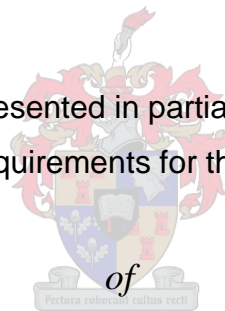# Evaluating the economics of and business models for metal recycling from waste printed circuit boards in a South African context

by

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MASTER OF ENGINEERING

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Supervisor

Prof. C. Dorfling

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ABSTRACT

Electronic waste is the fastest growing component of solid municipal waste in South Africa however limited processing capacity exists in the country to economically recover metals from printed circuit board (PCB) waste. Presently the majority of recycled e-waste is exported instead of adding value to the processed material.

A hydrometallurgical process entailing the selective leaching of gold and copper from waste printed circuit boards was developed by the research group (Rossouw, 2015; de Waal, 2018). This study sought to evaluate the economic and business viability of the proposed hydrometallurgical process within the South African context.

A PESTEL approach was used to evaluate the South African recycling landscape. In this approach the status of the political, economic, environmental, legal, social and technological aspects characterising e-waste recycling and management was undertaken.

Different business models were developed around the main activities of the associated operations of the proposed process. Screening and selection was done on two levels. An assessment of the collection and dismantling of e-waste was done and used as the first basis for screening the business models. It was established that the reverse logistics and associated manpower requirements and efficiencies associated with this operation were economically and operationally restrictive.

Detailed costing and profitability studies were done on the last eight business models. The recovery of copper and gold in the metallic state was found to be associated with the highest capital and annual operating expenditures.

Profitability studies were conducted on two levels. Performance analysis over the life of the project was done on the basis of discounted cash flows, time, cash and interest. The profitability ratios that included return on investment (ROI), capital ratio (CR), terminal capital rate of return ratio (CRR$_{term}$), present value ratio (PVR), net profit margin (NPM) and the (fixed asset turnover ratio) FATR were used to assess the annual performance of each business model.

Both assessments revealed that all the business models under the specified process conditions, were unviable and incapable of both capital and interest repayment. Detailed sensitivity studies of the group 4 business was done.

The risk of failure within five years was certain as evidenced by the perpetually decaying plots of the recapitalization risk curves. Four additional significant areas of risk were identified and evaluated that had an impact on the performance of the business models. These included supply and demand constraints, key raw material, technological and market related risks.

A consideration of alternative revenue streams from palladium, silver and tin recovery, did not improve the performance of the business models. Decreasing both the capital costs and operating costs by margins ranging between 48% and 65% shifted the performance of the business models. The success of this project was driven strongly by the gold business, evidenced by improved performance at elevated minimum gold concentrations ranging between 563ppm and 925ppm. These concentrations implied that a consistent supply of high grade WPCBs would have to be furnished to ensure project viability. It is necessary to consider alternative processing routes when using medium grade WPCBs.

OPSOMMING

Elektroniese afval is die vinnigste groeiende komponent van soliede munisipale afval in Suid-Afrika. Beperkte prosesseringskapasiteit bestaan egter in die land om metale uit gedrukte stroombaanpaneel (PCB) afval ekonomies te herwin. Tans word die meerderheid herwinde e-afval uitgevoer in plaas daarvan om waarde tot die geprosesseerde materiaal te voeg.

'n Hidrometallurgiese proses wat die selektiewe loging van goud en koper uit afval PCB behels, is deur die navorsingsgroep ontwikkel (Rossouw, 2015; de Waal 2018). Hierdie studie het beoog om die ekonomiese en besigheidslewensvatbaarheid van die voorgestelde hidrometallurgiese proses binne die Suid-Afrikaanse konteks te evalueer.

'n politiese, ekonomiese, sosiale, tegnologiese, omgewing en wettig (PESTEL) -benadering is gebruik om die Suid-Afrikaanse herwinningslandskap te evalueer. In hierdie benadering is die status van die politiese, ekonomiese, omgewings-, wettiglike, sosiale en tegnologiese aspekte wat die e-afval herwinning en bestuur karakteriseer, onderneem.

Verskillende besigheidsmodelle is ontwikkel rondom die hoofaktiwiteite van die geassosieerde bedrywighede van die voorgestelde proses. Sifting en seleksie is op twee vlakke gedoen. 'n Assessering van die versameling en afbreking van e-afval is gedoen en gebruik as die eerste basis vir sifting van besigheidsmodelle. Dis vasgestel dat die omgekeerde logistiek en geassosieerde mannekrag vereistes en doeltreffendhede geassosieer met hierdie bedryf ekonomies en operasioneel beperkend is.

Gedetailleerde kosteberekeninge en winsgewendheidstudies is gedoen op die laaste agt besigheidsmodelle. Dis gevind dat die herwinning van goud en koper in die metaal toestand geassosieer word met die hoogste kapitaal en jaarlikse bedryfsuitgawes.

Winsgewendheidstudies is uitgevoer op twee vlakke. Werkverrigting analise oor die lewensduur van die projek is gedoen op die basis van gediskonteerde kontantvloei, tyd, kontant en rente. Die winsgewendheid verhoudings wat opbrengs op belegging (ROI), kapitaal verhouding (CR), terminale kapitaal opbrengskoers verhouding (CRR_{term}), toonwaarde verhouding (PVR), netto winsgrens (NPM) en die FATR (vaste bate omset verhouding) is gebruik om die jaarlikse doeltreffendheid van elke besigheidsmodel te assesseer.

Beide assesserings het aangedui dat al die besigheidsmodelle binne die gespesifiseerde proses kondisies nie lewensvatbaar was nie en onbevoeg is om beide kapitaal en rente terugbetalings te doen. Gedetailleerde sensitiwiteitstudies van die groep vier besigheid is gedoen.

Die risiko van mislukking binne vyf jaar was seker soos bewys deur die onophoudelike afbreking stippe op die herkapitalisering risiko kurwes. Vier addisionele beduidende areas van risiko is geïdentifiseer en -evalueer wat 'n impak op die prestasie van die besigheidsmodelle gehad het.

Hierdie sluit in vraag en aanbod beperkinge, sleutel rou materiaal, tegnologiese en mark verwante risiko's.

'n Oorweging van alternatiewe inkomstestrome uit palladium, silwer- en tinherwinning het nie die doetreffendheid van die besigheidsmodelle verbeter nie. Deur beide die kapitaalkostes en bedryfskostes met marges binne bestek van 48% en 65% te verminder, het die doeltreffendheid van die besigheidsmodelle geskuif. Die sukses van hierdie projek is grootliks gedryf deur die goudbedryf, soos bewys deur verbeterde doeltreffendheid by verhoogde minimum goudkonsentrasies binne bestek van 563 ppm en 925 ppm. Hierdie konsentrasies dui aan dat 'n konsekwente toevoer van hoë-graad WPCBs (waste printed circuit boards) ingerig moet word om die projek se lewensvatbaarheid te verseker. Dit is nodig om alternatiewe prosesseringsroetes te oorweeg wanneer medium-graad WPKCBs gebruik word.

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I give all gratitude to YHWH the Father of our Lord Jesus Christ through whom all things in creation exist. Nothing happens without His knowledge He alone is the true source of wisdom.

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ABBREVIATIONS AND ACRONYMS

General abbreviations and acronyms

Symbol	Meaning and definition
ADF	Advance disposal fee
AHP	Analytical hierarchy process
ARF	Advance recycling fees
APME	Advanced Polymers, Macromolecular Engineering
BGS	British Geological Service
CAA	Clean Air Act
CWA	Clean water act
CERCLA	Comprehensive Environmental Response, Compensation and Liability Act
DEA	Department of Environmental Affairs
DEAT	Department of Environmental Affairs and Tourism
DP	Development Planning
DRC	Democratic Republic of Congo
ECA	Environment Conservation Act
ECU	Electronic control unit/s
EIA	Environmental impact assessment
ELV	End of life vehicle
EMPA	Swiss Federal Laboratories for Materials Science and Technology
EOL	End of life
EPR	Extended producer responsibility
EU	European Union
eWA	e Waste Alliance
e-WASA	E- waste association of South Africa
GIZ	Deutsch Gesellschaft für Internationale Zusammenarbeit
GKPER	Global Knowledge Partnerships in e-waste Recycling
ICCM	International Conference on Chemicals Management
ICs	Integrated circuits
IESSA	Illumination engineering society of South Africa
IMDS	A vehicle data management system
ITA	Information technology association
IndWMP	Industrial Waste Management Plan
IP& WM	Integrated Pollution and Waste Management
IWMSA	Institute of Waste Management Southern Africa
IndWMP	Industrial Waste Management Plan

Symbol	Meaning and definition
IP& WM	Integrated Pollution and Waste Management
IWMSA	Institute of Waste Management Southern Africa
LCA	Life cycle analysis
LCC	Life cycle costing
MCA	Multi criteria analysis
MFA	Material flow analysis
MRA	Metal Recyclers Association of South Africa
MTN	Mobile Telephone Network
NEMA	The National Environmental Management Act
NGO	Non-governmental organization
NRF	National Recycling Forum
NRF*	National Research Foundation
NWMS	National Waste Management Summit
OECD	Organisation for Economic Co-operation and Development
OHSA	Occupational Health and Safety Act
Pa	Pascals
PC	Personal computer
pH	Potential of hydrogen ions
PWB	Printed wiring boards
RAG	Recovery Action Group
RCRA	Resource Conservation and Recovery Act
RoHS	Restriction on the use of hazardous substances
RRIs	Resource recovery indices
SA	South Africa
SADC	Southern African Development Community
SAICM	Strategic Approach to International Chemicals Management
SAS	An American software company
SAWIS	South African Waste information system
SECO	Swiss State Secretariat for Economic Affairs
SMEs	Small and medium enterprises
SWEEEA	South African Waste Electrical and Electronic Association
TV	Television
UK	United Kingdom
UNEP	United Nations Environmental Program
UNICEF	United Nations Children's Fund
USEPA	United States Environmental Protection Agency
USGS	United States Geological Service

Symbol	Meaning and definition
US	United States
WBCSD	World Business Council for Sustainable Development
WCRAG	WCRAG Western Cape Recovery Action Group
ZimPhos	Zimbabwe Phosphate Industries

Economic abbreviations and acronyms

Symbol	Meaning and definition
A_{CF}	Annual cash flow
A_{DCF}	Annual discounted cash flow
A_d	Annual depreciation charge
BEP	Breakeven point
f_d	Discount factor
f_k	Capitalized cost factor
CAPEX	Capital expenditure/initial capital outlay
CBA	Cost benefit analysis
C_F	Fixed costs
C_{FC}	Fixed capital investment
C_K	Capitalised cost of equipment
C_L	Cost of land
C_{TC}	Total capital cost
C_R	Replacement cost
CRR	Capital rate of return ratio
CRR_{term}	Terminal capital rate of return ratio
C_{WC}	Working capital cost
DBEP	Discounted breakeven point
DCF	Discounted cash flow
DCFRR	Discounted cash flow rate of return
$DCFRR_{max}$	Discounted cash flow rate of return when project life has an unlimited number of years
EMIP	Equivalent maximum investment period
ERP	Equity risk premium
i	interest
IRP	Interest recovery period
IRR	Internal rate of return
JSE	Johannesburg stock exchange
LME	London metal exchange
LSE	London Stock exchange
n	Years
NPV	Net present value
OPEX	Operating expenditure
PBP	Payback period
PESTEL	Political, economic, sociological, technological, environmental, legal.
PVR	Present value ratio

Symbol	Meaning and definition
ROR	Rate of return
SWOT	Environmental analysis tool: Strengths, Weaknesses, Opportunities and Threats
S/V_s	Salvage or scrap value
∑	Summation of
WACC	Weighted average cost of capital

Technical abbreviations and acronyms

Symbol	Meaning and definition
Ag	Silver
$Ag(CN)_2^-$	dicyanidoargentate
AgS	Silver sulphide
Al	Aluminium
$Al(OH)_6^{3-}$	Hexaaquaaluminium ion
Al_2O_3	Aluminium trioxide
Al_2S_3	Aluminium sulphide
AVR	Acidification, volatilization and recovery
Au	Gold
$Au(CN)_2^-$	dicyanoaurate complex
Au_2Pb	Hunchunite
$AuPb_3$	Novodneprite
Bi	Bismuth
Br	Bromine
Ca	Calcium
$CaCO_3$	Calcium carbonate
CaO	Calcium oxide
$Ca(OH)_2$	Calcium hydroxide
CO_2	Carbon dioxide
Ce	Cerium
CIS	Carbon in solution
Cl^-	Chloride ion
cm	Centimetre
Co	Cobalt
$Co(CN)_6^{3-}$	Cobalt hexacyanide ion
Cu	Copper
$CuCl_2$	Copper II chloride
$Cu(CN)_3^{2-}$	Cupric tricyanide complex
$CuCO_3$	Copper II carbonate
CuO	Copper II oxide
CuS	Copper sulphide
CRTs	Cathode ray tubes
CN^-	Cyanide ion
DLC	Diffusion limiting current
ΔG	Change in free energy (Delta G)

Symbol	Meaning and definition
^{O}C	Degrees Celsius
EEE	Electrical and electronic equipment
E-pH	Electrode potential & pH
Fe	Iron
$Fe(CN)_6^{4-}$	Iron hexacyanide ion
$Fe(OH)_3$	Ferric hydroxide
FeO	Ferrous oxide
FeS	Ferrous sulphide
Ge	Germanium
g/L	Grams per litre
GW	General waste
HCl	Hydrochloric acid
$Hg(CN)_4^{2-}$	Mercury (II) tetracyanide complex
HNO_3	Nitric acid
H_2O_2	Hydrogen peroxide
H_2S	Hydrogen sulphide
H_2SiO_3	Metastannic acid
H_2SO_4	Sulphuric acid
HC	Harvest cycle
Hp/hp	Name plate horse power
hp_b	Full load horse power
HW	Hazardous waste
ICT	Information and communication technology
In	Indium
ITO	Indium tin oxide
IT	Information technology
Kg	Kilogram
KNO_3	Potassium nitrate
KW/h	Kilowatt/hours
La	Lanthanum
LCA	Life cycle assessment
LCC	Life cycle costing
LCD	Liquid crystal display
LEDs	Liquid electronic displays
Li	Lithium

Symbol	Meaning and definition
$LiCoO_2$	Lithium cobalt oxide
$LiFePO_4$	Lithium iron phosphate
$LiMn_2O_4$	Lithium manganese oxide
$LiNiO_2$	Lithium nickel dioxide
M	Molar
mm	Millimetre
Mn	Manganese
MCA	Multi criteria analysis
MFA	Material flow analysis
µmol/g	Micro mole per gram
MLCC	Multilayer ceramic capacitor
MnO	Manganese oxide
MnO_2	Manganese IV oxide
Na	Sodium
NaCl	Sodium chloride
NaClO	Sodium hypochlorite
$NaClO_3$	Sodium chlorate
NaHS	Sodium hydrosulphide
NaOH	Sodium hydroxide
NaS_2O_3	Sodium thiosulphate
Nd	Neodymium
NH_3	Ammonia
NH_5CO_3	Ammonium bicarbonate
NH_4CO_3	Ammonium carbonate
$(NH_4)_2SO_4$	Ammonium sulphate
$(NH_4)_2S_2O_3$	Ammonium thiosulphate
Ni	Nickel
Ni-Cd	Nickel cadmium
$Ni(CN)_4^{2-}$	Tetracyanonickelate complex ion
NiMH	Nickel metal hydride
NO	Nitrogen oxide
NO_2	Nitrogen dioxide
OCD	Operating current density
OLED	Organic light-emitting diode
P	Phosphorus
OLED	Organic light-emitting diode

Symbol	Meaning and definition
Pb	Lead
$PbCl_2$	Lead II Chloride
$Pb(CN)_4^{2-}$	Lead (II) tetracyanide
PbO	Lead oxide
β PbO	Beta lead oxide
$Pb(OH)_2$	Lead hydroxide
$Pb(OH)_3^-$	Lead trihydroxide ion
PbS	Lead sulphide
$PbSO_4$	Lead II sulphate
PBDD	Polybrominated dibenzo-p-dioxins
PBDE	Polybrominated diphenyl ethers
PBDF	Polybrominated dibenzo furans
PCB	Printed circuit board
PC	Personal computer
Pd	Palladium
PE	Polyethylene
PEL	Permissible exposure limits
PGMs	Platinum group metals
PLS	Pregnant leach solution
PMs	Precious metals
PP	Polypropylene
ppm	Parts per million
PWB	Printed wiring board
REE	Rare earth elements
REO	Rare earth oxides
Ru	Ruthenium
SART	Sulphuridization, Acidification, Recovery, Thickening acronym for post cyanide leaching of copper rich liquors in gold processing
Sb	Antimony
SDS	Safety data sheets
Se	Selenium
SiO_2	Silicon dioxide
Sn	Tin
SnO_2	Tin II oxide
SnS_2	Tin (IV) sulphide
SO_2	Sulphur dioxide
TBB	Tetrabromobenzotriazole

Symbol	Meaning and definition
TBBPA	Tetrabromobisphenol-A
TBP	Tributylphosphate
TDS	Total dissolved solids
Te	Terullium
TWA	Time weighted average
WAD	Weak acid dissociable
WEEE	Waste electrical and electronic equipment
WHS	Working hours of stripping machine
WOC	Weight of one sided copper cathode deposit
WPCB/s	Waste printed circuit board/s
Y	Yttrium
Zn	Zinc
$Zn(OH)_2$	Zinc hydroxide
$Zn(OH)_4^{2-}$	Zincate complex
ZnS	Zinc sulphide

Chapter 1:   INTRODUCTION

1.1. Project background

Electronic waste is the fastest growing municipal waste globally (Schluep *et al.,* 2009). This presents opportunities to exploit the rich source of metals found in waste printed circuit boards. The same remarks can be made for South Africa as alluded to by the Department of Science and Technology (DEA, 2012).

