is flashed and then directed to the solution purification stage of the process. The remaining solids are sent to the second autoclave for extraction of the residual zinc. The temperature in the second autoclave is also 150°C but an excess of acid is supplied and overall zinc extraction for the process is increased to 99% (98.5 to 99.5%, Collins *et al*, 1994) of the zinc entering in the feed. The solution from this second stage is recirculated to the first stage (i.e., counter-current processing) while the solid reports to sulphur removal. The return electrolyte contains 45 g/l zinc and 140 g/l H_2SO_4 . The solution reporting to the solution purification and electrowinning unit processes contains 1.5 g/l copper, 2 g/l iron, 9 g/l H_2SO_4 and 160 g/l zinc. The solid waste from the process contains 0.2 mass percent copper, 20 mass percent iron, 55 mass percent sulphur and 0.5 mass percent zinc. Extraction of zinc in the first autoclave is 75% and in the second autoclave in excess of 99%.

Sulphur Recovery

- In this stage sulphur is removed by flotation. The sulphur product is melted and sold. A further sulphidic stream which to which the majority of precious metals in the feed report is also produced. The main reason for developing the sulphur removal process was the recovery of these precious metals (as opposed to the production of a sulphur product). Relatively often this process is only commissioned if there is significant precious metal value in the solid waste from the autoclaves. The remaining solids are typically oxidic wastes and consist primarily of hydrated iron oxide and jarosite as described in the equations listed previously. These wastes are hazardous and are disposed of accordingly.

Solution Purification and Electrowinning

- These unit operations are the same as for the RLE process. However, there is the potential for more iron to be present in this stream than was the case in the RLE process and thus solution purification may require a further unit process to remove iron up-front of the electrowinning process. Deportment of the minor metals in this process has been found to be as expected from design conditions (Collins *et al*, 1994).

Calcine Recycle

- The ferrite recycle at Hudson bay is recycling a precious metal-rich waste from previous operations at the site. This practice will be discontinued when the stockpile is exhausted.

The main difference between operating a single stage pressure leach and a 2-stage leach is that the single stage process necessitates the neutralisation of excess acid in order to achieve high zinc extraction.

6.1.1.3 Bio-leaching

While bio-leaching is mentioned in the context of zinc production, there is extremely limited information available in the open literature. A joint venture between Mintek (Council for Minerals Technology, South Africa) and Bateman Project Holdings (Australia) to investigate the use of bio-leaching in base metals processing (Mining Weekly, 1997; Robinson, 1994). No significant findings are available as yet. In addition, Billitton Pty Ltd has carried out significant work in this area. However, their findings are proprietary and thus have not been published.

6.1.1.4 Imperial Smelting Process

The Imperial Smelting Process has been used for the refining of Lead-Zinc concentrates for a significant length of time. For example, the horizontal smelter, belonging to Britannia Zinc Limited (BZL), was commissioned in 1920 and closed in 1958. The next smelter to be built at the site was a vertical smelter in 1930. While the technology is obviously an old one, there have been improvements and refinements in its operation (BZL, 1999). BZL operates the biggest ISP plant in the world at Avonmouth, this plant was commissioned in 1967. The technology of simultaneously smelting lead and zinc, developed at Avonmouth, is used in Australia, Zambia, France, Germany, Romania, Japan, Canada, Poland, Sardinia, Macedonia, China and India, under license to RTZ.

The Avonmouth smelter annual production is:

- 100 000 tonnes of Zinc
- 50 000 tonnes of Lead Bullion
- 200 000 tonnes of Sulphuric Acid
- 400 tonnes of Cadmium

The feed to the process is 320 000 tpa. The other main input to the process is coke which forms the primary energy source for the blast furnace. Coke is usually added in a ratio of 0.4:1 coke to feed material.

ISP processes have been used both as dedicated process (processes attached to single ore bodies) or in the toll refining of a combination of concentrates. It is common practice to recycle plant zinc residues back into the smelter. In addition, residues from other processes, such as steel plant flue dusts, can be added to the feed material to the smelter. In the case of these dusts, their zinc content reports to the final product, while other metals are bound up in the waste slag from the smelter.

A flow diagram for the ISP is included in Figure 6.1-3 below. This process is discussed in the remainder of this section.

Sintering

- It is usual for the feed to the smelter to be "conditioned" in some manner. This usually takes the form of sintering in which recycled materials are blended with the feed material. The main purposes for the sintering process are to:
 - Desulphurise the feed (reducing the sulphur concentration from 6% to 0.6%)
 - Agglomerate the material
 - Render the feed to the furnace porous to allow sufficient gas-solid reaction in the furnace.
 - Yield lumps of sinter which do not soften in the furnace (i.e., at temperatures below 1140°C)

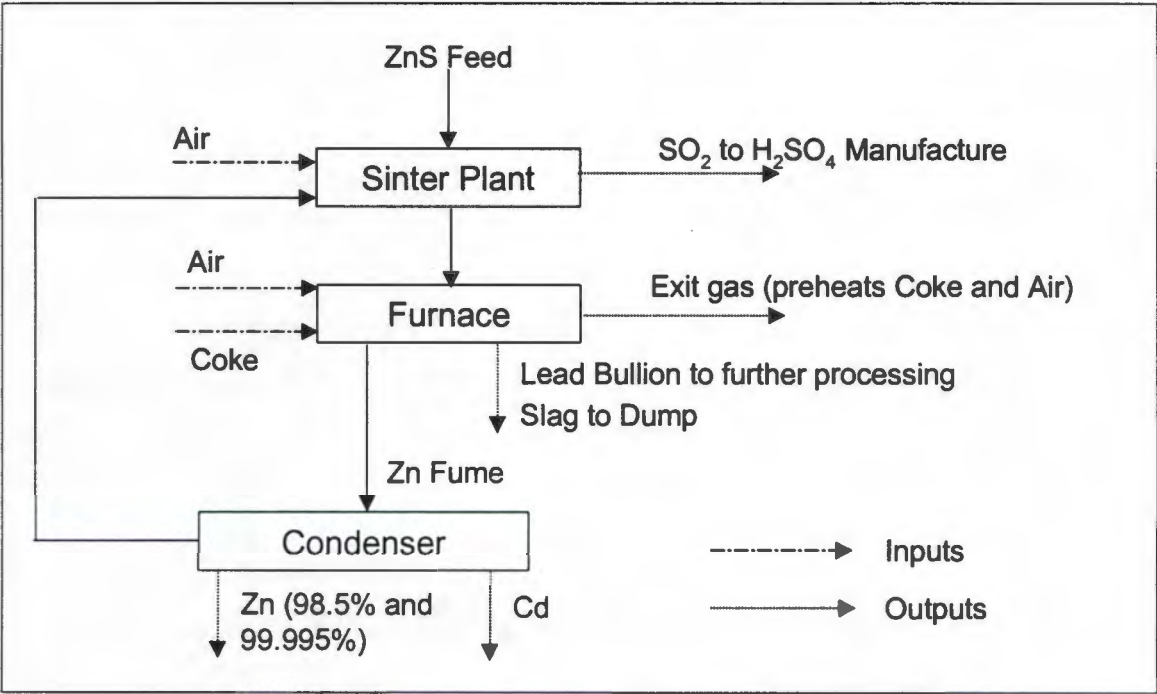


Figure 6.1-3 ISP Flowsheet

The feed to the sinter process is adjusted to ensure that the sinter which it delivers has the correct chemical and physical properties required by the blast furnace. The sinter plant operates by burning the feed material in the presence of air. This reaction continues spontaneously once it has been initiated. The air addition to the sinter plant is commonly in the ratio of 600 Nm³ per ton of feed. Sulphur reports to an SO₂ stream which can be used to make H₂SO₄. There is an internal recycle of sinter fines within the sinter plant. This recycle is internal to the unit process boundary as illustrated in Figure 6.1-3 and thus has not been included. Typical concentrations in the sinter product are included in Table 6.1-1 below:

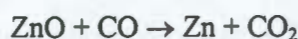
Table 6.1-1 Typical Sinter Product Concentrations (BZL, 1999)

Component	Mass Percent
Zn	40
Pb	20
CaO	5
SiO ₂	4
FeO	12
Cd	0.3
Balance	18.7

Sintering does not differ significantly from roasting and thus they can be modelled as a single unit process in the overall flowsheet.

Smelting

- The reaction taking place in the ISP is (Woollacott and Eric, 1984):



This reaction is reversible at temperatures below 900°C and thus temperatures within the furnace are kept in excess of 1000°C to ensure complete reaction of the Zinc. For this reason the inputs to the furnace are preheated. The main inputs to the furnace are included in Table 6.1-2.

Table 6.1-2 Typical Input to the ISP Blast Furnace

Input	Tons added per ton Sinter
Sinter	1
Coke	0.4
Air	1.5

Reversion of Zn to ZnO in the presence of CO₂ is avoided by the lead splash condenser. Zinc vapour from the furnace passes to the condenser where fine droplets of lead dissolve the zinc. This lead containing the zinc is then pumped into water immersion coolers where the temperature is reduced to around 500°C. The zinc separates from the condenser lead and the zinc is tapped into ladles before being conveyed to the zinc refinery. The lead oxide in sinter feed is reduced to lead in the furnace shaft and is batch tapped from the hearth of the furnace, simultaneously with slag. Lead is recovered, along with copper, gold and silver, as bullion blocks. The slag is then disposed of.

6.1.2 System Parameters

This section of the appendix contains the assumption sets governing the unit operations identified in the steps of the design hierarchy. As was the case with the gold case study presented in section 5 of this thesis, it is not possible to cite specific source references for a number of the values presented in this section as these values represent an aggregation of the performance assumption sets for zinc processing as a whole. In addition, these values have been checked by industry personnel as described in section 2 of this thesis. Where possible, references have been given. The reference list included in Appendix 2-6 can also be seen as a general reference list for the refining of zinc.

6.1.2.1 Mass Balance Assumption Sets

- *Roaster*

Table 6.1-3 Roaster Assumption Set

Roast		
Coke added	0.15	t coke per ton Concentrate
Coke composition	80.00	% Carbon
	20.00	% Ash
Sulphur oxidised	95.00	%
All Carbon is consumed		
Stoichiometric amount of oxygen are required		
Air is 20% oxygen and 80 % Nitrogen		
All water is vaporised		
Volatile metals leave in gas		

- *Leaching*

Table 6.1-4 Leaching Assumption Set

Leach		
Zn leached	94.35	%
Fe leached	5.00	%
Base metals Leached	94.35	%
S leached	100.00	%
Balance leached	5.00	%
Water required	0.50	t/ton ore processed
No ash leaches		
Solid waste is perfectly dry		
Light metals remain in solid		
No makeup acid is required		

- *Solution Purification and Electrowinning*

Table 6.1-5 Solution Purification and Electrowinning Assumption Set

Solution Purification and Electrowinning	
All contaminants removed before Zn is removed	
All Zn removed	
Electricity required	3.6 kWh/ton product

- *Pressure Leach*

Table 6.1-6 Pressure Leach Assumption Set

Pressure Leach		
Zn leached	94.35	%
Fe leached	5.00	%
Base Metals Leached	94.35	%
S precipitated	90.00	%
Balance leached	5.00	%
Water required	0.5	t/ton dry feed
Volatile metals Leached	5.00	%
No ash leaches		
Solid waste is perfectly dry		
Light metals do not mobilise		
No make up acid is required		

- *Imperial Smelting furnace*

Table 6.1-7 Imperial Smelter Assumption Set

ISP		
Coke Feed	0.4	t/ton dry feed
Coke composition	80.00	% Carbon
	20.00	% Ash
Sulphur Oxidised	95	%
Base Metals and Light Metals follow Zinc		
Zinc recovery	95	%
Base Metals	95	%
Light Metals	95	%
Air is 20% oxygen and 80 % Nitrogen		
All Carbon is consumed and produces CO ₂ (Rxn ZnO + CO -> CO ₂)		
Air addition	1.5	t/ton dry feed

- *Waelz Kiln*

Table 6.1-8 Waelz Kiln Assumption Set

Waelz Kiln		
Coke Feed	0.1	t/ton dry feed
Coke composition	80	% Carbon
	20	% Ash
Sulphur Oxidised	95	%
All Carbon is consumed and produces CO ₂ (Rxn ZnO + CO → CO ₂)		
Air is 20% oxygen and 80 % Nitrogen		
Stoichiometric Amounts of Air required		
Silica	0.02	t silica/ton feed
Limestone	0.07	t limestone/ton feed

- *Gas Scrubbing*

Table 6.1-9 Gas Scrubbing Assumption Set

Sulphur Scrubbing		
SO ₂ Removed	95	%
Water Required	1	m ³ per 1000 m ³ of gas
Gas Velocity	5	m/s
Tower height	10	m
Density of Gas	1.29	kg/m ³

- *Sulphuric Acid Plant*

Table 6.1-10 Acid Plant Assumption Set

Acid Plant		
Gas cleaned and dried		
Conversion SO ₂ to SO ₃	95	%
Conversion SO ₃ to H ₂ SO ₄	100	%
Product	99	% H ₂ SO ₄

6.1.2.2 Capital Cost and Energy Requirements

This section contains the equations used to size and cost the equipment for the zinc refinery.

Utility and labour costs are the same as those included in Appendix 5-1, details can be found in that appendix. Inflation rates are also included in that appendix.

- *Roaster*

The roaster has been modeled as a jacketed cylindrical vessel (Peters and Timmerhaus, 1991). This reference gives the cost of a carbon steel reactor as a function of capacity to be (this information has been read off the chart provided and a curve has been fitted to the data):

$$\text{Cost (US\$)} = 0.0035.\text{capacity(US gallon)}^2 + 10.4.\text{capacity} + 5316.8$$

This equation has been converted to ZAR (South African Rands and the currency value adjusted for the 1983 to 1991 period so that the cost is reported on the same basis as the other costs.

Economic information (Editors Inc, 1998)). This results in the following equation:

$$\text{Cost (R)} = 654.93.\text{capacity(m}^3)^2 + 7418.1.\text{capacity} + 14355$$

To calculate the capacity as a function of throughput the following equation set was used. The ratio of height to area for the Lurgi fluidised bed roasters installed at the Zincor zinc refinery in South Africa is (MacLagan *et al*, 1997):

Bank 1: 2.1 m

Bank 2: 1.95 m

A value of 2 for the ratio of height to area the model of 2 was chosen.

Thus the volume of the roaster would be:

$$V = \Pi r^2 h = 2 \Pi^2 r^4$$

The volumetric flowrate (f) of air is known for the roaster. Then the velocity of the air (v) can be calculated:

$$v = f/\text{Area} = f/(\Pi r^2)$$

The fluidising velocity can be calculated according to standard metrics (Yates, 1983).

It is common practice to multiply the design volume obtained using this system of equations by 1.5 to ensure that it is possible to reach the required reaction extents (Peters and Timmerhaus, 1991). This multiplication makes provision for the particles sizes larger than the average particle size used in the calculation.

Coke costs R180 per ton (DMEA, 1997).

- *Leaching*

Leaching is usually carried out in concrete lined vessels. These vessels are usually rectangular and can be painted with acid proof paint. The work of Fugleberg *et al* (1993) gives the following ratios:

Table 6.1-11 Ratio of Leach Volume to Feed flowrate

Unit operation	Volume:Feed ratio ($\times 10^{-3} \text{ m}^3/\text{ton}$)
1 st Leach	1.2
2 nd Leach	0.8
Iron Leach	4.3

The cost for a leach vessel can be calculated using the equation (Mintek, 1983):

$$\text{Cost (R)} = 261.9 \times \text{volume (m}^3\text{)}^{0.7448}$$

- *Solution Purification*

The three-stage purification consists of two sets of tanks (copper and cobalt removal) and a fluidised bed reactor (cadmium removal). The only information available is the existing process capacity relative to feed rates at the Kokkola Plant (Fugleberg *et al*, 1993). These are summarised in the table below.

Table 6.1-12 Ratio of Purification Reactors to Feed Rate

Unit operation	Volume:Feed ratio ($\times 10^{-3} \text{ m}^3/\text{ton}$)
Copper Removal	4.2
Cobalt Removal	0.31
Cadmium Removal	0.73

Cost of tanks (Mintek, 1983):

$$\text{Cost (R)} = 261.9 \times \text{volume (m}^3\text{)}^{0.7448}$$

Cost of fluidised bed reactor (from Roaster calculation above):

$$\text{Cost (R)} = 654.93 \cdot \text{capacity (m}^3\text{)}^2 + 7418.1 \cdot \text{capacity} + 14355$$

- *Electrowinning*

Electrowinning takes place in stainless steel tanks (Mintek, 1983):

$$\text{Cost (R)} = 9639 \times e^{0.0269 \times \text{capacity}}$$

Typical installation: through flow of 0.6 m/min with 75 cm between plates for the worst case scenario (van Niekerk and Begley, 1991).

Electricity requirements: Typical electricity requirements for the production of zinc by electrowinning is 3.5 kWh/t zinc produced (AES, 1997; van Niekerk and Begley, 1991).

- *Pressure Leach*

The pressure leach vessel is an autoclave (Mintek, 1983):

$$\text{Cost (R)} = 19384 + 3057 \cdot \text{volume(m}^3\text{)}$$

For the Cominco process the total autoclave volume is 75 m³. This volume processes a feed rate of 270 kg.min⁻¹ solids (Baldwin *et al*, 1995).

- *Imperial Smelting furnace*

The cost for the smelter is (Lurgi, 1999):

$$\text{Cost (R)} = 9000 \times \text{capacity (ton/hr)}^{3.4564}$$

- *Waelz Kiln*

The price of the kiln can be calculated from (Mintek, 1993):

$$\text{Price (R)} = 9119 \cdot \text{capacity (kg/h)}^{0.4564}$$

The power draw for the kiln is:

$$\text{Power (kW)} = 2.8 + 1.498 \cdot \text{capacity (kg/h)}$$

- *Gas Scrubbing*

This takes place in a stainless steel column:

$$\text{Cost (R)} = 9639 \cdot e^{0.0269 \cdot \text{capacity}} \text{ for capacity in m}^3$$

The required gas velocity is 5 m/s in a 10m high column (Perry and Green, 1984).

- *Sulphuric Acid Plant*

Minimum of three stainless steel columns with a residence time of 30 seconds in each:

$$\text{Cost (R)} = 9639 \cdot e^{0.0269 \cdot \text{capacity}} \text{ for capacity in m}^3$$

6.1.2.3 Product Selling Prices

The selling prices as included in Table 6.1-13 have been used in the development of the model. The conversion rates in this table have been converted using an exchange rate of 6.2 R/US\$ (EMG, 1999). A selling price for mercury was not readily available. The base metals have been grouped together, as have the light metals (Pb and Ag), as was discussed in section 6 of this thesis. Thus the selling prices for these groups needs to be calculated according to the ratios of the individual metals occurring in the feed to the plant.