WPCBs are composed of metallic and non-metallic material of which the bulk of the value realized is derived from the extraction of base and precious metals. Researchers have explored and described several methods that include mechanical, bio-metallurgical, pyrometallurgical and hydrometallurgical processes to recover these metals (Kamberović *et al.,* 2011; Rossouw, 2015; Birloaga *et al.,* 2016).

Waste valorisation through waste printed circuit board (WPCB) recycling is limited by the existing processing capacity and related infrastructure. At present no prospects for major capital expenditure exist owing to the low inflows of raw material as reported by major findings of recent surveys (Lydall *et al.,* 2017; GreenCape 2017; 2018).

The existent players in the field are concentrated upstream of the e-waste recycling chain. The major activities upstream of the recycling chain are associated with the collection, dismantling and refurbishment of the e-waste.

1.2. Project scope

There exists scope for increasing the value realisable from hydrometallurgical recycling of WPCBs. The detailed economic studies of the hydrometallurgical process and associated business models have not been studied for processing capacities warranting the installation of field plants. Most evaluations on a large scale have focused on pyrometallurgical processes (Hagelüken, 2006; Ghodrat *et al.,* 2016). Economic evaluations have been undertaken for mobile plants that process not more than 2000 tonnes per annum of WPCBs in South Africa (Lydall *et al.,* 2017). This project seeks to also identify and relate the internal and external factors that affect the development of business models suited to the South African e-waste recycling landscape and existing metals processing facilities in the country. Further afield, other researchers have studied the economics of proposed small scale hydrometallurgical processes for precious and base metals recovery (Kamberović *et al.,* 2011; Birloaga *et al.,* 2016). Opportunity exists for the creation of more jobs from e-waste recycling activities in South Africa (Godfrey *et al.,* 2015; Lydall *et al.,* 2017; GreenCape, 2018).

1

1.3.   Project objectives

A consideration of the best available technologies for recycling WPCBs was done. A study was then undertaken to investigate the effect of mechanical pre-treatment on the leaching of base metals from waste printed circuit boards (Rossouw, 2015). The findings of the investigation showed that there was no significant improvement in the leaching performance following pre-treatment of the WPCBs to concentrate the metal fractions. The results however revealed that scope existed for selectively leaching the base metals, prior to the recovery of the precious metals. The investigations formed the foundation upon which a hydrometallurgical process was developed to selectively leach base metals and recover the copper and gold in two separate leaching campaigns (de Waal, 2018).

Taking into consideration the material flows and existing legislation within South Africa, the economics of selective base and precious metals leaching must be examined to ascertain its viability and sustainability over a 20 year project life span. This project aims to add a capstone in the decision-making process by examining the economic viability of the proposed process flow sheets while exploring business models applicable to the South African context. Further, this study aims to determine the best possible way to position businesses in the e-waste recycling industry in the country.

In order to effectively examine the economic feasibility of the proposed hydrometallurgical process and its applicability in the South African context, the project was approached with three major objectives in mind which entailed:

- The development of an understanding of the current position of e-waste management and legislation affecting e-waste recycling in South Africa.
- Defining and comparing different business models for WPCB metals recycling.
- Evaluating and assessing the economic performance of each business model based on the proposed hydrometallurgical process.

1.4.   Findings and status of supporting projects

Research exploring the recovery of copper and gold from waste printed circuit boards was pioneered by investigations undertaken by previous researchers. Rossouw (2015) examined the impact of pre-treatment on the leaching of base metals in waste printed circuit boards (WPCBs). Two major options were presented for the recovery of copper and gold. The first incorporated a metals enrichment stage through physical processes. The second option did not include further physical pre-treatment after size reduction of the WPCBs.

It was noted that while the pre-treatment of the crushed solids allowed an enrichment of the lighter fractions with gold, some of the copper also reported to this fraction. It was therefore necessary to conduct selective leaching of the base metals. For the conditions specified, using nitric acid and sulphuric acid as the chief lixiviants, the effect of pre-treatment on the leaching performance was not altered significantly (Rossouw, 2015). In light of the arguments presented the second option without physical pre-treatment was selected for further development. Having defined optimum leaching conditions for the selective removal of base metals, the first preliminary flow sheet was developed. The flow sheet however did not include the detailed recovery of gold and other precious metals.

It has been demonstrated by researchers that the hydrometallurgical recovery of precious metals from WPCBs is technically feasible (Kamberovič *et al.*, 2011; Birloaga *et al.*, 2011; 2013; Rossouw, 2015).

Building on the previous investigations, a detailed hydrometallurgical process, for selectively leaching base metals prior to recovering gold was developed. The findings and proposed flow sheet (Rossouw, 2015) were developed further by de Waal (2018).

A pyrometallurgical process rated at an operating capacity of 30 000 tonnes per annum was designed by Ghodrat *et al.,* (2016). The researchers analysed the economic performance of the specified process. The potential available to recycle e-waste at this capacity motivated a comparative evaluation of a hydrometallurgical process for recycling e-waste, in this case WPCBs at a similar operating capacity. Detailed process flow sheets, accompanied by mass and energy balances to selectively leach solder with the subsequent recovery of copper and gold were therefore completed using an operating basis of 30 000 tonnes of WPCBs per annum. The scale of operations of the proposed hydrometallurgical process was reviewed with a consideration of the availability of the key raw material in the e-waste recycling chain and the available recycling activity that could feed into the main operations. Findings of the due diligence conducted on the business environment revealed that the operating scale would have to be reduced. A recycling rate of 400tonnes of WPCBs was used in the study.

The main process, operating parameters and major equipment requirements were specified. Using a commercially established technology, cyanide leaching of gold was selected as the means to extract the gold from the solids stripped of the base metals (de Waal, 2018).

1.5. Project methodology

The due diligence of the proposed operating capacity was undertaken in which a consideration of the raw material availability This study was undertaken for a plant rated at a processing capacity of 400 tonnes of WPCBs per annum.

1.5.1.   Literature review

A detailed literature study was undertaken which began with a discussion of printed circuit boards and their types based on the methods of manufacture. The valuation and generation of WPCBs was considered. A review of mechanical and hydrometallurgical recycling of WPCBs was also considered in the discussion. The section was concluded by a detailed review of the economics associated with recycling WPCBs.

1.5.2.   E-waste management in South Africa

The analysis of the South African e-waste recycling landscape was undertaken with an examination of the dominant technologies used for recycling e-waste. The first research objective was addressed in this section and it enabled a holistic understanding of the prevailing status of e-waste recycling and management in the country. In order to achieve this objective, a detailed desktop study was undertaken to examine the factors that affect e-waste management and its recycling in South Africa as a whole.

1.5.3.   Process description and design considerations

Findings of the assessment of the South African landscape revealed that there were low barriers to entry downstream of the e-waste recycling chain with respect to value adding activities. It was revealed that an opportunity existed to exploit this observed deficit along the e-waste recycling and value chain. The output of the previous and ongoing studies formed the foundation for the current project. The data from the previous and current research was used to specify the process parameters and plant operating considerations used for developing the proposed hydrometallurgical process.

1.5.4.   Considerations for project economic evaluations

Economic considerations used in the study were covered in chapter five. In this same chapter the assumptions used for the capital and operating cost estimates were also discussed. The development of production plans and specification of the organograms relevant for each production plan was also outlined in this section. Assumptions related to the plant maintenance and utilities were included.

1.5.5.   Development and selection of business models

Sixteen business models were defined based on the criteria that were described in chapter six. Key metrics influencing the strategic competitive position of the proposed venture was considered. The output of the study formed the initial basis upon which the business models were evaluated and screened.

1.5.6. Project costing and profitability assessments

The costing and economic evaluations were done for the selected business models using the assumptions discussed in chapter five. The profitability of the business models was evaluated on an annual basis and over the entire project life. Further screening of the remaining business models was done based on the output of the economic study.

1.5.7. Sensitivity and risk analysis

Chapter eight was committed to a detailed sensitivity analysis of the selected business models in order to examine the viability of the proposed venture in response to macro and microeconomic influences. The findings were used to assess the robustness of the proposed process as a going concern in the face of intrinsic and extrinsic influences. An assessment of susceptibility to environmental risk factors for the selected business models was supported by the output of the sensitivity analysis.

1.5.8. Conclusions and recommendations

Conclusions and recommendations were specified from the output of the studies covering an assessment of the South African e-waste recycling landscape and the economic evaluation of the business models specified for the hydrometallurgical recycling of WPCBs.

Chapter 2:   LITERATURE REVIEW

2.1. Introduction

Several benefits are associated with e-waste recycling. Some of the benefits include materials recovery, diversion of waste volumes from landfill, curbing potential illicit waste disposal practices, realisation of potential energy savings and socio-economic benefits associated with the recycling chain. The extraction of metals from low grade ore bodies is associated with the high energy usage and a significant carbon foot print. While e-waste is a rich source of metals in concentrations often four times or more than their respective native ores (Hagelüken, 2006). Metals recovery from polymetallic urban ores such as, waste printed circuit boards (WPCBs) is characterised by a lower carbon and energy footprint (Cui & Forrsberg, 2003; Wang & Gausted, 2012; Akcil *et al.*, 2015). Substantial research work combining different metallurgical processes has been undertaken. The investigations are accompanied by detailed economic project evaluations (Ghodrat *et al.*, 2016).

2.2. Printed circuit boards

2.2.1. Types of printed circuit boards

Printed circuit boards (PCBs) are complex polymetallic modules comprised of interconnected electronic components with circuits printed onto a non-conducting substrate. Connectors in the form of contact fingers enable PCBs and electrical devices to be joined together. There are three major types of PCBs and they include: single sided, double sided and multi-layered PCBs. Table 2.1 below classifies the PCBs based on their mechanical properties (streamlinecircuits, 2016).

Table 2.1: Classification of printed circuit boards (PCBs) based on mechanical properties

Type	Description
Single sided PCBs	This type has one layer of substrate onto which a thin layer of metal covers it. Electronic components and circuits are found on one side.
Double sided PCBs	Both sides of the substrate are lined with a thin layer of metal. Through hole or surface mount technology can be applied to these PCBs.
Multilayer PCBs	Extension of double sided PCBs. The number of layers is a function of the applicability of the board. The holes can be used to interconnect the boards to each other.
Rigid PCBs	This type has a substrate material composed of hard inflexible material chiefly fibre glass.
Flex PCBs	This type is made of substrate material that is usually a flexible plastic single, double or multilayer.
Rigid-Flex	These combine the properties associated with rigid and flex PCBs.

2.2.2. Characterisation of waste printed circuit boards

The metallic composition of a printed circuit board is dictated by the functional capabilities it has been designed to achieve. The varied metallic composition is attributed to the different conductivities and thermal stabilities associated with some of the metals which influence the extent to which they are incorporated in the manufacture of the specific printed circuit board.

Table 2.2 shows the variation of the metal compositions of WPCBs from different sources across the spectrum of the different e-waste classifications. E- waste sources from ICT equipment and personal electronic gadgets have a higher precious metals content than the larger electronic and electrical consumer goods.

Table 2.2: The metal compositions (%) of WPCBs from nine different sources (Excerpt from Cucchiella *et al.,* 2016)

Metal	Fridge	Washing Machine	Air con.	Desktop PC	Note book PC	Mobile phone	CRT TV	Stereo Sys.	Digital Cam.
Iron	2.10	9.50	2.00	1.30	3.70	1.80	3.40	1.20	3.00
Copper	17.00	7.00	7.50	20.00	19.00	33.00	7.20	15.00	27.00
Silver	-	-	-	-	0.10	0.40	-	-	0.30
Gold	-	-	-	-	0.10	0.20	-	-	0.10
Aluminium	1.60	0.10	0.70	1.80	1.80	1.50	6.20	2.90	2.40
Barium	-	-	-	0.20	0.60	1.90	0.20	0.10	1.60
Chromium	-	-	-	-	0.10	0.10	-	-	0.30
Lead	2.10	0.20	0.60	2.30	1.00	1.30	1.40	1.90	1.70
Antimony	0.30	-	-	0.20	0.10	0.10	0.30	-	0.20
Tin	8.30	0.90	1.90	1.80	1.60	3.50	1.80	2.20	3.90
Zinc	1.70	0.20	0.50	0.30	1.60	0.50	5.30	1.40	0.90

Table 2.3 provides a summary of the typical and variant WPCB metallic compositions as analysed by researchers over a 21 year period. The different WPCB samples analysed by the various researchers were subjected to different forms of mechanical pre-treatment. Between 10% and 35% of the precious metals in WPCBs reports to the fines fraction during the comminution stage (Hagelüken, 2006; Yazici, 2015). Ogunniyi *et al.,* (2009) noted that no standard literature is in existence that provides a baseline frame of reference by which methods of WPCBs can be analysed and characterised. The polymetallic nature of WPCBs further complicates the development of a standard frame of reference for WPCBs. Spectroscopic methods however yield good results for qualitative assays of WPCBs (Ogunniyi *et al.,* 2009).

Table 2.3: The typical and variant WPCB metallic compositions

Researchers	Ag (ppm)	Cu	Sn	Pb	Ni	Au (ppm)	Mn	Sb	Al	Fe	Zn	Ca	Pd (ppm)
Silvana *et al.,* (2013)	43.70	16.68	5.23	0.82	0.49	12.83	0.003	0.008	0.26	4.87	7.69	0.06	
Birloaga *et al.*, (2013)	668.00	30.57	7.36	6.70	1.58	238.00			11.69	15.21	1.86		
Birloaga *et al.*, (2013) second sample	689	31.77	6.98	4.92	1.96	369			10.54	13.77	2.23		
Yang *et al.,* (2009)		25.06	1.86	0.80	0.003				4.65	0.66	0.04		
Oishi *et al.,* (2007)		26.00	4.90	3.00	1.50		0.11	0.16	3.20	3.40	2.60		
Behnamfard *et al.,* (2013)	704.31	19.19	0.69	0.39	0.17	130.25	0.04	0.37	4.01	1.13	0.84		
Park & Fray (2008)	0.10	16.00	3.00	2.00					5.00	5.00	1.00		0.01
Hao *et al.,* (2008)	800.00	23.73	3.65	4.48	3.32	800.00		1.82	4.70				210.00
Ogunniyi & Vermaak (2009)	3301.00	23.47	1.54	0.99	2.35	570.00			1.33	1.22	1.51		294.00
Ogunniyi & Vermaak (2007)	1000.00	20.00		1.50	1.00	250.00			5.00	7.00			110.00
Hagelüken (2006)	2000.00	20.00	4.00	2.00	2.00	1000.00		0.40	2.00	8.00	1.00		50.00
Shuey & Taylor, (2004)	2000.00	20.00	4.00	2.00	2.00	1000.00		0.40	2.00	8.00	1.00		50.00
Sum (1991)	3300.00	26.80	1.00					0.06	1.90	5.30	1.50		
Zhao *et al.,* (2004)	110.00	10.00		1.20	0.85	280.00			7.00	12.00	1.60		
Zhang & Forrsberg (1997)	1240.00	15.60	3.24	1.35	0.28	420.00				1.40	0.16		10.00
Kim *et al.,* (not dated)		22.00	2.60	1.55	0.32	350.00				3.60			
Iji & Yokoyama (1997)	1300.00	17.85	5.28	4.19	1.63	350.00			4.78	2.00	2.17		250.00
Cui & Zhang (2008)	280.00	10.00		1.20	0.85	110.00			7.00	12.00			
Guo *et al.,* (2009)	3300.00	26.08	1.00		0.47	80.00			4.70	5.30	1.50		
Goosey & Kellner (2002)	0.05	16.00			2.00	0.03				3.00			0.01
Sum *et al.,* (2005)	0.20	20.00	4.00	2.00	2.00	0.10				8.00	1.00		0.005
Albertyn (2017)	589.50	26.42	5.45	1.23	0.44	158.00				0.71	2.07		

2.2.3.   Valuation of waste printed circuit boards (WPCBs)

The purchase price of unprocessed e-waste has been estimated to range between US$375 to US$500 per tonne and this variation is largely due to the grade of the e-waste (UNEP, 2007).The value of printed circuit boards is predominantly a function the intrinsic metal value embedded within it (Hagelüken, 2006). Three economic classifications of WPCBs were discussed by Hagelüken (2006). The classifications are based on the amount of gold contained in the WPCBs and are described in Table 2.4:

Table 2.4: Economic classification of WPCBs (Hagelüken, 2006)