Table 6.1-13 Commodity Selling Prices

Commodity	Price	Unit	Price (R/ton)	Reference	Group	Group Value (R/ton)
Zinc	1120	US\$/ton	6 900	LME (1999)	Zinc	6 900
Copper	1670	US\$/ton	10 000	LME (1999)	Base Metals	
Cobalt	19	US\$/lb	260 000	LME (1999)		
Cadmium	9500	R/ton	9 500	Mintek (1989)		
Silver	5.4	US\$/troy oz	11 000 000	LME (1999)	Light Metals	20 000
Lead	490	US\$/ton	3 000	LME (1999)		
Sulphur	200	R/ton	200	DMEA (1997)	Sulphur	200
Sulphuric Acid	100	R/ton	100	DMEA (1997)	Sulphuric Acid	100

However, these prices are based on pure metals, the co-products from the system are, barring cadmium, in the form of salts. Thus, the selling price used in the model was estimated to be one quarter that of the pure metal selling price. In the same vein, a buying price for the zinc concentrate of 25% of the selling price was also assumed.

6.1.2.4 Economic Factors

The economic factors assumed for the gold case study were used. See Appendix 5-1 for details.

The additional values used in this case study are:

- Cost of Silica is 10 R/ton

6.1.2.5 Equivalency Factors

Table 6.1-14 below contains the equivalency factors used in the development of the model (PEMS, 1998).

Table 6.1-14 *Equivalency Factors used (all metals assumed waterborne for human toxicity)*

Component	Landfill Vol m ³ per ton	Water Cons l/ton	Resource Depletion /year	Acidification ton SO ₂ per ton	Eco-toxicity aquatic m ³	Eco-toxicity terrestrial m ³	Greenhouse ton CO ₂ per ton	Human Toxicity ton / ton
Zn	7	0	0.05	0	0.38	2.6	0	0.003
S	2.3	0	0.04	64.52	0	0	0	0.038
Fe	7	0	0.01	0	0	0	0	0.004
Base Metals	8	0	0.03	0	38	3.0	0	0.63
Light Metals	10	0	0.09	0	1.9	0.41	0	0.76
Volatile Metals	11	0	0.04	0	495	29	0	4.65
H ₂ O	1	1000	0	0	0	0	0	0
C	0	0	0	0	0	0	0	0
Ash	0	0	0	0	0	0	0	0
O ₂	0	0	0	0	0	0	0	0
N ₂	0	0	0	0	0	0	0	0
SO ₂	0	0	0	1	0	0	0	0
CO ₂	0	0	0	0	0	0	1	0
O in Zn oxide	0	0	0	0	0	0	0	0
SiO ₂	2.3	0	0	0	0	0	0	0
CaO	2.3	0	0	0	0	0	0	0
H ₂ SO ₄	1	0	0	64.52	0	0	0	0.038
Balance	2.3	0	0	0	0	0	0	0

6.1.3 Results of Optimisation - Design Step 2

Two sets of results are presented here. The first set shows the preferred solution as well as the values obtained for all of the attributes at this solution. This preferred solution is a combination of technology suites. For comparison results for separate sets of technologies is also included.

The second set of results shows the shift in the preferred solution when the goals are changed.

6.1.3.1 Results for Initial Optimisation and Comparison with discrete technology suites

The results in Table 6.1-15 detail the preferred technology, the score for this technology (or the sum of the differences) and the actual values achieved for each attribute. In the case of the difference from the goal a negative number denotes that the attribute performs worse than the goal, while a positive number shows that the technology suite chosen performs better than the chosen goal.

Table 6.1-15 Preferred Technology

Preferred Technology Option	44% of feed to Roaster, 56% of feed to Pressure Leach, Electrowin; 100% of Roaster output to Imperial Smelter		
Sum of Differences (Goal Programming)	2.53		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	220.87	225.83	4.95
Water Consumption	2685.72	122.73	-2562.99
Resource Depletion	0.58	0.02	-0.57
Acidification	7.11	0.32	-6.79
Eco-toxicity	31.24	1.20	-30.04
Greenhouse	60.13	2.73	-57.41
Human Toxicity	1.18	0.04	-1.14
Cumulative Profit	3900.86	-1989.77	5890.63

Table 6.1-16 contains the results obtained by discrete technology choices. In order to render the results comparable, the goals set for the comparison included in Table 6.1-16 below were the same as those used in evaluating the preferred solution as detailed in Table 6.1-15.

6.1.3.2 Results for changing Goals

The goals were changed so that they were further away from the best attainable value for each attribute and the technology selection optimised for each of these new sets of goals. The results are included in Table 6.1-17.

Table 6.1-16 Results from discrete technology suites

Technology Option	Roast, Imperial Smelter		
Sum of Differences (Goal Programming)	5.19		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	209.36	225.83	16.46
Water Consumption	6094.64	122.73	-5971.90
Resource Depletion	0.32	0.02	-0.30
Acidification	16.14	0.32	-15.82
Eco-toxicity	51.04	1.20	-49.84
Greenhouse	136.45	2.73	-133.72
Human Toxicity	0.58	0.04	-0.54
Cumulative Profit	-468390.51	-1989.77	-466400.74
Technology Option	Roast, Leach, Electrowin		
Sum of Differences (Goal Programming)	5.02		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	218.17	225.83	7.65
Water Consumption	6136.72	122.73	-6013.99
Resource Depletion	0.45	0.02	-0.43
Acidification	15.48	0.32	-15.15
Eco-toxicity	55.94	1.20	-54.74
Greenhouse	35.88	2.73	-33.15
Human Toxicity	1.70	0.04	-1.66
Cumulative Profit	-471860.59	-1989.77	-469870.82
Technology Option	Pressure Leach, Electrowin		
Sum of Differences (Goal Programming)	4.07		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	229.94	225.83	-4.12
Water Consumption	0.10	122.73	122.63
Resource Depletion	0.79	0.02	-0.77
Acidification	0.00	0.32	0.32
Eco-toxicity	15.65	1.20	-14.45
Greenhouse	0.01	2.73	2.72
Human Toxicity	1.65	0.04	-1.62
Cumulative Profit	15134.08	-1989.77	17123.86

Table 6.1-17 Comparison of Preferred Technologies with Changing Goals

Goal Set	Best Performance		
Preferred Technology Option	21% of feed to Roaster, 79% of feed to Pressure Leach, Electrowin; 100% of Roaster output to Imperial Smelter		
Sum of Differences (Goal Programming)	3.89		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	225.67	229.94	4.27
Water Consumption	1265.70	153.42	-1112.28
Resource Depletion	0.69	0.02	-0.67
Acidification	3.35	0.40	-2.95
Eco-toxicity	23.00	1.40	-21.60
Greenhouse	28.34	3.41	-24.93
Human Toxicity	1.43	0.04	-1.39
Cumulative Profit	18466.93	461.80	18005.13
Goal	80% of Best Performance		
Preferred Technology Option	44% of feed to Roaster, 56% of feed to Pressure Leach, Electrowin; 100% of Roaster output to Imperial Smelter		
Sum of Differences (Goal Programming)	2.53		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	220.87	225.83	4.95
Water Consumption	2685.72	122.73	-2562.99
Resource Depletion	0.58	0.02	-0.57
Acidification	7.11	0.32	-6.79
Eco-toxicity	31.24	1.20	-30.04
Greenhouse	60.13	2.73	-57.41
Human Toxicity	1.18	0.04	-1.14
Cumulative Profit	3900.86	-1989.77	5890.63
Goal Set	60% of Best Performance		
Preferred Technology Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin; 100% of Roaster output to Imperial Smelter		
Sum of Differences (Goal Programming)	1.55		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	220.41	221.71	1.31
Water Consumption	2824.57	92.05	-2732.52
Resource Depletion	0.57	0.02	-0.56
Acidification	7.48	0.24	-7.24
Eco-toxicity	32.05	1.00	-31.05
Greenhouse	63.24	2.05	-61.20
Human Toxicity	1.16	0.03	-1.12
Cumulative Profit	0.00	-4441.34	4441.34

Goal Set	40% of Best Performance		
Preferred Technology Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin; 81% of Roaster output to Imperial Smelter, 19% of Roaster output to Leach, Electrowin		
Sum of Differences (Goal Programming)	1.28		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	221.17	217.60	-3.57
Water Consumption	2828.78	61.37	-2767.41
Resource Depletion	0.58	0.01	-0.57
Acidification	7.42	0.16	-7.26
Eco-toxicity	32.48	0.79	-31.68
Greenhouse	54.55	1.36	-53.19
Human Toxicity	1.25	0.03	-1.23
Cumulative Profit	0.00	-6892.92	6892.92
Goal Set	20% of Best Performance		
Preferred Technology Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin; 38% of Roaster output to Imperial Smelter, 62% of Roaster output to Leach, Electrowin		
Sum of Differences (Goal Programming)	2.28		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	222.94	213.48	-9.46
Water Consumption	2829.87	30.69	-2799.19
Resource Depletion	0.61	0.01	-0.60
Acidification	7.27	0.08	-7.19
Eco-toxicity	33.41	0.59	-32.81
Greenhouse	34.39	0.68	-33.71
Human Toxicity	1.48	0.02	-1.46
Cumulative Profit	0.00	-9344.49	9344.49
Goal Set	5% of Best Performance		
Preferred Technology Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin; 6% of Roaster output to Imperial Smelter, 94% of Roaster output to Leach, Electrowin		
Sum of Differences (Goal Programming)	3.03		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	224.32	210.39	-13.93
Water Consumption	2806.13	7.67	-2798.46
Resource Depletion	0.63	0.01	-0.62
Acidification	7.10	0.02	-7.08
Eco-toxicity	33.95	0.44	-33.50
Greenhouse	19.11	0.17	-18.94
Human Toxicity	1.65	0.02	-1.63
Cumulative Profit	0.00	-11183.17	11183.17

6.1.3.3 Changes in Attribute Values

The results included in Table 6.1-18 are a permutation of those to be found in Table 6.1-17.

Table 6.1-18 Changes in Attribute Values for Goals

Attribute	Value	Goal	% Difference	Goals
Landfill Volume	225.67	229.94	1.89	Best Performance
	220.87	225.83	2.24	80% of Best Performance
	220.41	221.71	0.59	60% of Best Performance
	221.17	217.60	-1.61	40% of Best Performance
	222.94	213.48	-4.24	20% of Best Performance
	224.32	210.39	-6.21	5% of Best Performance
Water Consumption	1265.70	153.42	-87.88	Best Performance
	2685.72	122.73	-95.43	80% of Best Performance
	2824.57	92.05	-96.74	60% of Best Performance
	2828.78	61.37	-97.83	40% of Best Performance
	2829.87	30.69	-98.92	20% of Best Performance
	2806.13	7.67	-99.73	5% of Best Performance
Resource Depletion	0.69	0.02	-97.15	Best Performance
	0.58	0.02	-97.02	80% of Best Performance
	0.57	0.02	-97.37	60% of Best Performance
	0.58	0.01	-97.82	40% of Best Performance
	0.61	0.01	-98.30	20% of Best Performance
	0.63	0.01	-98.64	5% of Best Performance
Acidification	3.35	0.40	-87.96	Best Performance
	7.11	0.32	-95.46	80% of Best Performance
	7.48	0.24	-96.76	60% of Best Performance
	7.42	0.16	-97.83	40% of Best Performance
	7.27	0.08	-98.89	20% of Best Performance
	7.10	0.02	-99.72	5% of Best Performance
Eco-toxicity	23.00	1.40	-93.92	Best Performance
	31.24	1.20	-96.17	80% of Best Performance
	32.05	1.00	-96.89	60% of Best Performance
	32.48	0.79	-97.55	40% of Best Performance
	33.41	0.59	-98.23	20% of Best Performance
	33.95	0.44	-98.70	5% of Best Performance
Greenhouse	28.34	3.41	-87.97	Best Performance
	60.13	2.73	-95.46	80% of Best Performance
	63.24	2.05	-96.76	60% of Best Performance
	54.55	1.36	-97.50	40% of Best Performance
	34.39	0.68	-98.02	20% of Best Performance
	19.11	0.17	-99.11	5% of Best Performance
Human Toxicity	1.43	0.04	-97.03	Best Performance
	1.18	0.04	-96.87	80% of Best Performance
	1.16	0.03	-97.29	60% of Best Performance
	1.25	0.03	-97.95	40% of Best Performance
	1.48	0.02	-98.64	20% of Best Performance
	1.65	0.02	-99.03	5% of Best Performance
Cumulative Profit	18466.93	461.80	-97.50	Best Performance
	3900.86	-1989.77	-151.01	80% of Best Performance
	0.00	-4441.34	0.00	60% of Best Performance
	0.00	-6892.92	0.00	40% of Best Performance
	0.00	-9344.49	0.00	20% of Best Performance
	0.00	-11183.17	0.00	5% of Best Performance

6.1.4 Recycle Permutations including Roast-Leach-Electrowin

Figure 6.1-4 contains details of the recycle structure for all the reactor-separators combinations identified in the design hierarchy.

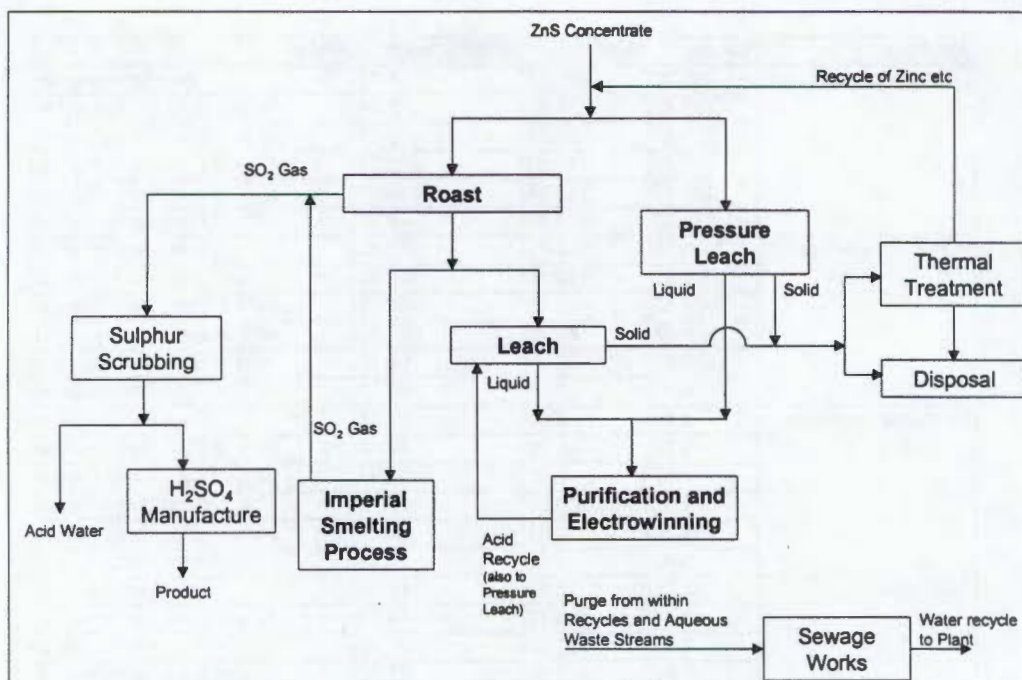


Figure 6.1-4 *Recycle Permutations including Roast-Leach-Electrowin*

6.1.5 Results of Optimisation - Design Step 3 Pre-EIA

6.1.5.1 Goals set for the Optimisation (80% of best performance)

The goals used for this step are included in Table 6.1-19 overleaf.

Table 6.1-19 Goals for Design Step 3

Attributes	Value (/ton Zn Produced)
Landfill Volume	205.63
Water Consumption	845.97
Resource Depletion	0.06
Acidification	0.19
Eco-toxicity	0.18
Greenhouse	0.91
Human Toxicity	0.02
Cumulative Profit	-2141.71

6.1.5.2 Results of Optimisation for Different Economic Goals

Table 6.1-20 contains details of the process for different economic goals.

Table 6.1-20 Preferred Solutions for Different Economic Goals

Preferred Solution	Split of Feed		Split of Solid Waste		Split of Off-gases			Split of Scrubber Water		Split of Sulphur	
	Roaster	Press. Leach	Waste Kiln	Disposal	Atmosphere	Scrubbing	Acid Plant	Waste	Acid Plant	Product	Acid Plant
Constrained to Hurdle Rate of 20%	0.46	0.54	1.00	0.00	0.71	0.28	0.01	0.00	1.00	0.33	0.67
Economic Goal = 20% Hurdle Rate	0.46	0.54	1.00	0.00	0.71	0.28	0.01	0.00	1.00	0.33	0.67
Economic Goal is 80% of Best Performance*	0.64	0.36	1.00	0.00	0.54	0.19	0.27	0.00	1.00	0.53	0.47

* the hurdle rate on this solution is 0.71% which means that it is barely profitable over the 40 year life of plant.

A summary of the performance of these solutions is included in Table 6.1-21.

6.1.5.3 Results of Optimisation

The preferred combination of technologies relative to the set of goals defined is described in Table 6.1-22. For this solution the constraint that the process be profitable was included.

6.1.5.4 Results of Sensitivity Analysis - Changing Weightings Applied to Goals

The tables included below contain the results obtained when weightings were applied to the attributes in the model. Two different weightings were applied:

- The attribute under consideration is twice as important as the other attributes
- The attribute under consideration is five times as important as the other attributes.

The goals for this analysis are consistent with those included in section 6.1.5.1 of this Appendix. These goals are also included in the body of the report.

Table 6.1-21 Summary of Performances relative to different Economic Goals

Economic Attribute	Constrained to Hurdle Rate of 20%		
Preferred Reactor-Separator Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin		
Preferred Waste Management for these Technologies	All solid waste to the Waelz Kiln; 71% of Off-gases to the atmosphere, 28% of gases to scrubbing, 1% of gases to acid plant; all water from scrubber to Acid Plant; 67% of Sulphur to acid production, 33% to Sulphur product		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	211.28	205.63	-5.66
Water Consumption	846.28	845.97	-0.31
Resource Depletion	0.06	0.06	0.00
Acidification	0.22	0.19	-0.03
Eco-toxicity	0.18	0.18	0.00
Greenhouse	1.69	0.91	-0.78
Human Toxicity	0.03	0.02	-0.01
Cumulative Profit	not applicable		
Economic Attribute	Economic Goal = 20% Hurdle Rate		
Preferred Reactor-Separator Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin		
Preferred Waste Management for these Technologies	All solid waste to the Waelz Kiln; 71% of Off-gases to the atmosphere, 28% of gases to scrubbing, 1% of gases to acid plant; all water from scrubber to Acid Plant; 67% of Sulphur to acid production, 33% to Sulphur product		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	211.28	205.63	-5.66
Water Consumption	845.97	845.97	0.00
Resource Depletion	0.06	0.06	0.00
Acidification	0.22	0.19	-0.03
Eco-toxicity	0.18	0.18	0.00
Greenhouse	1.69	0.91	-0.78
Human Toxicity	0.03	0.02	-0.01
Cumulative Profit	0.26	0.00	-0.26
Economic Attribute	Economic Goal is 80% of Best Performance		
Preferred Reactor-Separator Option	64% of feed to Roaster, 36% of feed to Pressure Leach, Electrowin		
Preferred Waste Management for these Technologies	All solid waste to the Waelz Kiln; 54% of Off-gases to the atmosphere, 19% of gases to scrubbing, 27% of gases to acid plant; all water from scrubber to Acid Plant; 47% of Sulphur to acid production, 53% to Sulphur product		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	211.28	205.63	-5.66
Water Consumption	845.97	845.97	0.00
Resource Depletion	0.06	0.06	0.00
Acidification	0.22	0.19	-0.03
Eco-toxicity	0.18	0.18	0.00
Greenhouse	1.69	0.91	-0.78
Human Toxicity	0.03	0.02	-0.01
Cumulative Profit	0.26	0.00	-0.26