Grade	Typical gold content (ppm)	Examples
Low	< 100	TV-boards, monitor boards, phones, calculators, shredded bulk material (Al and Fe free)
Medium	100 to 400	PC-boards, lap-tops and hand held computers, some mobile phones
High	> 400	Some mobile phones, ICs, MLCCs, circuit boards from main frames

Figure 2.1 shows the value distribution expressed in US dollars for the different types of WPCBs. Typical assays of WPCBs from TV boards, PC boards and PC main boards were used for the analysis (Cui & Zhang, 2008; USGS, 2017; Infomine, 2017).
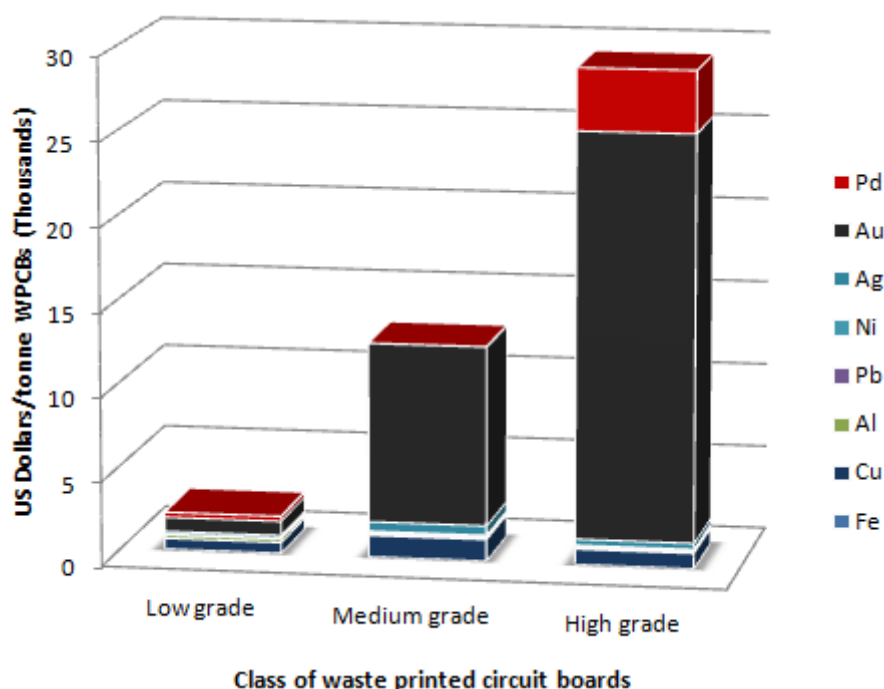
Figure 2.1: An analysis of the potential revenue realisable based on sales of pure metals using LME prices in low, medium and high grade WPCBs (Cui & Zhang, 2008; USGS, 2017; Infomine, 2017)

Cui & Zhang (2008) undertook a similar analysis of differing sources of electronic scrap. The materials analysed also included WPCBs from varied sources. The findings allowed the researchers to make a comparative illustration of the valuation of the electronic scrap. It was shown that WPCBs derived from mobile phones and PCs attracted the highest value. This value was derived from the precious metals content. A comparison of the gold content in each sample showed a similar trend observed by other workers (Legarth *et al.,* 1995; Zhang & Forrsberg, 1997; Theo, 1998; Hagelüken, 2006; Cui & Forrsberg, 2003; CPCB, 2007).

2.2.4. Environmental impacts of waste printed circuit boards handling

Waste printed circuit boards are made up of metallic and non-metallic materials that are classified as hazardous substances. An analysis of the current epidemiological data on the population of Guiyu China has linked the bioaccumulation of heavy metals in the residents of Guiyu to the pollution, both aquatic and terrestrial arising from e-waste recycling activity in the town. Results of soil assays within and around the town also show comparatively elevated levels of heavy metals associated with processing of e-waste.

Akcil *et al.,* (2015) identified significant deleterious effects of e-waste recycling activities that include, the generation of pollutant gases and acid effluent streams laden with heavy metals, the cumulative deterioration of the quality of terrestrial, human and aquatic life, depletion of useful and safe land space and the declining life expectancy of local communities and fauna.

Hazards and impacts emanating from e-waste recycling activities are derivatives of the primary, secondary and tertiary contaminants. The primary hazards and impacts are derived from the waste itself. These include the heavy metals contained within and the plastics impregnated with flame retardants (Schluep *et al.,* 2009). The flame retardants include tetrabromobisphenol-A (TBBPA) and polybrominated diphenyl ethers (PBBs). These substances have been associated with the generation of cancer precursors during low temperature combustion of the plastic materials in which they are contained.

Secondary hazardous components are generated from the recycling methods typically associated with informal practices. These hazards and impacts manifest as contaminated effluent discharges, toxic emissions of combustion activities and dumping of scavenged e-waste components. The effluent discharges are characteristically acidic and this promotes the migration of heavy metals from residual e-waste into both the stagnant and mobile water bodies (Nnorom & Osibanjo, 2008). The partial processing of e-waste also presents the additional risk of leaching the flame retardants from the matrix of the plastics when exposed to environmental conditions (Commission of the European Committees, 2000; Robinson, 2009).

Robinson (2009) discussed the mechanism of the migration of poly brominated diphenyl ethers (PBDEs) into the environment. The researcher noted that the absence of chemical bonds between the flame retardants and the plastic matrix increases the likelihood of the migration of the PBDEs into the environment. While bioaccumulation coefficients have been reported to be marginal, the effect in plant life was noted to still play a contributory role (Liu *et al.,* 2008; Robinson, 2009).

The tertiary contaminants emanating from e-waste processing constitute by-products of recycling operations that employ any of the technologies that include pyrometallurgy, hydrometallurgy, bio-hydrometallurgy or mechanical processing. Large scale pyrometallurgical operations are associated with gaseous emissions.

Flue gas scrubbers and other containment measures for handling the emissions have been used successfully in these installations like Umicore in Belgium (Hagelüken, 2006).

The drive towards sustainable practices along the value and supply chains of the electrical and electronic goods industry continues to be spear headed by forums that include the United Nations Development Programme (UNDP), Strategic Approach to International Chemicals Management (SAICM), the United Nations Environmental Programme (UNEP) and the International Conference on Chemicals Management (ICCM).

Key issues brought to the fore include agendas that will enable the realisation of a reduction in toxic substances during the design and manufacture of electrical and electronic goods, promote the detailed investigation and hence understanding of polybrominated flame retardants and promote global solutions for e-waste management problems. The solutions are also meant to cater also cater for both developing and transitioning economies. In these forums it has been agreed that concerted efforts are required to build capacity in these regions to better handle e-waste (SAICM, 2009).

2.2.5.   Motivation for WPCB recycling

An analysis of the typical compositions of varied sources of electronic waste shows that scrap cell phones, printed circuit boards and calculators report the highest metal value with respect to gold and other precious metal content (Anon, 2001; Cui & Zhang, 2008). The gold value locked in waste printed circuit boards has been reported to range between 80 and 1 500 grams/tonne (Veldhurizen & Sippel, 1994; Amey, 2002). Base and precious metals are found in higher grades than traditional ore bodies (Zhang & Forssberg, 1998; Cui & Zhang, 2008; Hagelüken 2006). The recent increases in base and precious metal prices has provided sufficient stimulus for increased value addition along the WPCB recycling chain (Wang & Gausted 2012; GreenCape, 2018). Table 2.5 provides a summary of the annual production volumes and demand of metals commonly used in the production of electrical and electronic equipment (EEE).

Table 2.5: Metals commonly used in the production of electrical and electronic equipment based on 2006 demand (Schluep *et al.,* 2009)

Metal	Primary production Tonnes/annum	By product from	Demand for EEE Tonnes/annum	Production or demand %	Applications
Silver	20000	Lead, Zinc	6000	30	Contacts, switches and solders
Gold	2500	Copper	300	12	Bonding wire, contacts, integrated circuits
Palladium	230	PGM	33	14	Multilayer capacitors and connectors
Platinum	210	PGM	13	6	Hard disk, thermocouple, fuel cell
Cobalt	58000	Nickel, Copper	11000	19	Rechargeable batteries
Bismuth	5600	Lead, Tungsten and Zinc	900	16	Solders, capacitor, heat sink
Selenium	1400	Copper	240	17	Electro-optic, solar cell, copier
Indium	480	Zinc and lead	380	79	LCD glass, solder, semiconductor
Ruthenium	32		27	84	Hard disk, plasma displays
Copper	15000000		4500000	30	Rechargeable batteries
Tin	275000		90000	33	Solders
Antimony	130000		65000	50	Flame retardant, CRT glass

The energy expenditure associated with recycling WPCBs is comparatively lower than processes associated with metal recovery from their native ores (Akcil *et al.,* 2015). A comparison of the material energy savings realised when metals were recovered from WPCBs was undertaken by researchers (Cui & Forrsberg, 2003; Wang & Gausted, 2012). The results of the studies conducted are summarized in Table 2.6. The results revealed significant energy savings associated with recycling of base metals especially copper and gold.

Table 2.6: Energy savings realized when metals are recovered from recycling of WPCBs.

Metals	Cui & Forrsberg (2003) % Basis – not specified	Wang & Gausted (2012) % Basis – per tonne WPCBs processed
Aluminium	95	5
Copper	85	1
Lead	65	
Silver		65
Gold		1
Zinc	60	
Iron	74	
Platinum		4
Tin		6
Palladium		14
Nickel		1

A reduction in the carbon footprint associated with mining and extraction of gold, cobalt and palladium is one of the attendant benefits of WPCB recycling. The production of a tonne of gold, palladium or platinum from native ores is associated with the generation of 17 000 tonnes of carbon dioxide. It is estimated that 23.4 million tonnes per annum of carbon dioxide are generated as a result of the production of copper, cobalt, tin, tantalum, silver, indium, ruthenium, platinum and palladium for use in the electrical and electronics industry. Gold and copper have been found to contribute the highest carbon dioxide emissions based on an analysis of the demand of metals in the EEE industry (Schluep *et al.*, 2009).

2.2.6.   Perceived threats to the viability of WPCB recycling

The evaluation of the overall recycling efficiency is a multiple function of the major activities across the recycling chain. The synergy between the government, recyclers and recycling enterprises has been identified as the means to enhance recycling efficiency of e-waste (Li *et al.,* 2016). The mobilization of existent participants into a more formalised structure is paramount to the success of the synergy.

Inadequate financing structures to support the reverse logistics of e-waste collection erode the viability of WPCB recycling. Decreasing intrinsic value of WPCBs as a result of the declining precious metals content further reduces the viability of WPCB recycling operations (SAICM, 2009). The development of alloys bearing similar performance characteristics to the precious metals have led to the aforementioned decline.

Product miniaturization has also contributed to an overall reduction in metals composition which has negative implications on the economic viability of WPCB recycling operations. Increasing the volumes of material processed to recover the same output of target metals yields a net increase in the volumes of effluent and emissions handled.

The viability of WPCB recycling with the objective of maximising metals recovery is also threatened by the existence of conflicting material recovery priorities. The result is a sacrificial loss of rare earth elements to landfills as residual material from copper, nickel, gold, silver, and PGMs recovery. The recycling of gallium in LEDs and ICs for example was shown to conflict with gold and copper recovery after a material flow analysis of gallium from EEE was undertaken (Ueberschaar, 2017).

The unregulated activities of the informal e-waste recycling sector, as seen in cases studies of India, China, Pakistan, Ghana and Nigeria for example, are typically associated with low metals recovery efficiencies and high rates of environmental pollution (Sinha 2004; Schluep *et al.*, 2009; Grant *et al.*, 2013; Shamim *et al.*, 2015; Hector, 2017).

The sustained existence and growth of the informal e-waste recycling sector is attributed to the general unwillingness of the formal sector or potential participants in the formal sector to commit financial resources towards proper e-waste disposal. Conflicting interests emanate from the potential that exists for making money in a less capitally and operationally intensive manner.

Owners of obsolete equipment would rather exploit the opportunity to resell the EEE rather than spend money to dispose of the end of life electrical and electronic equipment (EOL EEE) through the prescribed channels.

The failure to grasp the long and short term pathological implications and environmental impacts of informal e-waste recycling activities and the involved stakeholders weakens the efficacy of measures instituted by governments to merge and formalise the e-waste recycling sector (Hicks *et al.,* 2005).

The existence of weak institutional structures unable to effectively merge informal recycling activities, results in cumulatively low resource recovery rates ranging between 28% and 30%. Poor accountability and lack of homogeneous practices complicates data collection process relevant to all activity associated with the e-waste recycling chain. In systems where policy is enforced, weak institutional structures have been also be shown to exhibit a retrogressive impacts to the effective realisation of value in the long run (SAICM, 2009; Cuchiella *et al.,* 2016).

2.3. Recycling of metals from waste printed circuit boards (WPCBs)

2.3.1. Recycling of WPCBs

The recycling of WPCBs has been under study for over three decades. Much of the research work has exploited the varied properties of the waste printed circuit boards to extract metals of significant economic relevance. The technologies include hydrometallurgical, pyrometallurgical, mechanical and bio-hydrometallurgical processes. Combinations of the processing options have been studied to examine the best configurations for maximising the extraction of target metals in an economically and environmentally efficient manner.

Dismantling e-waste and recovery of WPCBs

Further to developing mechanical separation as the method of choice, is the question of adequately addressing dismantling of the components. To date the automation of the dismantling process still remains a challenge. Deep dismantling activities are characteristically associated with a high manual input (Schluep *et al.,* 2009). The manual sorting and dismantling of EOL electronic waste still offers a more precise method for effectively separating reusable parts from those ready for discarding or further processing in the recycling chain (Cui & Forssberg, 2003).

The extent to which deep dismantling is achieved determines the efficiency of accessing WPCBs in a form in which the contained metal value is preserved. Mechanical intervention has been shown to result in precious metal losses as a result of co-separation.

2.3.2.   Mechanical recycling of WPCBs

Mechanical treatment of secondary metal sources in recent years has emerged as an alternative and less chemically aggressive option for recovering metals from polymetallic secondary sources. This method exploits the various physical properties associated with the constituent materials in the scrap to achieve materials separation. A consideration of the heterogeneous composition of WPCBs as noted by Cui and Forrsberg (2003), necessitates a thorough characterisation of the material, key to the development of an efficient working process flow sheet for metals recovery. Cui & Forssberg (2003) noted that research to date on mechanical recycling of WPCBs and other electronic waste has shown an inclination towards recycling of electronic waste composed of a low precious metals grade. A plethora of physical separation processes are employed in combination owing to the complex material composition of EOL EEE like WPCBs. The work done to date exploiting physical separation methods has focused predominantly on the major elements like copper, iron and aluminium (Lee *et al.,* 2004; Birloaga *et al.,* 2016). Focus rapidly shifted in favour of fully exploiting and maximising value realisation from all constituent materials of WPCBs.

Detailed investigations have been undertaken to develop eco-efficient flow sheets to separate the various fractions of WEEE (Kiachevko *et al.,* 1983; Lee *et al.,* 2004; Li *et al.,* 2007). Li *et al.,* (2007) described a process for achieving a metal rich fraction using a two step crushing process that would liberate the metals from the basal plates. Alternative investigations have shown that separation of different metals occur in varying degrees in differing size fractions (Kogan, 2006; Zhao *et al.,* 2006).

The extent to which specific metals are liberated presents the challenge of co-separation. This has motivated the incorporation of stage wise separation and segregation of metals to be considered during the design of flow sheets.

In alternative research the use of water to circumvent the challenges of dry processing of WPCBs was considered. Xiang *et al.*, (2007) proposed a water based technology to capture and handle fines in the form of a paste. Froth flotation was also identified as a suitable method for the beneficiation of WPCB fines (Ogunniyi & Vermaak, 2009). The output of the investigations revealed that flotation models and the associated kinetics cannot be treated as simply as the naturally occurring ore bodies.

The economic success of metal recovery in the recycling of electronic waste is dependent upon effectively curtailing in-process losses of metals (van Heukelem, 2004; Huisman, 2004; Hagelüken, 2006). Between 10% and 35% of the precious metals in WPCBs reports to the fines fraction during the comminution stage (Hagelüken, 2006; Yazici, 2015).

In an analysis of the non-metallic fraction of comminution fines it was established that a copper assay of 2.1% was reported in the sample which was reported to be superior to several copper porphyry commercial ores (Yokoyama & Iji, 1997; Ogunniyi & Vermaak, 2009).

Other workers have reported that the gold composition of comminution fines of milled electronic scrap can be as high as 51% (Goosey & Kellner, 2002; Kogan, 2006). It was estimated that up to 10% of this value of gold reported in the fines is lost during physical processing.

The investigations of Cui (2005) also revealed that the economic attractiveness associated with manual dismantling and mechanical processing of high value WPCBs may be severely limited because of the precious metals co-separation.

Hanafi *et al.,* (2012) characterised milled WPCB fractions from three different sources, primarily, cell phones, computers and memory cards. The samples were treated with sulphuric acid and the crystals that were produced were subjected to a detailed analysis which revealed a notable degree of co-separation.

While the processes focus on the recovery of these dominant metals, researchers cautioned that the risk of losing precious metals cannot be overlooked as these losses contribute to a significant reduction of precious metals yield of the associated with the pre-processing stage (Hagelüken, 2006; Ghosh *et al.,* 2015; Khanna *et al.,* 2014).