Table 6.1-22 Preferred Solution Design Step 3

Preferred Reactor-Separator Option	46% of feed to Roaster, 54% of feed to Pressure Leach, Electrowin		
Preferred Waste Management for these Technologies	All solid waste to the Waelz Kiln; 71% of Off-gases to the atmosphere, 28% of gases to scrubbing, 1% of gases to acid plant; all water from scrubber to Acid Plant; 67% of Sulphur to acid production, 33% to Sulphur product		
Sum of Differences (Goal Programming)	1.99		
	Attribute Values (normalised with respect to ton Zn Product)	Goal	Difference from Goal
Landfill Volume	211.28	205.63	-5.66
Water Consumption	845.97	845.97	0.00
Resource Depletion	0.06	0.06	0.00
Acidification	0.22	0.19	-0.03
Eco-toxicity	0.18	0.18	0.00
Greenhouse	1.69	0.91	-0.78
Human Toxicity	0.03	0.02	-0.01
Cumulative Profit	0.26	0.00	0.26

Table 6.1-23 Sensitivity of Solution to Weighting of Landfill attribute

Attribute	Landfill Volume		
Weighting	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.80	0.82	0.46
Pressure Leach	0.20	0.18	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.44	0.43	0.71
Scrubbing	0.15	0.14	0.27
Acid Plant	0.41	0.42	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.98	1.00	0.33
Acid Plant	0.02	0.00	0.67
Attribute Values (normalised with respect to ton Zn Product)			
Landfill Volume	205.99	205.63	211.28
Water Consumption	845.78	846.01	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.20	0.20	0.22
Eco-toxicity	0.18	0.18	0.18
Greenhouse	2.77	2.84	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-4355.41	-4957.95	0.26

Table 6.1-24 Sensitivity of Solution to Weighting of Water Consumption attribute

Attribute	Water Consumption		
	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.67	0.67	0.46
Pressure Leach	0.33	0.33	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.52	0.51	0.71
Scrubbing	0.18	0.18	0.27
Acid Plant	0.31	0.30	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.58	0.58	0.33
Acid Plant	0.42	0.42	0.67
Attribute Values (normalised with respect to ton Zn Product)			
Landfill Volume	207.97	205.63	211.28
Water Consumption	837.39	845.97	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.21	0.19	0.22
Eco-toxicity	0.18	0.18	0.18
Greenhouse	2.37	0.91	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-1882.71	0.00	0.26

Table 6.1-25 Sensitivity of Solution to Weighting of Resource Depletion attribute

Attribute	Resource Depletion		
	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.64	0.59	0.46
Pressure Leach	0.36	0.41	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.53	0.57	0.71
Scrubbing	0.19	0.21	0.27
Acid Plant	0.27	0.23	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.53	0.47	0.33
Acid Plant	0.47	0.53	0.67
Attribute Values			
Landfill Volume	208.45	209.14	211.28
Water Consumption	845.96	845.98	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.21	0.21	0.22
Eco-toxicity	0.18	0.18	0.18
Greenhouse	2.27	2.13	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-1463.26	-960.68	0.26

Table 6.1-26 Sensitivity of Solution to Weighting of Acidification attribute

Attribute	Acidification		
Weighting	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.78	0.60	0.46
Pressure Leach	0.22	0.40	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.40	0.45	0.71
Scrubbing	0.15	0.24	0.27
Acid Plant	0.45	0.31	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.89	0.49	0.33
Acid Plant	0.11	0.51	0.67
Attribute Values			
Landfill Volume	206.35	209.05	211.28
Water Consumption	844.90	1012.39	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.19	0.19	0.22
Eco-toxicity	0.18	0.19	0.18
Greenhouse	2.70	2.15	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-3854.84	-1043.67	0.26

Table 6.1-27 Sensitivity of Solution to Weighting of Eco-toxicity attribute

Attribute	Total Eco-toxicity		
Weighting	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.63	0.62	0.46
Pressure Leach	0.37	0.38	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.54	0.55	0.71
Scrubbing	0.19	0.20	0.27
Acid Plant	0.27	0.25	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.52	0.50	0.33
Acid Plant	0.48	0.50	0.67
Attribute Values			
Landfill Volume	208.58	208.79	211.28
Water Consumption	845.96	846.45	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.21	0.21	0.22
Eco-toxicity	0.18	0.18	0.18
Greenhouse	2.24	2.20	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-1360.28	-1201.22	0.26

Table 6.1-28 Sensitivity of Solution to Weighting of Greenhouse gases attribute

Attribute Weighting	Greenhouse		
	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.45	0.22	0.46
Pressure Leach	0.55	0.78	0.54
Split of Solid Waste			
Waelz Kiln	1.00	0.99	1.00
Landfill	0.00	0.01	0.00
Split of Off-gases			
Atmosphere	0.72	0.83	0.71
Scrubbing	0.28	0.17	0.27
Acid Plant	0.00	0.00	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.33	0.26	0.33
Acid Plant	0.67	0.74	0.67
Attribute Values			
Landfill Volume	211.37	215.02	211.28
Water Consumption	844.17	845.37	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.22	0.19	0.22
Eco-toxicity	0.18	0.20	0.18
Greenhouse	1.67	0.94	1.69
Human Toxicity	0.03	0.03	0.03
Cumulative Profit	23.04	-253.22	0.26

Table 6.1-29 Sensitivity of Solution to Weighting of Human Toxicity attribute

Attribute Weighting	Human Toxicity		
	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.78	0.80	0.46
Pressure Leach	0.22	0.20	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.45	0.44	0.71
Scrubbing	0.15	0.15	0.27
Acid Plant	0.40	0.41	0.01
Split of Scrubber Water			
Waste	0.00	0.00	0.00
Acid Plant	1.00	1.00	1.00
Split of Sulphur			
Product	0.90	1.00	0.33
Acid Plant	0.10	0.00	0.67
Attribute Values			
Landfill Volume	206.28	205.95	211.28
Water Consumption	845.97	845.97	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.20	0.20	0.22
Eco-toxicity	0.18	0.18	0.18
Greenhouse	2.71	2.78	1.69
Human Toxicity	0.02	0.02	0.03
Cumulative Profit	-3920.97	-4418.35	0.26

Table 6.1-30 Sensitivity of Solution to Weighting of Cumulative Profit attribute

Attribute	Cumulative Profit		
	Twice	Five Times	Equal
Stream Splits			
Split of Feed			
Roaster	0.51	0.43	0.46
Pressure Leach	0.49	0.57	0.54
Split of Solid Waste			
Waelz Kiln	1.00	1.00	1.00
Landfill	0.00	0.00	0.00
Split of Off-gases			
Atmosphere	0.65	0.47	0.71
Scrubbing	0.24	0.29	0.27
Acid Plant	0.11	0.25	0.01
Split of Scrubber Water			
Waste	0.00	0.40	0.00
Acid Plant	1.00	0.60	1.00
Split of Sulphur			
Product	0.38	0.31	0.33
Acid Plant	0.62	0.69	0.67
Attribute Values			
Landfill Volume	210.45	211.87	211.28
Water Consumption	845.97	824.13	845.97
Resource Depletion	0.06	0.06	0.06
Acidification	0.21	0.19	0.22
Eco-toxicity	0.18	0.20	0.18
Greenhouse	1.86	1.59	1.69
Human Toxicity	0.03	0.03	0.03
Cumulative Profit	-277.66	115.95	0.26

Table 6.1-31 contains a rounded-off summary of the results from Table 6.1-23 to Table 6.1-30.

Table 6.1-31 Summary of Results from Weighting Preferences

Weighted Attribute	Feed	Off-gas		Sulphur	Economic Attribute (R/ton Zn Product)
	To Roaster	To Acid Plant	To Gas Scrubber	To Product	
Landfill Volume	0.81	0.41	0.15	1.00	-4500
Water Consumption	0.67	0.31	0.18	0.58	-500
Resource Depletion	0.62	0.24	0.20	0.50	-1000
Acidification	0.69	0.37	0.22	0.70	-2000
Eco-toxicity	0.62	0.26	0.20	0.50	-1200
Greenhouse	0.40	0.00	0.23	0.31	-100
Human Toxicity	0.79	0.40	0.15	0.95	-4000
Cumulative Profit	0.44	0.20	0.15	0.36	100
Equally Weighted	0.45	0.00	0.26	0.30	0
Range of Values	0.40 to 0.80	0.00 to 0.50	0.20 to 0.30	0.40 to 1.0	-4500 to 100

6.1.5.5 Results of Sensitivity Analysis - Optimising on Environment and Economics in Isolation

The technology combination which would deliver:

- The best environmental performance for all the attributes being weighted equally
- The best economic performance

were evaluated. These optimisations were carried out after the constraint on the economic attribute has been removed.

In addition, the preferred solution for the set of values which described the environment together as being as important as the single economic attribute was also determined.

This set of results is included in Table 6.1-32.