In view of the research findings it is crucial to realise that any process designed for recovery of target metals of the highest purity would entail several processing steps. This however would be at the expense of higher material recoveries of the target metals (Hagelüken, 2006).

Mechanical recycling of WPCBs has been shown to be a useful technology for WPCB recycling. On its own however, the precise segregation of the different commercial groups of metals is surpassed by hydrometallurgical processes. Used in combination with other recycling technologies it is useful for achieving the necessary size reduction and liberation of metals prior to further processing (Rossouw, 2015).

2.3.3.   Hydrometallurgical recycling of WPCBs

Hydrometallurgy lends itself as a technology well known for its selectivity, scalability, rapid kinetics and precision during the extraction of metals from native ores and urban polymetallic sources (Rossouw 2015; Birloaga *et al.,* 2016; Albertyn, 2017).

The comparatively rapid kinetics of extraction makes it a commercially attractive option to consider when extracting metals from polymetallic sources such as EOL WPCBs. It has been argued that hydrometallurgical leaching plants and associated processes are not advisable for the recovery of metals from WPCBs due to the complexity of handling the toxic waste effluent and solid residue (Schluep *et al.*, 2009).

In light of this the extent of the research has merged the advantages associated with hydrometallurgy together with other extractive processes to develop process flow sheets that broaden the scope of metals recovered from WPCBs. Several configurations for leaching operations combined with recovery technologies have been and continue to be proposed (Kamberovič *et al.*, 2011; Birloaga *et al.,* 2016).

Hydrometallurgy remains the technology of choice for bridging the operational gap associated with maximising returns for low volume feed while drawing on the comparatively higher kinetics (Kamberovič *et al.*, 2011; Birloaga *et al.,* 2016; Cuchiella *et al*, 2016). The hydrometallurgical extraction of base and precious metals has been studied using a plethora of lixiviants. Both aggressive mineral acids and strong bases and more environmentally friendly lixiviants have been investigated (Pyper & Hendrix, 1981; Monhemius, 1987; Yannapoulos, 1991; La-Brooy *et al.,* 1994; Wu *et al.*, 2008; Jing-Ying *et al.,* 2012; Gurung *et al.,* 2013).

The extraction of metals using thiourea was investigated as an alternative to cyanide (Pyper & Hendrix, 1981; Monhemius, 1987; Yannapoulos, 1991; La-Brooy *et al.,* 1994; Wu *et al.*, 2008; Jing-Ying *et al.,* 2012; Gurung *et al.,* 2013). Findings of the studies revealed that thiourea is prone to oxidative degradation at elevated temperatures, high pH and is sensitive to elevated ferric ion concentrations. The commercial application of thiourea is however limited (Gurung *et al.,* 2013). The high material usage ratios compounded by higher material costs make it economically unattractive (Wu *et al.*, 2008).

The basic leaching of base metals using NH_3/NH_4CO_3 was also studied. The investigations revealed that it is possible to recover a pregnant leach solution with a high copper content in comparison to the other base metals like tin. It was also shown that the preferential extraction of tin from WPCBs can be achieved prior to copper removal using ammonia and ammonium carbonate (Liu *et al.,* 2009; Yang *et al.,* 2012; Xiao *et al.,* 2013; Somansundaram *et al.,* 2014).

Sulphuric acid leaching was also considered by researchers that attempted to extract both base and precious metals in varying conditions. The economics and technical efficiencies of the proposed processes were compared to leaching campaigns in which alternative lixiviants were employed (Behnamfard *et al.,* 2013; Birloaga *et al.,* 2013; 2014; 2016; Rossouw, 2015; Birloaga & Veglio, 2016).

Detailed investigations examining the selective extraction and recovery of precious metals were undertaken using aqua regia and halides (Ramadorai, 1993; Korte & Coulston, 1998; McNulty, 2001; Hongguang & David, 2002; Kinoshita *et al.,* 2003; Kogan, 2006; Sheng & Estell, 2007; Syed, 2012; Sadeghi *et al.,* 2013; Somansundaram *et al.,* 2014; Yazici & Devici, 2015).

The commercial application of the proposed technologies is limited by the aggressive nature and costs of the lixiviants. Halides exhibit marked handling difficulties as a result of the toxic and corrosive nature (Yannapoulos, 1991; Korte & Coulston,1998; Ramadorai, 1993). Further, the recovery of gold chloride through carbon adsorption has been shown to be thwart with process losses of gold particles resulting from their adhesion to the surface of activated carbon particles.

The potential of thiosulphate as another alternative to cyanide has also been investigated. Researchers have explored the use of diverse operating conditions to maximize precious metals recoveries using thiosulphate (Marsden & House, 2006; Ha *et al.,* 2010; Zhang *et al.,* 2012; Petter *et al.,* 2014; Akcil, 2015; Albertyn, 2017). In systems containing copper the process efficiency with respect to thiosulphate usage deteriorated. This was due to the irreversible decomposition of the thiosulphate which was catalysed by copper in solution. In comparison to cyanide its commercial relevance is dwarfed.

Ionic liquids on the other hand have been found to possess attractive features that include negligible vapour pressures, excellent thermal stability and thermal conductivities. This has stimulated research that explores their applicability as lixiviants (Huang *et al.,* 2014). The researchers were able to optimise process conditions. The commercial application of this technology has however not yet been reported.

Thiocyanate is associated with high precious metals recoveries, especially gold. Research findings have demonstrated that thiocyanate displays greater stability when used under normal process conditions. Complexes were formed with gold I (Au (I)) and gold III (Au (III)) species (Barbosa-Filho & Monhemius, 1994; Kuzugundenli & Kantar, 1999; Senanayake, 2004). The commercial application of thiocyanate is yet to be established.

2.3.4.   WPCB recycling process flow sheets

Different researchers have proposed a varied number of flow sheets to extract the metals of interest and separate the constituents for either in-house or external processing. The flow sheets are outlined in Table 2.7.

Table 2.7: A summary of some of the different flow sheets developed for recycling WPCBs and other electronic waste

Process	Description	Researchers
Cu, Sn and Pb recovery	Mechanical pre-processing (crushing and grinding) followed by copper leaching with NH_3/NH_5CO_3. After distillation to recover NH_3 which is returned into the main circuit. The resultant $CuCO_3$ is calcined to yield CuO. Co-dissolution of other base metals noted with high zinc levels. HCl used to leach Pb and Zn.	Liu *et al.,* (2009)
Non ferrous metals and ferrous metals	Disassembly, hand sorting, shredding and then isolation of ferrous metals. Further size reduction followed by magnetic and then eddy current separation to isolate the metals from the non metals.	Tuncuk *et al.,* (2012)
Base and precious metals	Illustration of potential process options that embraces physical separation, size reduction and isolate ferrous and non ferrous metals. Mechanical separation can be by-passed to progress to hydrometallurgical or pyrometallurgical treatment.	Yazici & Devici (2015b)
Base and precious metals copper, gold and silver	Mechanical pre-treatment, selective two stage counter current leaching of base and precious metals.	Birloaga & Veglio (2016)
Ferrous metals, filter dust with entrained metal fines, aluminium and non ferrous metals, WPCBs and plastics	Mechanical process for pre-treating high grade electronic waste enabling isolation of WPCBs. The flow sheet makes use of manual sorting prior to size reduction that is followed by pneumatic, magnetic and eddy current sorting. At the final stage optical sorting is utilized to segregate plastics from WPCBs	Bigum *et al.,* (2012)
Recovery of high purity gold	Embraces a nine stage process for recovering gold from scrap. After dissolution with aqua regia iron is removed. The system is acidified with HCl. The gold rich liquor undergoes extraction, stripping, reduction and removal of impurities. Regeneration of HCl and HNO_3 is included in the loop.	(Syed, 2006; 2012)
Recovery of gold and base metals	Feed material is comprised of WPCBs, glass bangles and mirrors. The recycling process entails size reduction, washing and boiling in formic acid. After skimming further boiling is done with 20% aqueous potassium persulphate. Final washing with Borax and KNO_3 to recover gold.	(Syed, 2006; 2012)

Table 2.7: An summary of some of the different flow sheets developed for recycling WPCBs and other electronic waste con't.

Process	Description	Researchers
Recovery of precious metal Ag, Au and Pd	Hydrometallurgical process. Base metal leaching with sulphuric acid, chloride and cyanide leaching of precious metals. Cementation and activated carbon adsorption for precious metals recovery.	Quinet *et al.,* (2005)
Recovery of gold	Thermal degradation of waste followed by nitric acid leaching, solvent extraction and reduction.	Chmielewski *et al.,* (1997)
Recovery of base metals	Mechanical pre-processing followed by nitric acid leaching and electro-hydrometallurgy.	Mecucci & Scott (2002)
Recovery of base and precious metals	Mechanical pre-processing to segregate non metals and concentrate metal fraction prior to hydrometallurgical treatment to recover copper. Metallic concentration ineffective for gold and silver.	Kasper *et al.,* (2011)
Recovery of copper and precious metals. Fe, Al, Cu, Ni, Zn, Ag, Au,	Three process flow sheets were proposed that all entail mechanical pre-processing followed by selective base metal leaching and the final leaching stage is for the extraction of precious metals. The second process flow sheet provides for the removal of nickel and zinc at the last stage. The third process flow sheet relies on environmentally friendly leachants. An economic evaluation of the sulphuric acid and thiourea based process flow sheet was undertaken. The evaluation indicated that for a gold value in excess of 500ppm the project was feasible.	(Kamberović *et al.,* 2009; 2011)
Recovery of Cu, Au, Ag	Incorporates physical pre-treatment of WPCBs. Followed by selective base metal leaching. Selective leaching of precious metals with HNO_3. Leaching of gold followed by reduction that is refined to recover pure gold.	Zhang *et al.,* (2012)
Sn, Cu and Pb as PbO_2	Mechanical pre-treatment mainly size reduction by shredding. Leaching WPCBs with HNO_3 to remove Sn. Recovery of Sn through dissolution of metastannic acid. Resultant solution neutralized prior to electro-winning to recover copper. Nitric acid is regenerated.	Mecucci & Scott (2002)
Au, Ag, Pd and Cu	Milling of WPCBs to -300 microns. Directs feed through a four stage leaching step that commences with a two stage leaching of copper, followed by solvent extraction of copper and cementation with iron to recover the copper fines.	Behnamfard *et al.,* (2013)
Selective recovery of gold and silver	Mechanical treatment (size reduction) followed by calcination. Non selective leaching of WPCBs with acido thiourea. Selective recovery of gold and silver by adsorption with CPT gel.	Gurung *et al.,* (2013)
Recovery of ultrafine copper particles	Crushing of PCBs. Mechanical pre-treatment to enrich the metals fraction. Leaching of copper with NH_3/NH_4SO_4 followed by solvent extraction, stripping and reduction to recover ultrafine copper particles.	Yang *et al.,* (2012)

2.4.   Techno-economic feasibility studies of WEEE recycling

2.4.1.   The economics of recycling

Several methods are available for determining the techno economic feasibility of a recycling process. Ferreira *et al.,* (2012) described three methods that include the Input-Output method, Life Cycle Costing (LCC) and Cost Benefit Analysis (CBA). Of these methods LCC is most popularly used. Two major costs are associated with LCC of recycling operations and include internal costs and external costs. The internal costs include the conventional costs also referred to as direct costs and the indirect costs. The indirect costs are made up of the hidden costs and the less tangible costs. The external costs are incurred by virtue of operating within a business environment and can include costs such as carbon taxes, environmental taxes and licencing fees, emissions and effluent discharge fees. These costs are not derived from the operations of the business and are normally accounted for as indirect costs when computing the operating expenditures.

The economic success of e-waste recycling is dependent on the effective segregation and concentration of the constituent materials feed prior to reporting to further processing steps. The most significant contributors to the cost of e-waste recycling is associated with collection, storage and handling, dismantling and pre-treatment and segregation of hazardous components (Schluep *et al.*, 2009).

2.4.2.   Comparing reuse versus recycling

Miniaturization has played a part in the increased volumes of e-waste with lower relative metal concentrations. Attempts to increase the metals recovered at the already low concentrations would yield no significant benefit due to high logistics costs that overtake revenue generation rates. The combined effects of product miniaturization, low outflows of high grade raw material feedstock and unstructured e-waste collection rates in emerging and developing economies have culminated into the observed low margins associated with e-waste and WPCB recycling. The observed trend as a result, has been the combination of both e-waste recycling and refurbishment in both small and larger well established facilities. A bias for refurbishment activities in smaller establishments has been observed by researchers (Geyer & Blass 2010; Lydall *et al.,* 2017).

2.4.3.   E - waste recycling in Southern Africa

A case study looking at the economic feasibility of e-waste recycling in Tanzania embraced a business model that examined the viability of refurbishment activities combined with value addition using a manual system. The supply risk of obsolete electronic equipment was identified as a threat to the viability of the undertaking.

This observation agrees with findings made by other workers (Patil & Patil, 2015; Li *et al.*, 2016). Findings of the study revealed that a financing mechanism at the onset of the project to facilitate collection activities would be necessary to build up a healthy operating raw material inventory.

Exogenous factors were also identified that presented a threat to the viability of the business namely, reduced quality of raw material feedstock due to changing appliance composition, informal cannibalization of the bulk material and a drop in commodity prices. Rather than recycle the e-waste or attempt to separate materials, the resale of salvaged parts was found to be a more lucrative option. The study also revealed that the sale and processing of printed wiring boards was a significant revenue driver. It was proposed to merge the collection of WPCBs at a regional level to curtail the interruption of material flows in the business. It was noted that a combined throughput of 980 tonnes per year of WPCBs would be required to ensure the viability of the business. A strategy proposed to manage the supply chain entailed a centralised collection of WPCBs at a regional level. Implementation of the strategy is strongly dependent on regional cooperation.

The success of regional co-operation it was noted, is dependent on an active awareness campaign supported by a regional policy framework which will impact domestic e-waste policy frameworks in regional member states, combining efficient and supportive local and regional policy frameworks, value adding activities complementing core activities and reduce the total dependence on the intrinsic value of raw material, enhancing collection activities would contribute positively towards sustainable recycling business in Tanzania (Blaser & Schluep, 2012).

2.4.4. Evaluating the economics of operating configurations

2.4.4.1. Economics of field plants

Kang & Shoenung (2006) studied the economics of electronic waste recycling that focused on processing of monitors, CRTs and PCs. The economic evaluations did not take into account though, the overhead costs. In this study the researchers were able to show that almost 40% of the annual operating costs were attributable to pre-processing activities that involved collection, sorting, testing and dismantling. A further interrogation of the operating cost distribution revealed that labour and materials together contributed 60% of the overall operating costs.

A sensitivity analysis examining the influence of capacity utilization and the CRT tonnages treated revealed that the rate of change of revenue changed faster than the rate of change of treatment cost as observed by Ghodrat *et al.,* (2016). To improve the economics of a recycling facility, the researchers noted that the e-waste quality would need to be high. This would have to be accompanied by the automation of labour intensive processes coupled with high capacity utilization.

2.4.4.2.   Economics of mobile plants

A mobile recycling plant process consisting of two containers was developed with the objective of sustainably recycling e-waste while reducing the environmental burden of existing e-waste volumes (Zeng *et al.,* 2015). Economic studies of the process suggested that field installations associated with the process displayed greater profitability than the mobile plants.

An alternative hydrometallurgical process was developed for the recovery of base and precious metals that included copper, gold, silver, iron, nickel, lead and tin (Kamberović *et al.,* 2011). A techno-economic feasibility study was done using batch sizes of 50kg and 100kg. The impact of gold concentration in the WPCBs feedstock, operational time and the effect of the batch size on the profitability were studied. The process exhibited the greatest sensitivity to gold content in the feed and the batch size. When the batch size doubled the total operating costs almost increased in a similar fashion. A 49% slump in the total operating costs was realised at a comparative gold concentration of 500ppm at which a 50kg batch became profitable, the payback period slumped by 66%, indicating a high sensitivity to increased capacity utilization. Costing and evaluation was done only for the main process and did not include WPCB pre-treatment and the detailed effluent treatment programme.

Mobile plants have shown promise in alleviating the growing burden associated with the inaccessibility to efficient recycling technologies in more remote regions (Zeng *et al.,* 2015). The associated operational costs for field plants were demonstrated to be higher than the mobile plant. The operational costs are a function of raw material, utility and maintenance costs and the largest contributor is governed by the process selected and the raw material quality.

2.4.4.3.   Comparing field and mobile plants

Cuchiella *et al.,* (2016) examined the profitability of WPCB recycling and other electronic equipment. Using the projected estimates as discussed in UNEP reports of e-waste growth rates and the estimated contribution of the WPCBs to the total weight of a component the researchers were able to work out the profitability of field and mobile plants. The WPCB purchasing cost was shown to be sensitive to commodity and metal price fluctuations and the composition of the WPCBs. A comparative analysis of the economic performance of e-waste field and mobile recycling plants was undertaken by Zeng *et al.,* (2015).