Table 6.1-32 *Sensitivity of Solution to including different attributes*

Attributes included	Environment	Economics	Env together = Economic	Both
Stream Splits				
Split of Feed				
Roaster	0.80	0.34	0.43	0.46
Pressure Leach	0.20	0.66	0.57	0.54
Split of Solid Waste				
Waelz Kiln	1.00	0.00	1.00	1.00
Landfill	0.00	1.00	0.00	0.00
Split of Off-gases				
Atmosphere	0.44	1.00	0.71	0.71
Scrubbing	0.15	0.00	0.29	0.27
Acid Plant	0.41	0.00	0.00	0.01
Split of Scrubber Water				
Waste	0.00	1.00	0.00	0.00
Acid Plant	1.00	0.00	1.00	1.00
Split of Sulphur				
Product	1.00	1.00	0.32	0.33
Acid Plant	0.00	0.00	0.68	0.67
Attribute Values				
Landfill Volume	205.95	211.71	211.65	211.28
Water Consumption	845.98	1096.41	845.96	845.97
Resource Depletion	0.06	0.07	0.06	0.06
Acidification	0.20	0.53	0.21	0.22
Eco-toxicity	0.18	0.23	0.18	0.18
Greenhouse	2.78	1.25	1.62	1.69
Human Toxicity	0.02	0.03	0.03	0.03
Cumulative Profit	-4424.56	250.30	86.65	0.26

6.1.5.6 Results of Sensitivity Analysis - Groupings impacts and weighting the Groups

The impacts were grouped relative to their spatial effect - be this local, regional, or global. The sensitivity of the solution to changing the weights on these groups was investigated. In addition, the solution to the system of equations was also determined for the cases where the only attributes included were those in the group. Impacts were always valued equally within the group.

Table 6.1-33 Sensitivity of Solution to Weighting grouped impacts - Local

Attribute Group	Local			
Weighting	Group Only	Twice	Five Times	Equal
Stream Splits				
Split of Feed				
Roaster	0.80	0.80	0.80	0.46
Pressure Leach	0.20	0.20	0.20	0.54
Split of Solid Waste				
Waelz Kiln	0.68	1.00	1.00	1.00
Landfill	0.32	0.00	0.00	0.00
Split of Off-gases				
Atmosphere	0.44	0.44	0.44	0.71
Scrubbing	0.28	0.15	0.15	0.27
Acid Plant	0.28	0.41	0.41	0.01
Split of Scrubber Water				
Waste	n/a	0.00	0.00	0.00
Acid Plant	n/a	1.00	1.00	1.00
Split of Sulphur				
Product	n/a	1.00	1.00	0.33
Acid Plant	n/a	0.00	0.00	0.67
Attribute Values				
Landfill Volume	205.63	205.95	205.95	211.28
Water Consumption	1570.27	845.68	846.15	845.97
Resource Depletion	0.06	0.06	0.06	0.06
Acidification	0.32	0.20	0.20	0.22
Eco-toxicity	0.18	0.18	0.18	0.18
Greenhouse	2.77	2.78	2.78	1.69
Human Toxicity	0.02	0.02	0.02	0.03
Cumulative Profit	-4454.48	-4418.36	-4419.15	0.26

n/a means that this stream split was not affected

Table 6.1-34 Sensitivity of Solution to Weighting grouped impacts - Regional

Attribute Group	Regional			
Weighting	Group Only	Twice	Five Times	Equal
Stream Splits				
Split of Feed				
Roaster	0.61	0.65	0.65	0.46
Pressure Leach	0.39	0.35	0.35	0.54
Split of Solid Waste				
Waelz Kiln	0.96	1.00	1.00	1.00
Landfill	0.04	0.00	0.00	0.00
Split of Off-gases				
Atmosphere	0.26	0.38	0.38	0.71
Scrubbing	0.20	0.18	0.18	0.27
Acid Plant	0.54	0.44	0.44	0.01
Split of Scrubber Water				
Waste	0.57	0.35	0.35	0.00
Acid Plant	0.43	0.65	0.65	1.00
Split of Sulphur				
Product	0.00	0.54	0.54	0.33
Acid Plant	1.00	0.46	0.46	0.67
Attribute Values				
Landfill Volume	208.91	208.27	208.27	211.28
Water Consumption	845.97	846.08	845.97	845.97
Resource Depletion	0.06	0.06	0.06	0.06
Acidification	0.19	0.19	0.19	0.22
Eco-toxicity	0.21	0.19	0.19	0.18
Greenhouse	2.18	2.32	2.32	1.69
Human Toxicity	0.02	0.02	0.02	0.03
Cumulative Profit	-1154.79	-1673.22	-1672.98	0.26

Table 6.1-35 Sensitivity of Solution to Weighting grouped impacts - Global

Attribute Group	Global			
Weighting	Group Only	Twice	Five Times	Equal
Stream Splits				
Split of Feed				
Roaster	0.24	0.54	0.21	0.46
Pressure Leach	0.76	0.46	0.79	0.54
Split of Solid Waste				
Waelz Kiln	0.52	1.00	1.00	1.00
Landfill	0.48	0.00	0.00	0.00
Split of Off-gases				
Atmosphere	0.10	0.61	1.00	0.71
Scrubbing	0.45	0.20	0.00	0.27
Acid Plant	0.45	0.18	0.00	0.01
Split of Scrubber Water				
Waste	0.50	0.00	0.55	0.00
Acid Plant	0.50	1.00	0.45	1.00
Split of Sulphur				
Product	0.17	0.41	0.21	0.33
Acid Plant	0.83	0.59	0.79	0.67
Attribute Values				
Landfill Volume	213.15	209.95	215.19	211.28
Water Consumption	703.91	751.76	760.71	845.97
Resource Depletion	0.06	0.06	0.06	0.06
Acidification	0.58	0.21	0.19	0.22
Eco-toxicity	0.23	0.18	0.19	0.18
Greenhouse	0.91	1.96	0.91	1.69
Human Toxicity	0.03	0.02	0.04	0.03
Cumulative Profit	-89.79	-500.17	-361.36	0.26

6.1.6 Results for Design Step 3 - after EIA

6.1.6.1 Results of Value Function

Table 6.1-36 Preferred Technology Combinations - Step 3

Option	A
Technology Combination	Roast, Imperial Smelter, all off-gases to an acid plant
	Attribute Values (normalised with respect to ton Zn Product)
Landfill Volume	5.08
Water Consumption	44.59
Resource Depletion	0.06
Acidification	0.05
Eco-toxicity	0.25
Greenhouse	3.41
Human Toxicity	0.01
Cumulative Profit	606.99
Option	B
Technology Combination	Pressure Leach, Electrowin, all S ^o to product, solid waste to Waelz Kiln
	Attribute Values (normalised with respect to ton Zn Product)
Landfill Volume	5.47
Water Consumption	0.00
Resource Depletion	0.07
Acidification	0.10
Eco-toxicity	0.23
Greenhouse	0.23
Human Toxicity	0.04
Cumulative Profit	625.43
Option	C
Technology Combination	43% of feed to Roast, ISP; 57% of feed to Pressure leach, Electrowin; all solid waste to Waelz Kiln; all gases to acid plant; all S ^o to acid manufacture
	Attribute Values (normalised with respect to ton Zn Product)
Landfill Volume	5.30
Water Consumption	0.00
Resource Depletion	0.06
Acidification	0.09
Eco-toxicity	0.24
Greenhouse	1.61
Human Toxicity	0.03
Cumulative Profit	605.94

6.1.6.2 Preferred Flowsheet Option

Table 6.1-37 includes details on how the scores for each flowsheet option were calculated.

Table 6.1-37 Preferred Flowsheet Option

Attribute	Actual			Range		Weights	Weighted Attributes		
	Option A	Option B	Option C	Best	Worst		Option A	Option B	Option C
Landfill Volume	5.08	5.47	5.30	5.08	5.47	2.64	0.00	2.64	1.49
Water Consumption	44.59	0.00	0.00	0.00	44.59	4.07	4.07	0.00	0.00
Resource Depletion	0.06	0.07	0.06	0.06	0.07	3.65	0.00	3.65	0.11
Acidification	0.05	0.10	0.09	0.05	0.10	3.83	0.00	3.83	2.95
Eco-toxicity	0.25	0.23	0.24	0.23	0.25	4.07	4.07	0.00	1.76
Greenhouse	3.41	0.23	1.61	0.23	3.41	4.06	4.06	0.00	1.76
Human Toxicity	0.01	0.04	0.03	0.01	0.04	3.47	0.00	3.47	1.97
Cumulative Profit	606.99	625.43	605.94	625.43	605.94	1.79	1.69	0.00	1.79
Score							13.89	13.59	11.83

6.1.6.3 Results of Sensitivity Analyses

Two sensitivity analyses were conducted. A complete set of results for each of these is included below.

- Sensitivity to Increased Weights

Table 6.1-38 Increase in Flowsheet score with increased Attribute Weighting

	Score	Percentage Increase
Initial Flowsheet	9.09	
Landfill Volume	9.19	1.07
Water Consumption	9.09	0.00
Resource Consumption	9.10	0.04
Acidification	9.22	1.39
Ecotoxicity	9.36	2.92
Greenhouse	9.38	3.19
Human Toxicity	9.19	1.08
Cumulative Profit	9.12	0.32

- *Sensitivity to Increase in Attribute Value*

Table 6.1-39 Increase in Flowsheet Score with increased Attribute value

	Score	Percentage Increase
Initial Flowsheet	9.09	
Landfill Volume	16.95	86.35
Water Consumption	9.09	0.00
Resource Consumption	11.99	31.85
Acidification	9.93	9.14
Ecotoxicity	22.32	145.43
Greenhouse	9.44	3.85
Human Toxicity	9.59	5.50
Cumulative Profit	10.54	15.94

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Appendix 6-2: Summary of South African environmental legislation

Environmental legislation in South Africa is complex as a significant number of government departments have been allocated different responsibilities all of which combine to form environmental legislation. It is only possible to state categorically that environmental legislation sets no legal criteria on impact categories if the environmental Acts of government departments involved in the management of environmental issues in South Africa are reviewed in their entirety.

The information contained in this section is a summary of the *Draft White Paper on Integrated Pollution and Waste Management for South Africa: A policy on Pollution Prevention, Waste Minimisation, Impact control and Remediation* (DEAT, 1998). This document deals with a number of areas in detail.

6.2.1 Lack of Appropriate Ambient Standards

Although both the Department of Water Affairs and Forestry's *Minimum Requirements for Waste Management* (DWAF, 1997) and the South African Bureau of Standard's (SABS) *Code of Practice for Mine Residue Deposits* (SABS, 1996), address the current lack of appropriate standards for waste disposal these now have to be adequately enforced. Only for the limited case of hazardous waste generation, collection and transportation is legislation currently being drafted by the Department of Transport. These deal solely with the handling of the material, as far as quality is concerned, appropriate ambient standards need to be formulated.

6.2.2 Water Pollution: Fresh water

6.2.2.1 Existing legal situation

The control of fresh water pollution is governed by thirty-one pieces of national legislation, at least three provincial ordinances and by a significant number of local government by-laws. The most important national legislation is the Water Act (DWAF, 1996) of which sections 21 and 22 are particularly relevant to pollution control. Section 21 requires that water used for industrial purposes be purified in accordance with standards as published in *The Government Gazette* and returned to the point from which it was abstracted. However, it is sufficient for the user to be exempted from these provisions, subject to the conditions that the minister may impose. Thus these guidelines can not be seen as legal limits as such.

Section 22 of the Act empowers the Minister to direct that the person that has control over land to take steps to prevent water pollution.

6.2.2.2 Institutional arrangements

With regard to national ministries responsible, only the Department of Water Affairs and Forestry has any significant control over fresh water pollution and activities associated with its mitigation.

6.2.2.3 Administration

Most of the legislation dealing with the control of fresh water pollution concerns water used in a specific area, and generally speaking does not require the issuing of permits. The main exception is the Water Act that specifies the issuing of a permit for the use of a quantity of water exceeding 150 cubic metres per year for industrial purposes. A second category of exemptions is the by-laws of some local authorities that do not accept trade effluent into the sewage system which they manage in the absence of a duly authorised permit.

6.2.3 Gas, Smoke and Dust

6.2.3.1 Existing legislation

Atmospheric pollution by gas, smoke and dust is, for the most part, governed by the Atmospheric Pollution Prevention Act (DNH, 1965). With respect to noxious gases, 69 processes are listed as scheduled and these processes are required to obtain a registration certificate in order to operate. Dust, smoke and vehicle emissions are also included in the Atmospheric Pollution Prevention Act. These aspects of the Act are administered at local authority level. Aside from the Atmospheric Pollution Prevention Act, there are several provincial Ordinances, in as well as local government ordinances, which prohibit the generation of atmospheric pollution in the form of burning, offensive or unpleasant smells and empower a municipality to promulgate by-laws concerning atmospheric pollution prevention.

6.2.3.2 Institutional arrangements

The diversity of institutions in the control of atmospheric pollution prevention is principally horizontal, i.e., it is sub-divided between the national government (in the form of the Ministry of Environment Affairs and Tourism) and provincial government. The Chief Air Pollution Officer (Chief Officer) within the Department of Environment Affairs and Tourism (DEAT) is empowered to delegate some or all of their powers to a specified person at local authority level. The Act also provides for the administration of smoke, dust and noise regulations by local authorities where they have been declared smoke or dust control areas or where the noise regulations have specifically been made applicable to these areas.

6.2.3.3 Administration

In respect of noxious gasses which are listed in schedule 2 of the Atmospheric Pollution Prevention Act, administration takes place by way of the issuing of a registration certificate in terms of section 9 & 10 of the Act. Dust is controlled by empowering the chief officer to prescribe steps which must be taken to reduce dust generated by industrial processes and which is causing a nuisance to persons in a vicinity of this vicinity is deemed to exceed 20,000 cubic

metres in volume. In special circumstances these steps may be taken and paid for by the State. The Minister is also empowered to develop regulations concerning dust emissions, and these regulations may be delegated for administration to any person so authorised by the Minister.

6.2.4 Solid Waste

6.2.4.1 Existing legislation

Solid waste is principally governed by the Environment Conservation Act 73 of 1989, and in particular section 20 of that Act (DEAT, 1989). However, solid waste is also dealt with in a wide range of other legislation including:

- the advertising on Roads and Ribbon Development Act 21 of 1940 (which prohibits the deposition of waste near specified roads).
- In the National Building Regulations and Building Standards Act 103 of 1977 which empowers a local authority to prohibit the erection of a building which is to covered with refuse
- the Environmental Act 122 of 1984 which prohibits the dumping or scattering of litter in a forest
- the Rural Areas Act (House of Representatives) 9 of 87 which allows the Minister of Housing to plan for a town commonage for the dumping of rubbish soil and the like
- The Minerals Act 50 of 1991 which governs solid waste generated by the Mining Sector
- the Local Government Transitions Act 209 of 1993 which prescribes the powers and duties of Metropolitan Councils in relation to among other things, waste disposal facilities.

There are also various pieces of Provincial legislation dealing with waste.

6.2.4.2 Institutional arrangements:

Although the Environment Conservation Act is administered by DEAT, application for the establishment of a waste disposal site has to be made to the Minister of Water Affairs.

6.2.4.3 Administration

Solid waste management is administered both in terms of the Environment Conservation Act (and the various other pieces of legislation referred to above), and in terms of directives and regulations issued by the Department of Water Affairs and Forestry, not all of which have the force of law. A permit is issued in terms of section 20 of the Water Act and, in administering the Act, the Minister of Water Affairs is obliged to maintain a register in which details of all waste disposal sites for which permits have been issued are recorded. However, many of the waste disposal sites that operate in South Africa are not duly permitted.

6.2.5 Hazardous Waste (and hazardous substances)

6.2.5.1 Existing Legislation

Current legislation regarding hazardous waste is fragmented, and lacks clear identification of responsibilities. In addition there is an obligation under the Basel convention to establish a central focal point and notifying authority. The most relevant statutes are:

- the Hazardous Substances Act (15 of 1973)
- the Fertilisers, Farm Feeds, Agricultural Remedies and Stock Remedies Act (36 of 1947)
- the Environment Conservation Act (73 of 1989) (the "ECA") Part 4 of which is headed *Control of Environmental Pollution*.

6.2.5.2 Institutional arrangements

As the existing hazardous waste laws are fragmented, so to are the institutions which administer them although DEAT has, in practise, taken upon itself to administer certain aspects of waste management. The Department of Water Affairs and Forestry (DWAF) plays a key role as regards permitting of waste disposal sites.

6.2.5.3 Administration:

DWAF administers a key section of the Environmental Conservation Act dealing with waste management (section 20). It imposes permit requirements for the operation of waste disposal sites. Local authorities are included under this provision and are required to comply with it like any private operator. This section is complimented by regulations for waste disposal. In practise the *Minimum Guidelines for the Management of Solid Waste* (DWAF, 1998) are also used although these are not referred to in the legislation.

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Guidelines for using CD-ROM

Guidelines

These guidelines are intended to guide the user through the procedure of copying and opening the files used to model the various flowsheets. These files can be found on the CD-ROM accompanying this thesis. These files are not intended to be a marketable piece of software, as such the interface is unsophisticated. There are three main sets of files and they are stored in three separate directories:

- *Sub-sectors*

This directory contains the models of the sub-sectors of the South African minerals industry

- *Gold Case study Gold*

This directory contains the files used in modelling technology permutations for the case study included in Section 5.

- *Case study - Zinc*

This is the case study that was included in Section 6.

In order to run any of the models it will be necessary to copy the complete directory structure onto the root of the C: drive. The reason for this is that the files are linked in *Excel®* and *Excel®* retains the directory structure in the links between the files.

G.1 Sub-sectors

Copy the entire *Sub-sectors* directory onto the root of your C: drive. There are a number of sub-directories:

- Gold
- Coal
- Base Metals
- Platinum Group Metals
- Ferro Alloys
- Mineral Sands

The structures of these sub-directories is different and so each sub-directory will be described individually.

G.1.1 Gold

There are three files in the *Gold* sub-directory:

- Goldprod.xls

This file contains information on the technology in place in the gold industry. It forms the basis of the mass weighted averages for the assumptions in the unit operations.

- Gold.xls

This file contains the controlling flowsheet, assumption set, input and output listings. Changing assumptions in this sheet will change the information in the Units spreadsheets.

- Units.xls

This file contains the unit operation mass balances. There is one mass balance per spreadsheet.

In order to operate the model all three of these files must be open. For the option "Do you wish to update all linked information" select "Yes".

The overall mass balance can be seen on the *Outputs* spreadsheet in Gold.xls. If the difference between the inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Units.xls
2. Goldprod.xls
3. Gold.xls

If this does not solve the problem then please contact the author of this thesis.

G.1.2 Coal

There are two files in the *Coal* sub-directory:

- Coal.xls

This file contains the controlling flowsheet, assumption set, input and output listings. Changing assumptions in this sheet will change the information in the Units spreadsheets.

- Units.xls

This file contains the unit operation mass balances. There is one mass balance per spreadsheet.

In order to operate the model both of these files must be open. For the option "Do you wish to update all linked information" select "Yes".