The researchers observed that recycling of both low and high grade e-waste displayed marginal profitability. It was also shown that this set up would be suitable for economies with highly dispersed e-waste collection points, fluctuating returns and feed volumes that do not appeal to the economics of large scale field plants.

The disparity between the findings discussed by Cuchiella *et al.,* (2016) and Zeng *et al.*, (2015) demonstrates that the geographical location of the plants has a bearing on the economic performance of an operation. Field plants were shown to have longer lives and larger net present values (NPVs) in comparison to the mobile plants.

The comparative NPV per capita (total initial capital investment) between field and mobile plants for all European countries in the EU is variable. The NPV per capita (total initial capital investment) of the field plants however remained greater than that of the mobile plants (Zeng *et al.,* 2015; Cuchiella *et al.,* 2016).

2.5.   Chapter summary

The literature study was opened by a discussion of the definition, classification and profiling of printed circuit boards. A disparity exists between the valuation and of the types of waste printed circuit boards. The composition of the waste printed circuit boards varies according to the end use of the commodity for which it is manufactured. Three major economic classes have thus defined to classify the grades of WPCBs based on the content of gold contained.

It was demonstrated in the study that the recycling of WPCBs offers several benefits that include a reduction in the carbon footprint associated with precious metals recoveries and potential for energy conservation. Several studies have been done to investigate technically and economically sound methods to recover base and precious metals from WPCBs. The methods include pyrometallurgical, hydrometallurgical, mechanical and bio-hydrometallurgical processes. The recovery of the metals has been shown in a number of research papers to be dependent upon the degree of liberation of the metals from the non-metallic matrix of the WPCB.

Hydrometallurgy remains the technology of choice for bridging the operational gap associated with maximising returns for low volume feed while drawing on the comparatively higher kinetics. The technical feasibility of the varied hydrometallurgy processes was demonstrated. The commercial success of the proposed flows sheets was shown to be limited mainly by the restrictive operating costs and in some cases the aggressive nature of the reaction media selected. A comparison of the economic evaluations of different WPCB recycling campaigns was also examined. It was seen that the compositional variation of the WPCBs had a strong influence on the process developed and the overall project profitability.

The performance of mobile and field installations were also compared. Contrasting observations made by different researchers, revealed the importance of factoring the effect of geographical location on the performance of a project. It was generally observed that the recycling of WPCBs is a low margins business that deteriorates with the increasing degree of process complexity and decreasing precious metals value in the WPCB feedstock.

The high costs of reverse logistics associated with WPCB recycling also contributed to the observed low margins. Based on findings of previous research however, the financial attractiveness of WPCB recycling is threatened by the observed decrease in precious metals content in the raw material feedstock. Compounded by low outflows of high grade raw material feedstock and unstructured e-waste collection rates in emerging and developing economies, recovery rate of metals will remain unsatisfactory. The economic studies conducted on the proposed installations are not holistic. Most investigations focused on the main process. While some researchers incorporated the contribution of overheads in their studies, other aspects of the economic evaluations such as detailed effluent management and treatment costs were not addressed in great detail. The findings are relevant in providing other researchers and concerned stakeholders a platform on which to plan and focus attention in order to make informed decisions.

Chapter 3:   E-WASTE MANAGEMENT IN SOUTH AFRICA

3.1.   Classifying waste electrical and electronic equipment

WEEE is comprised of ferrous and non-ferrous metals, plastics, ceramics, glass, rubber, wood and plywood in varying compositions specific to the design and the intended use of the component. Five differing definitions of e-waste have been advanced by researchers (OECD, 2001; Sinha, 2004; StEP, 2005; Widmer *et al.*, (2005); GreenCape, 2018).

E-waste can be categorized into two groups: WEEE normally considered e-waste and WEEE not normally considered e-waste. A comprehensive list of the associated electronic and electrical devices falling into the respective categories is detailed in literature (Robinson, 2009).

The European Parliament and the council of the European union distinguished ten different categories of e- waste and listed them as follows; large household appliances, small household appliances, IT and telecommunications equipment, consumer equipment, light equipment, electrical and electronic goods, toys, leisure and sports equipment, medical devices, monitoring and control instruments and finally automatic dispensers. These ten categories are outlined in Table 3.1. In an alternative study a more condensed categorization was discussed (Baldĕ *et al.,* 2015). Six categories of e-waste were defined that include temperature exchange equipment, screens and monitors, lamps, large equipment, small equipment and small IT and telecommunication equipment.

Table 3.1: Categorization of e-waste sources (Commission of the European Communities, 2000)

Category	Details	Examples (the list is not exhaustive)
1	Large household appliances	Refrigerators, washing machines, microwaves, hot plates.
2	Small household appliances	Vacuum cleaners, bathroom scales, clocks, weaving and processing equipment for textiles.
3	IT and Telecommunications	Centralised data processing: mainframes, minicomputers, printer units, personal computing: personal computers (cpu, mouse, screen and keyboard included), laptop.
4	Consumer equipment	Radios, televisions, amplifiers, video recorders.
5	Lighting equipment	Low pressure lamps, sodium lamps, metal halide lamps.
6	Electrical and electronic tools	Sewing machines, tools for riveting, nailing or screwing, welding.
7	Toys, leisure and sports equipment	Sports equipment with electric and electronic components, video games and consoles.
8	Medical devices with the exclusion of contaminated items	Freezers, radio therapy equipment, pulmonary ventilators
9	Monitoring and control instruments	Smoke detectors, thermostats, weighing or adjusting appliances
10	Automatic dispensers	Food, beverages or cash dispensers

3.2.   E-waste regulations

3.2.1.   Regulation of e-waste

3.2.1.1.   The Basel convention

The inception of the Basel convention which was adopted in 1989 and came into effect in 1992 (Hector, 2017) was motivated by the need to curtail the unrestricted dumping and transboundary movement of hazardous wastes from producer nations to potential consumer nations and dumping grounds (Needhidasan *et al.,* 2004). The convention does not seek to violate the sovereignty of States over their territorial sea established in accordance with international law, the sovereign rights and the jurisdiction which States have in their exclusive economic zones and their continental shelves in accordance with international law.

The Basel convention mandates generators of hazardous waste to execute their duties regarding the transportation of hazardous waste and waste in an environmentally sustainable manner. Liability is assumed in accordance with international law. Member nations are required to take into account the limited capabilities of developing countries to manage hazardous wastes and other wastes.

Independent research findings have highlighted the failures of the Basel Convention and its subsequent revision the Basel Ban, in curtailing the transboundary movement of e-waste from the developed nations to the developing economies (Hector, 2017). The observed failures were attributed to the existent gaps in the system such as the inconsistencies in defining e-waste and specifying what a hazardous material is across differing geographical boundaries. The unregulated movement of large shipments of e-waste, which in many cases are beyond refurbishment, into the country potentially poses a risk of burdening the country with volumes of material which after the various forms recycling result in the generation of large quantities of by-products that have a lower market absorption rate. Such by-products include non-recyclable plastic fractions.

3.2.1.2.   The Swiss e-waste management approach

The principle of extended producer responsibility (EPR) is the basis upon which WEEE management in Europe is founded. In other developed and developing economies EPR has percolated into e-waste management systems in varying degrees while it remains more voluntary in Canada and Australia (Schluep, 2014).The concept of EPR is engrained within the Swiss e-waste management approach. Four stakeholder levels, used in the Swiss e-waste management approach have been specified and the inclusive obligations of these levels in the e-waste management and recycling chain have been discussed in literature (Schluep, 2014). The stakeholder approach used in the Swiss e-waste management system is outlined in Figure 3.1.

This approach is more condensed when compared to the stakeholder analysis described in literature (Finlay & Liechti, 2008) in an assessment of the state of e-waste management practices in South Africa.

DEVELOPING AN E-WASTE STRATEGY

Country assessment
Framework conditions

Develop structured strategy
Multi stakeholder approach
Policy and legislation
Business and finance
Technology and skills
Monitoring and control
Marketing and awareness

Implement strategy
Develop road map embracing
population aspects in structured
strategy

Immediate   **Mid term**   **Long term**

Figure 3.1: Framework for developing e-waste strategy (Schluep, 2014)

3.2.1.3.   Extended Producer Responsibility

Extended producer responsibility (EPR) describes a comprehensive obligation and approach that businesses and organizations have and must adopt to reduce the environmental impact of their products and packaging.

The adoption of EPR is at an advanced stage in the developed nations and is either non-existent or in the early stages of its adoption in developing and transitioning economies. Varied schools of thought regarding the approaches, extent of implementation and effectiveness of EPR in the developing and transitioning economies have been discussed in detail by different researchers (Nnorom & Osibanjo, 2008; Schluep, 2014; Baldĕ *et al.,* 2015).

Manufacturers embed the concept of enhanced recyclability within the design of their products and cost of recycling activities that are passed onto consumers within their product costing structures. The effectiveness of EPR must not be limited by geographical boundaries but must follow products thereby contributing to addressing the negative impact of transboundary movement of e-waste.

Three key objectives for EPR principle have been defined and include the following: the objectives must holistically embrace the manufacture of goods and materials, secondly manufacturers are obligated to prioritize environmentally sound designs with recyclability in mind and finally the products should possess reasonable utilization efficiency while the recoverability of the materials must not be neglected.

The concept of product self management must be included as part of the product specification. Information pertaining to the design attributes of the components or device as a whole is included. This is meant to inform recyclers on the technical aspects of the product affecting its recyclability, safe handling and management of any other aspect associated with it.

It has also been envisaged that the adoption of EPR as an e-waste management approach aims to achieve a closed loop in which the downstream accumulation of pollutants at the end of a product's life cycle without any solution for reuse or reintroduction into the main production chain is eliminated. The take back system is the crucial fulcrum upon which the EPR system functions.

Four schools of thought have been advanced regarding the adoption of EPR that include the product take back programs, regulatory approaches, voluntary industry practices and economic instruments (OECD, 2001).

3.2.1.4.   E- waste regulation in South Africa

In 2004 an initiative was launched by the Federal Laboratory for Materials Testing and Research (EMPA) to assess the e-waste streams and final pathways in South Africa. It was proposed that the major players would include the National Recycling Forum (NRF), Institute of Waste Management of Southern Africa (IWMSA), Non-Ferrous Metal Industry Association, the Metal Recyclers Association, Foundry Manufacturing Association and the South African Iron and Steel Institution.

Metal Recyclers Association of South Africa (MRA) regulates the metal recycling activities in South Africa. The MRA is made up of members involved in the collection and processing of over 80% of scrap metal in the country. An executive committee convenes almost on a monthly basis to address matters that affect the scrap metal recycling industry in South Africa.

The association also oversees the smooth interaction between the recyclers and consumers. It helps to ensure that local industry requirements are attended to so that members positively contribute to the metal recycling industry in a manner that does not compromise the principles of sustainable practices pertaining to the metals recycling industry. Finally, the association polices the adherence of the members to the set code of conduct of the association (MRA, 2017).

South Africa, India and China joined the global e-waste knowledge sharing program spearheaded by the Swiss State Secretariat for economic affairs (SECO). Implementation has been the responsibility of EMPA (Finlay & Liechti, 2008). South Africa was a signatory to the Basel Convention. A Basel Convention Regional Centre representing Africa was set up, one of four, to institute training programs with the objective of empowering participants with technical and practical expertise in effective hazardous waste management.

A synchronised collection and waste management approach within a nation is essential in promoting an efficient e-waste economy within a country that also has globalized approach to e-waste management with interdependent structures (Schluep *et al.*, 2009; Schluep, 2014). The absence of a clear policy and legislation addressing collection, dismantling, pre-processing and end processing were highlighted and have been identified as the most significant barriers to e-waste management systems in developing and transitioning economies. While the South African formal system has attempted to develop structure in this arena several loop holes and system inefficiencies still exist as will be discussed in sections to follow.

A model based on the Swiss e-waste management approach was developed and tested. A review of the model's performance however exposed the shortcomings that were associated with a poor appreciation of the differences in the operating environments, namely the European context versus the South African context (Lawhon, 2013). The existent discourses on e-waste management and recycling in South Africa were also examined (Lawhon, 2013). It was noted that key stakeholders were at a point of conflict regarding the approach to adopt in managing e-waste in South Africa. Lawhon (2013) noted that one side of the argument was in favour of fully adopting the Swiss recommended model. The counter argument favoured the provision of a uniquely tailored solution for South Africa described in alternative reports (Schluep *et al.*, 2009; Lawhon, 2013; Schluep, 2014).

The e-waste Association of South Africa was formed in 2008 as a body comprised of key stakeholders in the e-waste sector. Like recommendations made by Schluep *et al.*, (2009) a new forum for manufacturers in South Africa, ITA-PEG was formed.

This was a brain child of English Africa Hewlet Packard in which proposals for collection, transport and storage schemes of e-waste were drafted. The architects of the proposed scheme were confident that while the recycling rate has not matched the performance of counterparts in more developed economies the initiative would provide sufficient stimulus to draw greater participation from other interest groups. It was believed that the proposed scheme would boost the efficiency of collection and positively impact recycling of e-waste in South Africa.

The concerted effort led to broader alliances. E-WASA in collaboration with eWA (e Waste Alliance), RAG (Recovery Action Group), WCRAG (Western Cape Recovery Action Group), GIZ (Deutsch Gesellschaft für Internationale Zusammenarbeit), MTN (Mobile Telephone Network), ITA-PEG, Nokia's Ulrike Volt, DELL, IWMSA (Institute of Waste Management Southern Africa), documented guidelines for emerging entrepreneurs venturing into the business of e-waste recycling. The initiative was undertaken to promote controlled and well managed e-waste recycling activities without endangering human health and the environment.

The publication allows new participants a platform to understand what e-waste is, legislation influencing the activities they intend to embark on and offers guidelines for proper accounting mechanisms in managing an e-waste business (Pocket information guide for Southern African e-waste Collectors, 2013).

3.2.2.   Legislation

The South African e-waste management systems and status were assessed by different researchers. Earlier recycling activities of e-waste in South Africa have been independent of significant government intervention (Widmer & Lombard, 2005). More recently the e-waste management and recycling activities have been increasingly influenced by existing legislation pertaining to environmental and waste management regulations in the country.

The e-waste management and legal landscape

Godfrey & Oelofse (2017) identified the four stages characterising the development of the legislative background influencing e-waste recycling in South Africa. This has spanned a 30 year time frame since 1989 and is outlined in appendix H, Table H.1. which shows that there is an active and concerted effort within the South African waste recycling landscape to move towards a more circular economy by varied stakeholders that also include formal and informal recyclers and beneficiaries along the e-waste recycling chain. The shift has been progressing with varying degrees of adoption without a clear policy laid out for e-waste management and recycling.

The challenges, proposed actions and responses to the measures were also discussed by the researchers (Godfrey & Oelofse, 2017). The strides in the developed nations have played a significant role in influencing the direction of e-waste management legislation in South Africa. This has been evidenced by the active initiative to adopt and implement EPR in varying degrees as the South Africa shifts to a more circular economy (Dittke, 2007; Godfrey & Oelofse, 2017).

A summary of relevant legislation affecting e-waste management and recycling in South Africa is detailed in appendix H, Table H.2 provides a more comprehensive account of legislation influencing e-waste recycling activities was however discussed by Dittke (2007).

From the overview it is clear that a more distinct legal framework unique to e-waste management is required. The adoption of the legislation described in Table H.2 varies at provincial and municipal levels as will be described in the following discussion.

Weaknesses of the current legislation

Several research findings revealed the absence of a unique legislation in South Africa for e-waste management and associated recycling activities (Finlay, 2005; Dittke, 2007; Lawhon, 2013; Mouton & Wichers, 2016; Godfrey & Oelofse, 2017). It was also noted that the enforcement of legislation in the different provinces was and is not homogeneous. Investigations also revealed that the monitoring of enforcement differs with the level of priority accorded to the subject matter (Dittke, 2007). No mention of e-waste has been made in any of the laws that can be applied to the management and handling of e-waste. Dittke (2007) exposed the vagueness of some of the provincial legislation in terms of scope and applicability. The extent of the effectiveness in attaining national e-waste recycling targets becomes varied and in the worst cases is illusive.

The fourth observation revealed that the existent laws in place do not adequately address the difficulties associated with handling complex e-waste. This has manifested by the current challenge associated with the provision of a suitable and simple disposal plan for e-waste like CRTs and nickel-cadmium batteries.

Based on the research findings it can be concluded that the crafted legislation must be holistic and uniform in its enforcement and monitoring. The legal instruments must be capable of addressing areas that include handling and storage of e-waste, licensing, supporting architecture, emissions and effluent management, water management, dissemination of information of public concern, logistics and supply chain management, occupational health and safety related matters, emergency response plans and procedures in the e-waste recycling chain and the monitoring and evaluation of e-waste related activities.

Legislation relating to buildings and associated infrastructure unique to e-waste handling must also be defined to provide for the appropriate storage of hazardous material. Training, certifications and associated accreditations should be clarified with provision for assessment and continuous development around e-waste management. By embracing these aspects of e-waste management, it is essential that uniformity in adoption of the crafted legislation is achieved at the point of execution at government levels.