The overall mass balance can be seen on the *Flowsheet* spreadsheet in Coal.xls. If different between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Units.xls
2. Coal.xls

If this does not solve the problem then please contact the author of this thesis.

G.1.3 Base Metals

There are ten files in the *Basemet* sub-directory:

- InitConc.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the initial concentration stage of the base metals industry. Changing assumptions in this sheet will change the information in the UnitUnit spreadsheets.

- UnitInit.xls

This file contains the unit operation mass balances for the initial concentrator. There is one mass balance per spreadsheet.

- Copper.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the copper refining stage of the base metals industry. Changing assumptions in this sheet will change the information in the UnitCopp spreadsheets.

- UnitCopp.xls

This file contains the unit operation mass balances for the copper refinery. There is one mass balance per spreadsheet.

- Zinc.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the zinc refining stage of the base metals industry. Changing assumptions in this sheet will change the information in the UnitZinc spreadsheets.

- UnitZinc.xls

This file contains the unit operation mass balances for the zinc refinery. There is one mass balance per spreadsheet.

- Alumin.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the aluminium refining stage of the base metals industry. Changing assumptions in this sheet will change the information in the UnitAlum spreadsheets.

- UnitAlum.xls

This file contains the unit operation mass balances for the aluminium refinery. There is one mass balance per spreadsheet.

- Phosphat.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the Phosphate concentration stage of the base metals industry. Changing assumptions in this sheet will change the information in the UnitPhos spreadsheets.

- UnitPhos.xls

This file contains the unit operation mass balances for Phosphate concentration. There is one mass balance per spreadsheet.

In order to operate the model all ten of these files must be open. The main reason for this is that information from the Initial Concentrator is fed into the Copper and Zinc processes, and the Copper and Phosphate processes are linked. For the option "Do you wish to update all linked information" select "Yes". The macros in the spreadsheet were used in the calculation of recycles.

The overall mass balance for each block in the generalised base metals flow diagram (see Section 2 of this thesis) can be seen on the *Inputs* spreadsheet in the controlling file for that element (i.e., InitConc.xls, Copper.xls, Zinc.xls, Alumin.xls and Phosphate.xls). If different between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. UnitInit.xls
2. UnitCopp.xls
3. UnitZinc.xls
4. UnitPhos.xls
5. Unit Alum.xls
6. InitConc.xls
7. Copper.xls

8. Phosphat.xls

9. Zinc.xls

10. Alumin.xls

If this does not solve the problem then please contact the author of this thesis.

G.1.4 Platinum

There are three files in the *Platinum* sub-directory:

- Platinum.xls

This file contains the controlling flowsheet, assumption set, input and output listings. Changing assumptions in this sheet will change the information in the either in the UnitsBMR (base metals refinery) or the UnitInit (initial concentrator) spreadsheets (please see Appendix 2-5 for more details of these processes).

- UnitInit.xls

This file contains the unit operation mass balances for the initial concentration stage of the platinum industry. There is one mass balance per spreadsheet.

- UnitsBMR.xls

This file contains the unit operation mass balances for the base metals refinery stage of the platinum industry. There is one mass balance per spreadsheet.

In order to operate the model all three of these files must be open. For the option "Do you wish to update all linked information" select "Yes". The macros included are used to calculate the recycles streams.

The overall mass balances for both the initial concentrator and the base metals refinery can be seen on the *Input* spreadsheet in Platinum.xls. If different between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links.

This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. UnitInit.xls
2. UnitsBMR.xls
3. Platinum.xls

If this does not solve the problem then please contact the author of this thesis.

G.1.5 Ferro-Alloys

There are six files in the *Fe-Alloy* sub-directory:

- Vanadium.xls

This file contains the controlling flowsheet, assumption set, input and output listings for both the pyrometallurgical and chemical processing of vanadium. (see Appendix 2-7 for more details on this sector). Changing assumptions in this sheet will change the information in the VanUnits spreadsheets.

- Mangan.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the production of Manganese, Ferro-Manganese, Silico-Manganese and Ferro-Silicon. (see Appendix 2-7 for more details on this sector). Changing assumptions in this sheet will change the information in the ManUnits spreadsheets.

- Steel.xls

This file contains the controlling flowsheet, assumption set, input and output listings for the production of by conventional routes as well as using the Corex® process. (see Appendix 2-7 for more details on this sector). Changing assumptions in this sheet will change the information in the SteUnits spreadsheets.

- VanUnits.xls

This file contains the unit operation mass balances for both the pyrometallurgical and the chemical processing of Vanadium. There is one mass balance per spreadsheet.

- ManUnits.xls

This file contains the unit operation mass balances for the production of Manganese, Ferro-Manganese, Silico-Manganese and Ferro-Silicon. There is one mass balance per spreadsheet.

- SteUnits.xls

This file contains the unit operation mass balances for both the conventional and the Corex ® processes for steel production. There is one mass balance per spreadsheet.

In order to operate the model open all of these files. For the option "Do you wish to update all linked information" select "Yes".

The overall mass balance can be seen on the *Inputs* spreadsheet in each of the controlling files (Mangan.xls, Steel.xls and Vanadium.xls). If different between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. ManUnits.xls
2. VanUnits.xls
3. SteUnits.xls
4. Mangan.xls
5. Vanadium.xls
6. Steel.xls

If this does not solve the problem then please contact the author of this thesis.

G.1.6 Minerals Sands

There is one file in the *Sands* sub-directory:

- Sands.xls

Due to the shortage of information on the mineral sands industry all of the information is included in this one file.

G.2 Gold Case Study

Copy the *Gold Case Study* directory onto your machine. There are four files in this directory:

- Case Study.xls

This file contains information on the economic and environmental performance of the gold industry in South Africa as a whole. This information is used in the normalisation of the objective functions.

- Impacts.xls

These are the impact calculations for the South African Minerals industry as a whole. The information from this file is used in the scaling of the objective functions.

- Case Study - Flowsheet.xls

This file contains all inputs to and outputs from the flowsheet developed. The economic and environmental objective functions are formulated. There is a controlling spreadsheet - *Manipulated Variables* - in this spreadsheet stream splits, number of plants (small and large scale) and objective function weights can be changed.

- Case Study - Units.xls

These are the unit operations for the flowsheet as modelled. These included mass balances as well as the calculation of capital and operating cost for each unit operation.

In order to operate the model open all of these files. For the option "Do you wish to update all linked information" select "Yes".

The overall mass balance can be seen on the *Manipulated Variables* spreadsheet. If the difference between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Case Study - Units.xls
2. Case Study - Flowsheet.xls
3. Impacts.xls
4. Case Study.xls

If this does not solve the problem then please contact the author of this thesis.

G.3 Zinc Case Study

Copy the *Zinc Case Study* directory onto your machine. There are three sub-directories:

- Design Step 2
- Design Step 3 - Before EIA
- Design Step 3 - After EIA

G.3.1 Design Step 2

This design step uses Goal Programming. There are three files in this subdirectory:

- Design Step 2.xls

This file is the controlling spreadsheet. Stream splits can be changed in the *Control* spreadsheet. Goals and weights are also set here.

- Design Step 2 - Flowsheet.xls

This file contains all of the flowsheet information, input-output values and the formulation of the economic and environmental objectives.

- Design Step 2 - Units.xls

These are the mass balances for each unit operation.

The overall mass balance can be seen on the *Control* spreadsheet. If the difference between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Design Step 2 - Units.xls
2. Design Step 2 - Flowsheet.xls
3. Design Step 2.xls

If this does not solve the problem then please contact the author of this thesis.

G.3.2 Design Step 3 - Before EIA

This design step uses Goal Programming on a more comprehensive set of technology permutations, please see details in Section 6 of this thesis. There are three files in this subdirectory:

- Design Step 3 before EIA.xls

This file is the controlling spreadsheet. Stream splits can be changed in the *Control* spreadsheet. Goals and weights are also set here. *Solver* is also set up in this spreadsheet.

- Design Step 3 before EIA - Flowsheet.xls

This file contains all of the flowsheet information, input-output values and the formulation of the economic and environmental objectives.

- Design Step 3 before EIA - Units.xls

These are the mass balances for each unit operation.

The overall mass balance can be seen on the *Control* spreadsheet. If the difference between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Design Step 3 before EIA - Units.xls
2. Design Step 3 before EIA - Flowsheet.xls
3. Design Step 3 before EIA.xls

If this does not solve the problem then please contact the author of this thesis.

G.3.3 Design Step 3 - After EIA

This design step uses Value Functions on set of technology permutations which were taken to the EIA process, please see details in Section 6 of this thesis. There are three sub-directories specific to:

- Option A
- Option B
- Option c

The structure in each of these sub-directories is identical and will be discussed together.

- Design Step 3 after EIA.xls

This file is the controlling spreadsheet. Stream splits can be changed in the *Control* spreadsheet. Goals and weights are also set here. *Solver* is also set up in this spreadsheet. The results from the questionnaire (see Section 6 for discussion) can be found in *Trade-off Results*.

- Design Step 3 after EIA - Flowsheet.xls

This file contains all of the flowsheet information, input-output values and the formulation of the economic and environmental objectives.

- Design Step 3 after EIA - Units.xls

These are the mass balances for each unit operation. Note, the mass flow through these unit operations may be zero dependent on the technology permutation being modelled.

The overall mass balance can be seen on the *Control* spreadsheet. If the difference between inputs and outputs does not equal zero there is a problem. This problem relates to *Excel's* management of the dynamic links. This can usually be solved by closing all of the spreadsheets and then opening them in the following order:

1. Design Step 3 after EIA - Units.xls
2. Design Step 3 after EIA - Flowsheet.xls
3. Design Step 3 after EIA.xls

If this does not solve the problem then please contact the author of this thesis.