3.2.3. Policy implementation

Proceedings in Geneva 2009 revealed that a solid policy regarding e-waste management in African nations must be established and consolidated. A request was made for the more experienced nations to assist African nations in developing architecture for managing e-waste, developing the legislation and curbing illegal transboundary movement of e-waste. The adoption of EPR as an approach to managing e-waste volumes in South Africa for instance, is in its infantile state.

A number of constraints have been identified that hinder the progress of the EPR in developing economies. These constraints include the resolution of conflicting priorities associated with manufacturers and recyclers, the achievement of a balanced alignment of the EPR with the existent business models and strategic objectives of stakeholders, distinguishing between what must be defined an industrial standard and what is law and finally defining the specific responsibilities and limits of authority.

It has been proposed that a unique model can be tailored for the developing and transitioning economies (Schluep *et al.,* 2009; Schluep, 2014). The model is developed by merging the success elements of the European recycling and e-waste management system together with the high collection rates associated with the informal sector and resources of the respective country.

Schluep (2014) noted that the institution of an efficient e-waste management system to curtail monopolistic practices without necessarily depending on the enforcement of rigid and punitive policy measures would suffice to suppress these undesirable practices. A case in point illustrating the objectionable ramifications resulting from the failure to circumvent loop holes in any instituted policy framework, is the continued transboundary movement of e-waste to West Africa, Eastern Europe, parts of Asia like India, Pakistan and China despite the institution of global protocols like the Basel Convention (Widmer *et al.,* 2005; Nnorom & Osibanjo, 2009; Chatterjee & Kumar, 2009; SAICM, 2009; Schluep *et al.*, 2009; Chatterjee, 2012; Hector, 2017).

Although South Africa is a signatory to the Basel Convention, there have been reports of illegal shipments of e-waste from the US into the country, current legislation and involvement of the government was ineffective in aborting further movement of a reported load into South Africa from the Durban port (Lawhon, 2013). This incident has brought to light the weakness in the Basel Convention in curbing illegal transboundary movement of hazardous materials (Nnorom & Osibanjo, 2009; Chatterjee & Kumar, 2009; Chatterjee, 2012; Hector, 2017).

The co-operation of stakeholders must be strengthened to promote the efficacy of policy implementation at all levels (Chatterjee & Kumar, 2009; Chatterjee, 2012; Lawhon, 2013; Dittke, 2007; Finlay & Liechti, 2008; Schluep, 2014; Mouton & Wichers, 2016). Recommendations have thus been advanced to remedy the observed deficiencies in the system.

The proposed solutions include: the establishment of an institutional framework to allocate responsibilities, schedules and objectives within the e-waste sector, the creation of awareness at all governance levels and the public at large, the identification, quantification and evaluation of existing support markets, the identification of collection and data management systems, the development of legal frameworks achieved through a review of existing legislations followed through by the development of a qualified and efficient e-waste recycling sector (Finlay, 2005; Dittke 2007; Lawhon 2013; Mouton & Wichers, 2016; Godfrey & Oloefse, 2017; Lydall *et al.,* 2017; GreenCape, 2018).

Widmer *et al.,* (2005) attempted to put these ideas into context by developing an assessment indicator system with the main objective of developing a system to measure and compare WEEE management systems. The model is depicted in Figure 3.2.

Assessment Indicator System to measure and compare WEEE management systems

E-waste Management System

Structural framework

Recycling system

Impacts

Politics and legislation
Ratification of Basel
convention and BAN
ammendment
Status of national
waste legislation
Status of national
e-waste legislation
Corruption perception

Economy
Capital costs
Secondary raw
material markets

Society and culture
Civil and political liberties
NGO activities
Recycling culture
Environmental awareness in
society

Science and technology
Knowledge in WEEE
technology
Supporting research

Material flow
WEEE generation
per capita
Closed loop recycling
management

Technologies
Efficiecy of
recovery
Quality of
recovered material

Financial flow
Financial coverage
Externalities coverage
Financial incentives for
ecodesign

Environment
Disposals
Emissions and
effluent

Human health
OHSAS
Impact on local
populations

Labor
Job creation
Income distribution

Figure 3.2: Assessment indicator system for comparison and measurement of WEEE management systems (Widmer *et al.,* 2005)

The assessment indicator shown in Figure 3.2 specifies the key pillars that enable users of the framework to holistically address the operational, administrative and measurable long and short term effects of the e-waste management system adopted. By addressing the three major pillars input of all stakeholders (direct and indirect) is taken into consideration and monitored for progress in its contribution towards the broad e-waste management objectives. Financing structures and performance measures are crystallized at all levels along the e-waste chain.

3.3.   E-waste volumes in South Africa

E-waste flows in the country are derived from the direct procurement of new EEE typically from international suppliers. Secondly the importation or receipt of second hand goods in the form of donations or second hand purchases and the receipt of obsolete equipment from within the continent (Lawhon, 2013). Between 3% and 20% of the WEEE entering the South African e-waste recycling economy is sourced from the SADC region. Domestically, e-waste is derived from the government, households and businesses (Lydall *et al.,* 2017).

3.3.1.   Accounting for e-waste flows

The quantification of e-waste generation rates remains a grey area, subject to the weaknesses of inadequate accounting and documentation of collection activities, disposal and recycling rates (Schluep *et al.*, 2009; Nnorom & Osibanjo, 2009; DEA, 2012; Lawhon, 2013; Shamim *et al.,* 2015; Lydall *et al.,* 2017; GreenCape, 2018).

Statistics describing the growth and accumulation rates of waste electronic equipment revealed that the South African e-waste management and recycling data base is limited (Ogunniyi & Vermaak, 2009). The existent data collected has been the output of inferences derived from market Figures from active recyclers and dealers of electronic equipment (Lydall *et al.*, 2017; GreenCape, 2018). Most estimates have been and continue to be derived from mathematical deductions determined from sales volumes, market forecasts and the projected life spans of equipment (Furter, 2004; Mackay, 2004).

Finlay & Liechti (2008) have provided a summary of the life spans of typical electrical and electronic products associated with the South African consumer market. The data is summarized in Table 3.2.

Table 3.2: Life spans of electrical and electronic equipment (Liechti & Finlay, 2008)

Equipment or gadget	Life (years)	Mass (kg)
Refrigerator	10	45
Washing machine	8	65
Microwaves	7	10
Televisions	10	20
Desktop PCs with monitors	4	27
Printers	5	8
Mobile phones	1.50	0.15
Laptops	4	3.50

E-waste flows on a provincial level were analysed by researchers. A projected e-waste growth rate of 4% was determined from the output of previous survey reports and mathematical extrapolations of baseline data (DEA, 2012; Lydall *et al.,* 2017). Gauteng and the Western Cape contribute towards the major portion of e-waste inflows in the country. Gauteng however dominates as the major e-waste processing zone in the country (Lydall *et al.,* 2017; GreenCape, 2018).

Currently the majority of recycling activities gravitates around economically active regions that include KwaZulu Natal, the Western Cape and Gauteng. The smaller operators concentrate their activities within the provinces housing their operations.

The distribution generally mirrored the degree of economic activity associated with each of the provinces. The remaining 35% was spread over the other seven provinces indicating a need for the installation of processing and recycling facilities suited to match the lower outputs and transient material flows.

3.3.2.   Estimating e-waste volumes in South Africa

The growth rate of WEEE is a general indicator of the generation rate of WPCBs. It is estimated that of the WEEE volumes estimated, WPCBs comprise between 3% and 5% of the total weight of WEEE (Hadi *et al.,* 2015).

The growth rate of electronic waste and subsequently WPCBs is stimulated by the push for innovativeness in the electronic sector to address the changing needs of the market. Improving productivity, safety and advancing operational efficiencies of large commercial enterprises, has provided a significant stimulus for the development of robotics and computing software (Akcil *et al.,* 2015). On the consumer side, increased personal disposable income within the populations of emerging economies and a desire for a better quality of life has stimulated the growth of the consumer electronic and electrical goods sector.

Previous research findings revealed that e-waste volumes comprising of white goods, consumer electronics and IT are within households and range between 1 000 000 and 2 200 000 tonnes (Finlay & Liechti, 2008). Estimates described in alternative investigations were far lower than those reported by the researchers (Godfrey, 2015).

Fetzer (2009) reported an e-waste generation rate of 100 000 tonnes per annum and added that 20% of this tonnage is recycled by formal recyclers whereas another report alluded to a tonnage of 322 000 per annum (GreenCape, 2017). Results of the survey revealed that not less than 36% of e-waste in the country is comprised of consumer electronics and IT goods. Recent market surveys have projected a compounded annual growth rate of 7.3% for ICT products which are of great interest in the e-waste recycling business, due to the high intrinsic value (GreenCape, 2018). Between 15 000 and 43 000 tonnes of PC equivalents are reported to finally end up in the landfill. This is only a fraction of the e-waste accounted for.

Material outflows into the e-waste recycling chain are low. A summary of the published estimates of e-waste volumes in South Africa are summarized in Table 3.3. Attempts to develop a statistical study reference point based on waste generation trends influenced by macroeconomic variables were done by different researchers. Results generated were used to derive a dispersion model of WEEE generated by provinces (DEA, 2012; Lydall *et al.,* 2017).

Table 3.3: The published estimates of e-waste generated

Source	Generation rate of e-waste (Tonnes/year)	Estimated e-waste in storage (Tonnes)	Material being handled or received by recyclers (Tonnes per year)
Finlay and Liechti (2008)	-	1 129 000 to 2 108 000	-
Fetzer (2009)	100 000	-	-
DEA (2012)	64 045 with a global estimated growth rate of 4%.	-	-
StEP (2013)	For the year 2014 346 000 tonnes per annum were projected with an average annual growth rate of 10% per year.	-	-
GreenCape (2017)	322 000	-	-
GreenCape (2017)	-	-	315 000 to 555 000
eWASA (2017) Mail & Guardian (2018)	360 000	-	-

Based on the data summarized in Table 3.3 estimation of e-waste volumes generated within the country and in storage is still a grey area. The margin between the lowest and highest estimates stands at almost 850% indicating that the average global estimates cannot be used with great certainty for estimating e-waste growth rates in South Africa. The estimates rely on subjective data that is limited and dependent on the active participants in the available statistical data pool. In the absence of a central statistical national data base on e-waste the variations in reported data as displayed in Table 3.3 can be expected.

3.4.   E- waste recycling

3.4.1.   E-waste recycling in comparison to other waste streams

E-waste recycling rates have been shown to be the lowest in comparison to other recycling sectors. In a survey of the current status of the e-waste recycling landscape in South Africa, it was established that e-waste flows into the main recycling value chain are derived from the government. An analysis of the different waste streams in South Africa for the year 2016 was summarized graphically as shown in Figure 3.3. The recycling of e-waste reported the lowest activity which has been attributed to varied factors discussed in greater detail in section 3.6.
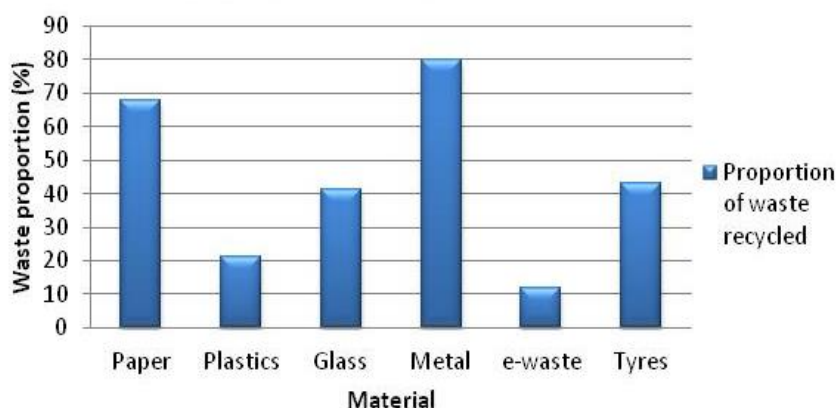
Figure 3.3: Proportion of waste streams recycled in South Africa in 2016 expressed as percentages of the total tonnage of waste handled (GreenCape, 2017)

The analysis of the current and potential market values associated with e-waste recycling activity, revealed a similar trend with e-waste reporting the lowest market size and revenue depicted in Figure 3.4.
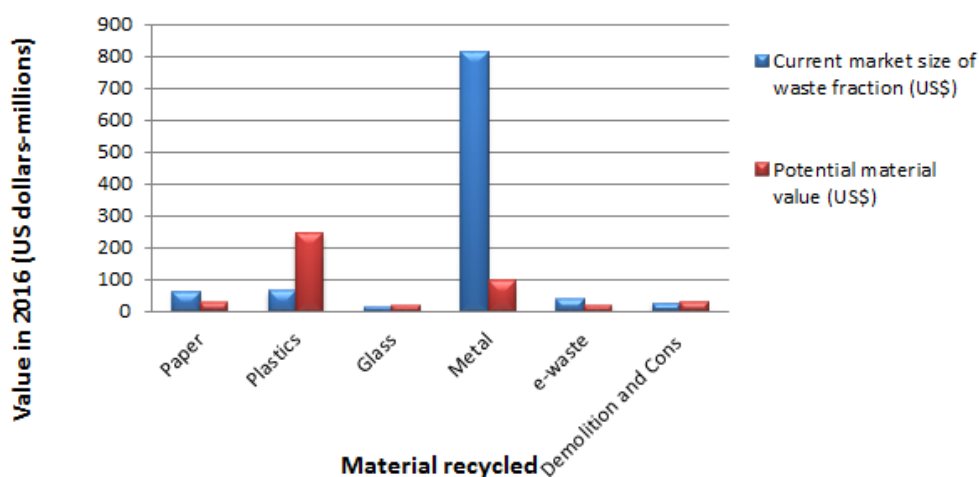
Figure 3.4: A comparative analysis of the current market size and potential material values of waste streams in South Africa 2016 (US$) (GreenCape, 2017)

The potential material value shown in Figure 3.4 describes the revenue realisable from sales of the unrefined or unprocessed waste stream in its raw form. Enhancing the quality of the final product along the waste recycling chain of each of the materials shown in Figure 3.4 improves the current market size (due to attractiveness of product to the market) of the associated waste fraction. This is evidenced by the superior market sizes associated with the paper, metals and e-waste fractions. The influence of global metal prices contributes to increasing the market size for the metal and e-waste fractions. Scope exists to boost the market potential and revenue realisable from the exploitation of greater volumes of e-waste and fractions that are directed to landfill. This is achieved through an expansion of marketable by-product streams derived from e-waste recycling and associated support activities. The expansion of the product portfolio is however ineffective without a corresponding market readiness and absorption capacity.

An analysis of e-waste recycling activity in 2016 revealed that 48% of the e-waste recycled in South Africa contains high value material in the form of WPCBs. This proportion combined both the recycling of consumer equipment and information technology as shown in Figure 3.5 (GreenCape, 2017). Based on the pessimistic and optimistic estimates, between 1 150 and 5 190 tonnes of WPCBs are recycled annually (Schluep *et al.*, 2009; Fetzer, 2009; GreenCape, 2017).

Based on the inferences made from the estimated recycling rates of e-waste in as at 2015 (Lydall *et al.*, 2017) it is estimated that 3 000 tonnes of WPCBs were processed and recycled in 2016.
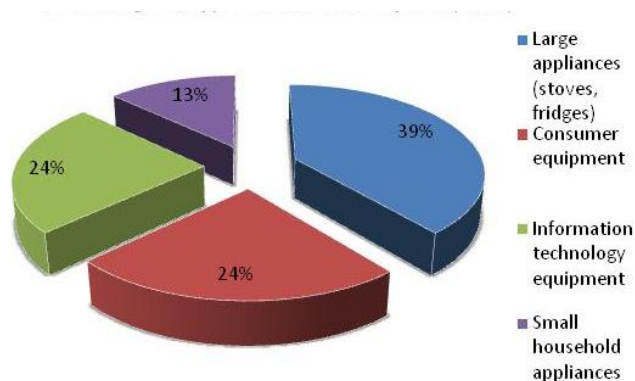
Figure 3.5: Analysis of the e-waste recycling activity in South Africa 2016 (proportion of tonnage processed) (GreenCape, 2017)

3.4.2.   Employment creation

The development of the e-waste recycling sector has the potential to generate employment. E-waste recycling is associated with a higher job factor of when compared to other waste streams (30 jobs per 100 tonnes e-waste diverted) (GreenCape, 2017). Figure 3.6 shows that e-waste has the largest jobs potential in comparison to the other major waste streams recycled in the country.
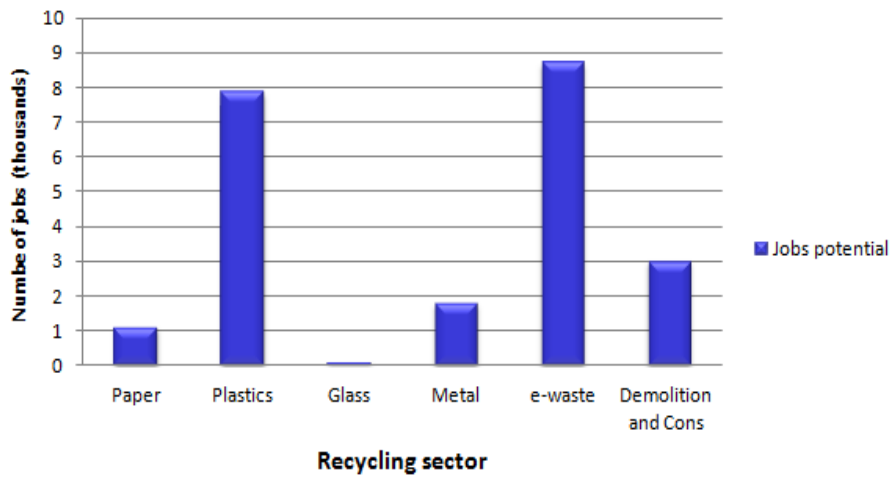
Figure 3.6: Potential jobs that can be generated from recycling of different waste streams in South Africa for 2016 (GreenCape, 2017)

3.4.3.   E-waste recycling activity

E-waste recycling activities are composed of the six interrelated elements the form the pillars of an e-waste value chain. The generic structure evolves according to the specific geographical location, nature of e-waste streams being handled, existing legislative structures and frameworks in place, the relevant technologies adopted for processing the e-waste and handling waste streams thereof and finally, the business models that define the targeted revenue streams. Figure 3.7 is an illustration of a generic picture of the WEEE recycling chain.

The closed loop philosophy does not apply to the model illustrated in Figure 3.7 It is a simplified version of the existing recycling system in South Africa. What is apparent from the schematic is that failure to maximise material recovery results in a net accumulation of metallic and non-metallic fractions in concentrations above the permissible quantities that disturb the ecosystem and inefficient utilization of resources expended to recover fewer targeted end products.

GENERIC WEEE RECYCLING CHAIN

WEEE within the system	Collection	Dismantling	Preprocessing	End Processing	Disposal
Domestic	Formal and	and segregation			waste
Commercial	informal	Formal and informal			Byproducts
Storage					Final products

Reuse

Handling of final waste
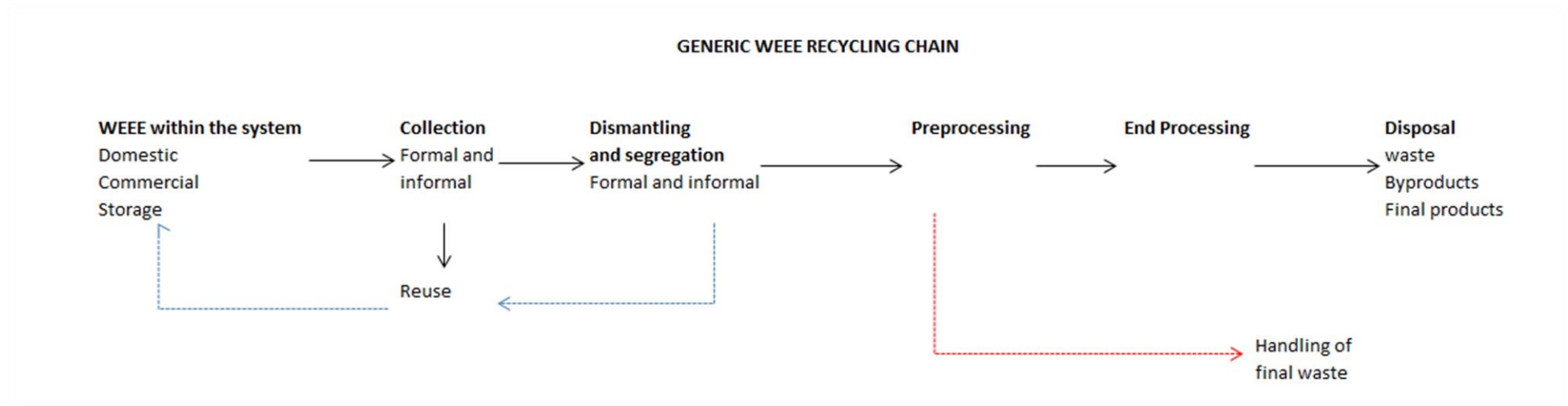
Figure 3.7: The generic WEEE recycling chain describing e-waste recycling activity in South Africa (Mouton & Wichers, 2016; Lydall *et al.*, 2017; GreenCape, 2017; 2018)

The existence of an established metals processing industry forms a critical supporting structure for the downstream processing of outputs from the pre-processing of e-waste. The sustained development of the e-waste recycling business in the country is therefore achieved (Widmer *et al.,* 2005). The exploitation of the country's vibrant metals processing industry and potential to upgrade and modify existing plants to process greater e-waste volumes, is a crucial step in bolstering South Africa's e-waste processing capability (Schluep *et al.,* 2009). The South African government has responded by intensifying the prioritization of e-waste management activities over the years, as highlighted by Godfrey & Oloefse (2017).

In recent studies it has been noted that with the full adoption of operation Phakisa, it is expected that outflows of e-waste into the e-waste recycling economy from government sources will be enhanced further (GreenCape, 2018). The bulk of the e-waste inflows are derived from the government. Figure 3.8 provides an illustration of the distribution of e-waste from the three major sources into the WEEE value chain as of 2017.
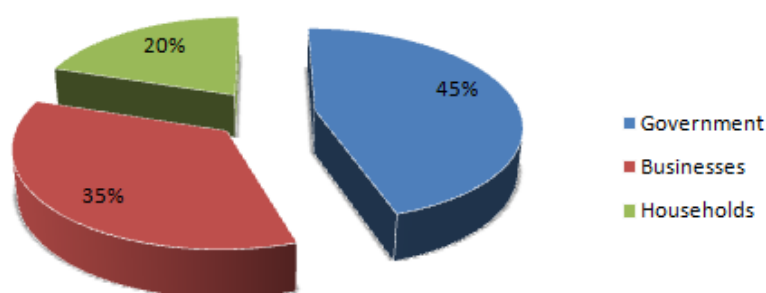
Figure 3.8: Distribution of e-waste inflows into the WEEE value chain in South Africa (2017) (GreenCape, 2018)

A study of 27 formal recycling companies in South Africa indicated that 30% of the companies studied actively export the WPCB material they handle. It was noted that the WPCBs are usually shredded prior to export as opposed to the selling of whole WPCBs done for the local market (GreenCape, 2017; 2018; Lydall *et al.*, 2017). Most of the companies studied are involved in collection and dismantling activities.

Figure 3.9 provides a summary of significant recycling technologies used by the surveyed registered recyclers (Lydall *et al.*, 2017). In addition to collection activities four major categories used in isolation or as a combination exist to process e-waste and include mechanical, pyrometallurgical, hydrometallurgical and bio-hydrometallurgical technologies. The different coloured bars depict the technology or activity associated with each of the recyclers. In South Africa the use of hydrometallurgical processing is not as well established as mechanical processing as revealed by the higher frequency of blue bars in comparison to the pink bars depicted in Figure 3.9. Great scope exists for expansion in this arena.
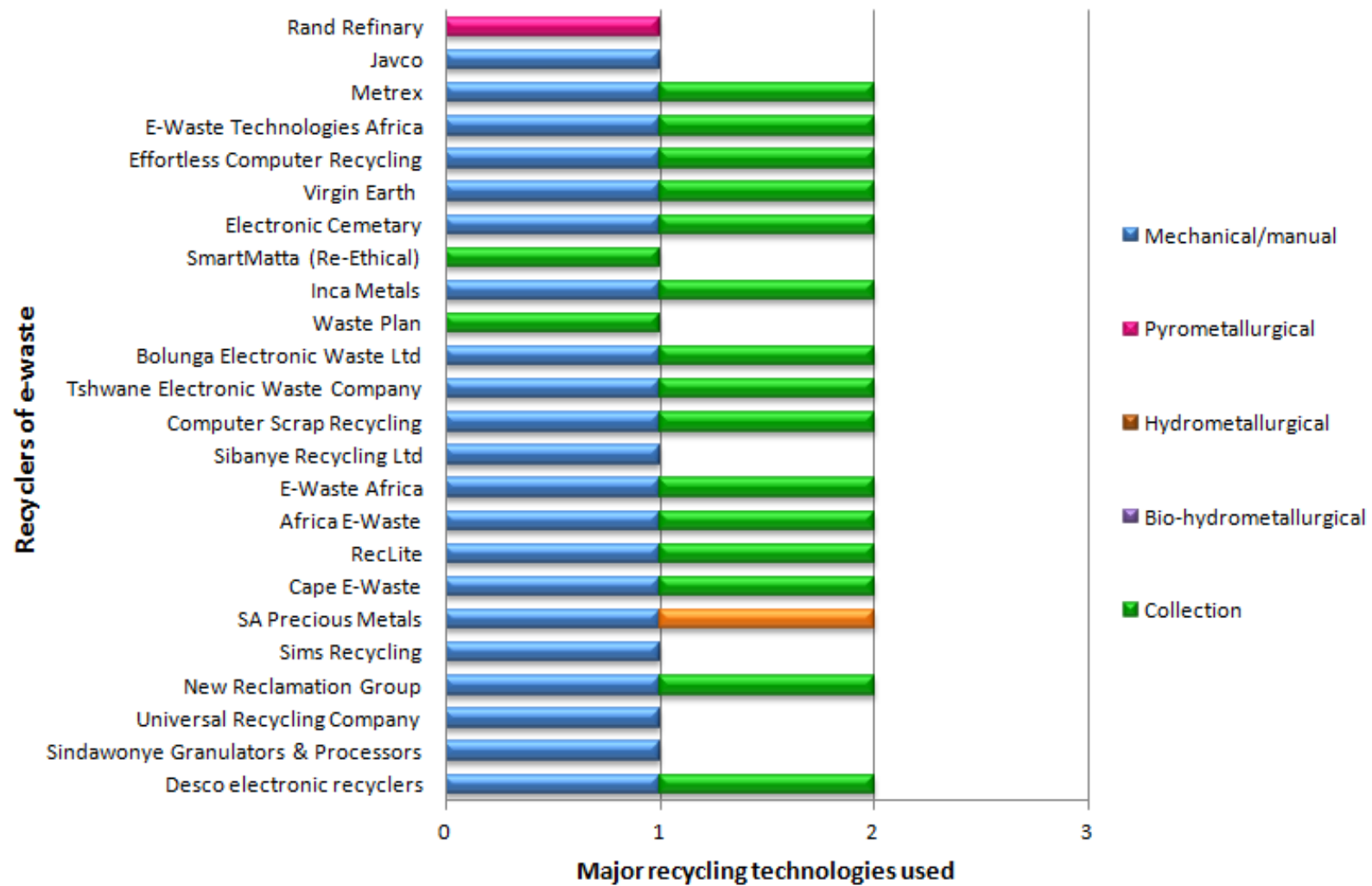
Figure 3.9: The distribution of technologies applied by some e-waste recyclers in South Africa (Finlay, 2005; Widmer & Lombard, 2005; Lydall *et al.,* 2017)

Based on the distribution of some of the formally registered e-waste recyclers shown in Figure 3.9, there is evidence that the dominant technology upstream of the WEEE value chain is associated with the manual and mechanical recycling of e-waste. The distribution of the formal recyclers along the WEEE value chain based on the focal points of their business models is depicted in Figure 3.10. The pie chart is based on results of a survey of registered e-waste recyclers (Lydall *et al.*, 2017).
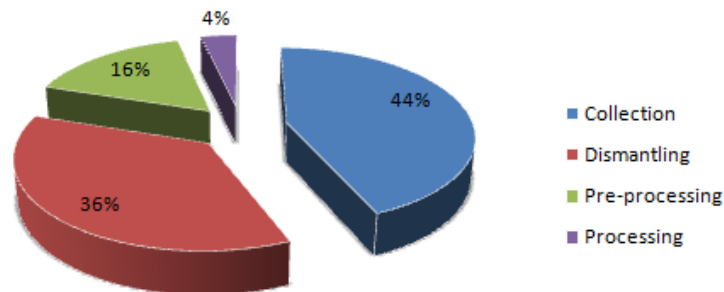
Figure 3.10: Distribution of some of the formal recyclers across the WEEE value chain (2017)

High barriers to entry were reported at the pre-processing and processing stages with respect to investment in technology. The reverse is true for the collecting and dismantling stages. The pyrometallurgical processing of WEEE on a larger scale is restricted by the low and inconsistent material flows of e-waste into the recycling chain. The current e-waste volumes reporting into the value chain cannot at present support the large pyrometallurgical processing of e-waste for the recovery of metals at economically feasible capacities. The operations cannot be expected to run at an annual capacity utilization in excess of 10% if one were to considered running operations at similar levels done by e-waste recycling firms in the more developed economies.

An analysis of the current value and supply chain reveals great scope for a hybrid horizontal growth in e-waste business as shown by the distribution of activity and technologies in Figures 3.9 and 3.11. Such activities can include marketing, consultancy services or waste disposal businesses revolving around the existent e-waste value and supply chain. Figures 3.9 and 3.11 further demonstrate the existence of a platform to exploit the established collection and dismantling network that is more mature than advanced value adding recycling activities such as the hydrometallurgical processing of e-waste in South Africa. The primary and secondary activities described in Figure 3.11 refer to the collection and dismantling and pre-processing activities that dominate the e-waste value in South Africa.
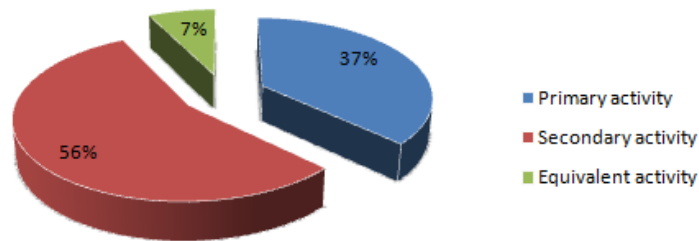
Figure 3.11: Analysis of the focus of WEEE recycling activity of some of the registered recyclers (Lydall *et al.,* 2017)

Small to medium scale enterprises have been identified as key drivers to achieving this growth. Mobile operating configurations are suited for this. At present vertical expansion is limited by the low e-waste volumes are being handled in the country.

Fluctuations in the key raw material supply (e-waste) and capacity utilization make it difficult for concerns to secure long term funding for further capital expenditures. Irregular revenue streams increase a firm's susceptibility to environmental risk factors.

Based on the baseline report published by the DEA (2012) and estimates of WEEE generation rates a supporting framework for planning collection and storage systems, configuration of recycling facilities and distribution centres is requisite.

3.4.3.1.   Status of the formal recycling sector

The formal e-waste recycling of South Africa is broadly characterised by the diverse in-homogeneity of pricing structures and material movement largely dictated by supply and demand factors. These factors influence pricing structures and the resultant adoption of the most feasible business models to maximise economic viability of a going concern.

Material flows

Low e-waste material outflows have led to the active business process re-engineering by formal recyclers. The players in the formal sector have undertaken the initiative to actively re-engineer their business strategies and models to circumvent the prevalent supply risk of raw material. This has been achieved by entering into long term supplier agreements and strategic collaborations along the e-waste value chain. Raw materials are also sourced from other parts of the country even in the SADC region. A degree of e-waste pre-processing is undertaken in the source countries. This has eased the burden of the costs associated with reverse logistics of bulkier material volumes being moved into South Africa.

The low material volumes of WEEE entering the recycling chain have restrained significant acquisition of large capital equipment for major expansion projects. The capacity utilization for the companies studied ranged between 10% and 60%.

Figure 3.12 provides a graphical summary of the distribution of the major fractions associated with formal e-waste recyclers. The recyclers surveyed represent not more than 30% of registered formal recyclers in the country (Lydall *et al.*, 2017). Based on the graphical plot, WPCBs, plastics, ferrous and non-ferrous metals and plastic materials commanded the largest market share destined for local pre-processing. The bulk of processing activity of WPCBs is done by foreign processors. Lydall *et al.*, (2017) reported that at least 90% of WPCBs are exported for final processing. The recycling of glass, phosphor powders and other fractions is still low. The trend depicted demonstrated the existence of a technological risk factor. The capacity to handle large outflows of glass and phosphors derived from e-waste is low. This is also aggravated by depressed market Figures for these streams.
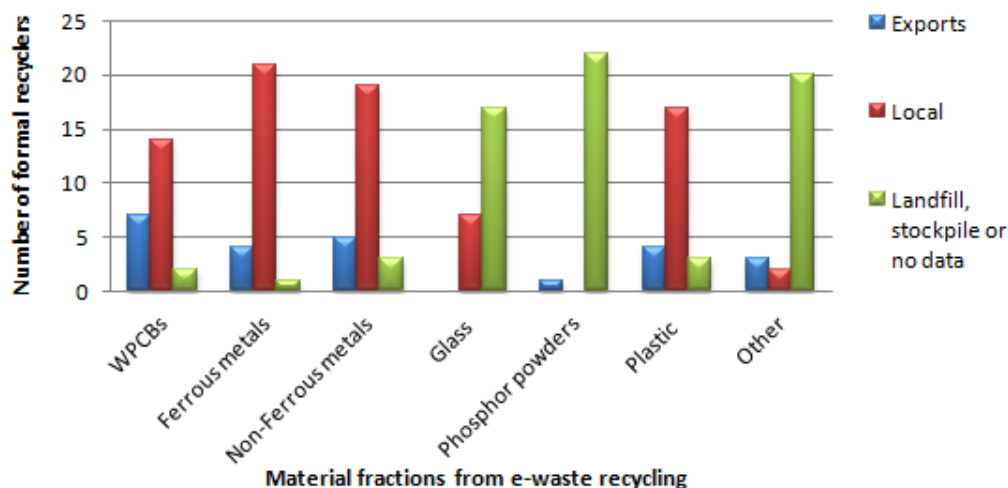
Figure 3.12: Market distribution for fractions derived from e-waste recycling by formal recyclers in SA (Lydall *et al.,* 2017)

An analysis of the distribution of recycling volumes handled by some of the formal recyclers in South Africa shown in on Figure 3.13 reveals that the bulk of e-waste handled is concentrated to a few recyclers. The five major e-waste recyclers processed almost 80% of the e-waste handled in 2015. All the recycling was associated with pre-processing and collection activities, with no further value addition through the exploitation of either one or both hydrometallurgical or pyrometallurgical technologies.

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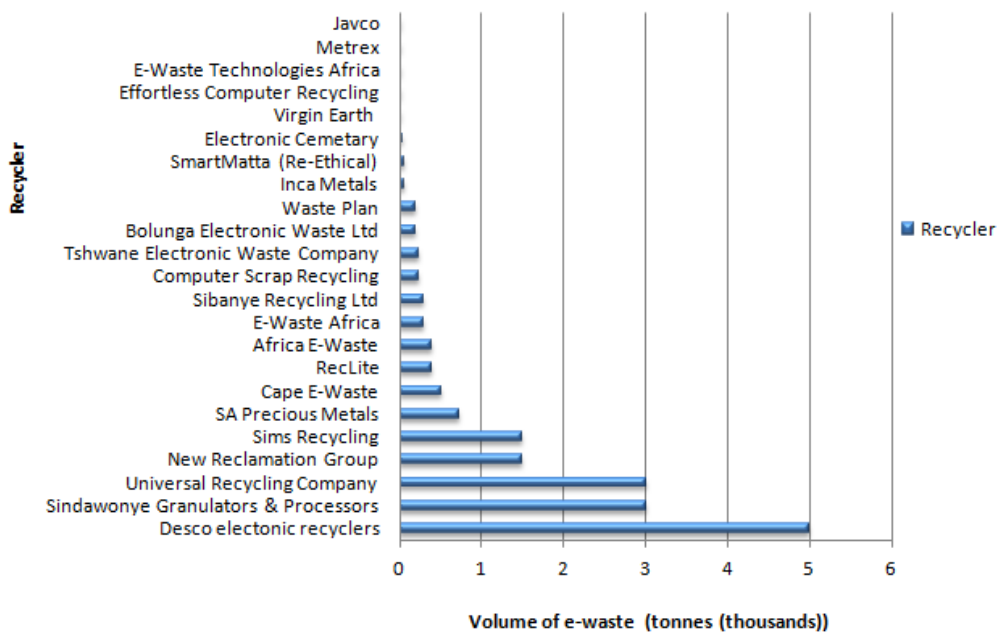
Figure 3.13: Volumes of e-waste handled by at least 25% of formal recyclers in South Africa (2015) (Lydall *et al.*, 2017)

Developing technologies

A trial hydrometallurgical process that has been developed in South Africa, for over 16 years to recover precious metals from WPCBs of all grades, is currently in operation, with reportedly no effluent and emissions discharges (Lydall *et al.*, 2017). The core activities are associated with the precious metals business. The precious metals are extracted and recovered as intermediate chemical products coupled with the production of copper cathode. The concern sources all its feedstock from the nine provinces of South Africa. Additional feedstock is supplemented by supplies within the SADC region, namely Zambia, Namibia, Botswana and Mozambique. The plant has a maximum processing capacity of 1 300 tonnes per annum of WPCBs but is however utilizing 50% of it.

Market dynamics

The predatory pricing of raw materials is low due to the need to maintain inflows of materials. Larger more established firms command a significant bargaining power. Recycling of WEEE is generally a low margins business. The combined refurbishment of e-waste with recycling and export of processed e-waste are the more favourable options for most of the recyclers. The export revenues exceed domestic revenues by a 70% margin (Lydall *et al.,* 2017). Upstream activities in the recycling chain are labour intensive. Most of the recyclers recruit labour on a contractual basis as opposed to running with a large permanent work force. The degree in the variation of the customers' specific requirements has slowed movement of selected by-product fractions from the processing of e-waste.

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The absorption for example of complex impregnated with flame retardants, glass and phosphors is still low in comparison to other fractions recovered from the recycling of e-waste (see Figure 3.12).

3.4.3.2. The informal sector

This sector has not been well documented, and no concrete statistics are in place to account for informal recycling and dumping practices. A divide still exists between the formal and informal sectors. Operating structures are loosely defined and prone to high levels of risk within the business environment. Finlay & Liechti (2008) have estimated that between 40 000 and 50 000 informal recyclers are active in South Africa. Regulation of activities in this sector is largely rudimentary. Attempts to regulate the activities of this sector have been driven largely by the intervention of nongovernmental organizations (Lawhon, 2013).

Researchers in the field have argued that the low level of synchronised activities of both the formal and informal sectors has contributed towards the observed inefficiencies in the e-waste management and recycling scene in both transitioning and emerging economies.

Efforts are however in place in some of these countries to promote sustainable management of e-waste in an economy that has been and continues to witness a burgeoning of e-waste (Chatterjee & Kumar, 2009; Jain & Garg, 2011; Chatterjee, 2012).

The barrier to the successful merger of the formal and informal sectors is reinforced by the availability of exploitable economic benefits of e-waste recycling (Chatterjee & Kumar, 2009; Shamim *et al.,* 2015). Unless equivalent or superior benefits can be derived from a marriage of the two sectors, significant effort warranting punitive measures, where potentially applicable, would have to be enforced. This demands more policing to effectively monitor the success of any proposed strategies to broaden the involvement of participants in e-waste management endeavours. If the systems are prone to corruption and monopolistic behaviour, this will further sabotage progress and the success of legislative measures to manage e-waste on a consensus between the formal and informal sectors.

3.4.3.3. Overview of stakeholder analysis in South African e-waste economy

A highly integrated network exists among the recyclers within the South African e-waste recycling landscape (GreenCape, 2018). The output from smaller recyclers both formal and informal is eventually passed on to the large recyclers. No data base exists however, to accurately monitor the inflows of e-waste from the informal recycling sector.

The rather centralised positioning of major e-waste recycling facilities has however limited the extent to which other parts of the country can adequately benefit from the activities of larger formal e-waste recyclers (Mouton & Wichers, 2016). Table 3.6 provides an overview of the stakeholder analysis in the South African e-waste recycling chain.

Table 3.4: Summary of key stake holders in the South African e-waste recycling chain (Widmer & Lombard, 2005; Finlay, 2005; Lydall *et al.*, 2017; GreenCape 2017; 2018)

Stakeholder	Description
Importers and manufacturers	The white goods sector is dominated by foreign brands. Commitment of the foreign companies is non-existent or lacking in commitment. They are not involved in e-waste recycling or take back activities of large electronic consumer goods.
Distributers	Includes major retail chains and smaller stakeholders. Some of the retailers form crucial links in the e-waste recycling chain by hosting collection bins on their premises. Collection of white goods is complicated by the need for more land space.
Collectors and facilitators	Informal collectors are reported to contribute 25% of the e-waste volume handled in SA. Formal collection is typically associated with contractual and structured agreements with the government and large businesses. The formal collectors include large integrated waste collection companies, smaller operators and consumers. The formal collection normally handles large consignments of e-waste. Stealing of collection bins occurs so that the collected e-waste is lost to the informal sector. The informal collectors are involved in the salvaging and scavenging of e-waste at municipal waste dumps. There is a high level of risk and criminal activity associated with their activities.
Refurbishers	Through the refurbishment of devices and gadgets the useful service life of EEE is prolonged, delaying entry into the main stream recycling chain. Some of the refurbished material is exported while the other material is sold locally. Refurbished gadgets and devices are derived from imports from both the international suppliers and the continent.
Recyclers	Recycling operations are not uniform. They are predominantly characterized by collection, dismantling and mechanical recycling activities. Pyrometallurgical, bio-hydrometallurgical and hydrometallurgical processing activity is still low. Hydrometallurgical processes fall under strict control but the looming threat of the illicit extraction operations exists. Major exports by formal recyclers include PC boards, non ferrous metals and plastics. The main operations are based in Gauteng and Western Cape provinces. Stock piles of hazardous material like nickel cadmium batteries or lead containing materials are held until a time they are specially disposed of in a registered landfill. The e-waste was exported in its raw form without further processing to maximise its value as a result of metal enrichment.
Downstream vendors	Downstream vendors further process and refine material from recyclers. They include plastics recyclers and metal refiners and smelters. The major precious metals refinery Rand refinery deals with gold refinery. Expected to increase capacity of operations by 5% at the time of reporting. A comprehensive list of metals refiners is available from the South African Department of Minerals and Energy.
Final dispensers	The recyclers themselves can be final disposers. Upon failure to recycle some of the residue or by-product, the material is ultimately disposed of at the hazardous waste disposal sites managed by the municipality. The illicit disposal of smaller electronic gadgets also occurs. Discarded fractions from activities of informal recyclers also occur.

3.5.   Proposed models for e-waste management

GKPER and SECO initiative

Models have been developed to address e-waste management and recycling activities in the country. Small scale e-waste recycling projects were initiated in Cape Town in 2004 supported by Global Knowledge Partnerships in e-waste Recycling (GKPER) a brain child of Swiss State Secretariat of Economics Affairs (SECO). The small-scale programmes sought to achieve a decentralised approach to e-waste collection and dismantling and broadly create a pipeline through which local awareness on the significance of proper e-waste recycling could be undertaken in more remote regions. Through the success of the pilot programme, it was expected that it would be the representative model upon which other regions lacking in e-waste management programs of a similar nature could build upon (Schluep *et al.,* 2009).

The Cradle to Cradle EEE management system

The Cradle to Cradle EEE management system seeks to maximise material reuse, reduce disposal rate and hence landfill accumulation rates and formalise job creation in the e-waste recycling sector (Widmer & Lombard, 2005). The institution of this model is dependent on the active and continued positive partnership of concerned stakeholders. The researchers noted that the institution of this model would be dependent on the active and continued positive partnership of concerned stakeholders that would include the interested and affected parties, consumers, traders and ICT industry captains. Stumbling blocks of a socio-political nature have limited the effective institution of wealth creating initiatives associated with e-waste recycling (Widmer & Lombard, 2005).

Modified Swiss adapted approach

A four-step process flow chart embracing the formal and informal sectors and existent industries was described by Chatterjee & Kumar (2009). The model was designed to enhance e-waste management practices in India. The flow chart encapsulates ideas that can be applied to the South African scene such as the exploitation of existent supply and value chain relationships in the current recycling chain. Widmer *et al.,* (2005) developed a model that specified the key players and activities in the sector. The features of the existing model embrace concepts from the Swiss structure.

Three step framework for developing e-waste strategy

A three step methodology to developing a structured approach to e-waste management was advanced by Schluep (2014) (see Figure 3.1). The methodology entailed a detailed assessment of a country's legal structure, socio-political environment and materials inventory and infrastructural capabilities to develop a structured strategy involving key stakeholders.

The obligations of the stakeholders are clarified regarding policy and legislation, business and finance, technology and skills, monitoring and control and marketing and awareness. The final step involves the implementation phase which must be followed up, monitored and evaluated against set performance criteria.

The nation can draw lessons from the success of initiatives championed by EMPA and GIZ and the progress in Kenya (Schluep *et al.*, 2009). It was reported that in Kenya small scale models for collecting e-waste through the active engagement of the informal sector and formal sector comprising of formally registered collectors, demonstrated that sustainable e-waste initiatives can be adopted in developing and transitioning economies and regions. Proper training at all levels, the formalisation of the informal sector coupled with strong government support, is key to the success of such models.

South African Tailor-made approach to e-waste recycling and disposal

Based on detailed research and findings of site visits within South Africa and Europe, a model for e-waste management in South Africa was developed (Mouton & Wichers, 2016). The model considered the existent weaknesses in the prevalent South African system, chiefly: the non-uniformity of recycling systems in the country, inadequate infrastructure to support collection, recycling, refurbishment and disposal of e-waste and its by-products. The researchers proposed a holistic model that has attempted to address legislative and financial controls, define supporting structures, enhance and homogenise collection activities of e-waste, provide for monitoring and control and cover all geographical locations to ensure that no regions are isolated from e-waste collection activities.

Measures were also proposed to safe guard against corruption and other objectionable business practises that countermand sustainable e-waste management activities (Mouton & Wichers, 2016). Within the framework of this model all four approaches to EPR were embraced. An analysis of the assessment indicator system developed by Widmer *et al.*, (2005) parallels the architecture of the proposed model by the researchers.

Main observations of the proposed models

The areas crucial to enhancing the e-waste management and recycling activities in South Africa were identified. Widmer *et al.*, (2005) described five major parameters that must be considered during the evaluation of an adopted e-waste management system or model. The parameters address the following areas: system coverage, financing of the system, producer responsibility, compliance and legal regulation. A broad evaluation of the proposed models and approaches to e-waste management was done. The evaluation was based on the observed areas supporting the South African e-waste management landscape (Dittke, 2007; Lawhon, 2013; Lydall *et al.*, 2017) and the five aforementioned parameters described by Widmer *et al.*, (2005).

The broad assessment is summarised in Table 3.5. The general picture over the last decade has shown a trend towards increasing the scope of issues addressed in developing a more efficient e-waste management system in South Africa. Earlier models did not clearly define specific methods for improving data collection and management in the e-waste economy. Recent research has however alluded to a concerted drive towards the need to improve data capture and management in the South African e-waste recycling economy (Lydall *et al.*, 2017; GreenCape, 2018).

Table 3.5: Comparative evaluations of proposed e-waste management models

Areas addressed by the proposed models	Cradle to cradle EEE management system. Widmer & Lombard, (2005)	Modified Swiss adopted approach. Widmer *et al.*, (2005)	Global Knowledge Partnerships in e-waste Recycling (GKPER) & SECO small scale initiatives. Schluep *et al.*, (2009)	Frame work for developing e-waste strategy Schluep (2014)	A model for effectively recycling and disposing of e-waste in South Africa Mouton & Wichers (2016)
Formalising job creation and improve synchronization of the formal and informal sectors	Yes	Yes	Yes	Yes	Yes
Moving towards a more circular economy	Yes	Yes	Not clear	Yes	Yes
Financing the national waste strategy on e-waste management	Not clear	Yes	Not clear	Yes	Yes
Expansion of market absorption capacity	Not clear	Not clear	Not clear	Yes	Yes
Mitigating high logistics costs and improving e-waste collection rates	Not clear	Not clear	Yes	Not clear	Yes
Improving data collection and management	Not clear	Yes	Not clear	Not clear	Yes
Increasing awareness of society on e-waste and scope of stakeholder involvement	Not clear	Yes	Yes	Yes	Yes
Homogenize adoption and execution of regulations and statues on e-waste management.	Not clear	Yes	Not clear	Yes	Yes

3.6.   Threats to successful e-waste management and recycling

3.6.1.   Supply chain dynamics - material flows

Researchers in China, India and the United States of America for example, have acknowledged that the low collection rate of e-waste is one of the most significant retardants to achieving a high recycling efficiency (Kang & Shoenung, 2006; Patil & Patil 2015; Li *et al.,* 2016). Between 70% and 90% of known e-waste in South Africa is reported to be in storage pending processing and safe disposal (Lydall *et al.,* 2017; GreenCape, 2018). The result has been the high inventory of e-waste and hazardous material in domestic governmental and private institutes. It has been revealed by researchers that a significant supply risk of e-waste into the main stream recycling chain is prevalent. The result is that most recycling concerns operate at capacities not exceeding 60% (Lydall *et al.,* 2017). This not a financially sustainable model conducive to supporting expansion or major recapitalisation projects as alluded to earlier. The existence of a centralised geographical location of the major e-waste collection centres further erodes the financial merits associated with the surrender or collection of e-waste (Finlay, 2005; Mouton & Wichers, 2016).

3.6.2.   Data management and planning

The accurate accounting and validation of the specific volumes is still outstanding though efforts are in place to actively remedy the deficiency through the establishment of a structured legal framework. Efforts are in place to institutionalise the reuse, recovery and recycling through the National Waste Management Strategy that was approved by cabinet in 2011. Responsibility for collection and management of the data will fall under the auspices of national data repository headed by Statistics South Africa. The efficiency of this strategy is hinged upon the accurate recording of information on waste which has been a challenge (DEA, 2012; Lydall *et al.*, 2017).

The factors contributing to the lack of a broad updated e- waste intelligence data base include the absence of a mandatory enforcing regulation, the sensitivity of the information and the potential implications associated with releasing of obsolete storage devices sensitive data. The available statistical data pool is yet to be validated. There is also an observed a lack of analytical data pertaining to waste streams. The organizations that do take the initiative to populate the statistical pool do so, on a voluntary basis. A formalised mechanism for collecting data from the informal sector is yet to be established. Variations exist in the quantity and type of data collected and the proportion of active companies furnishing their WEEE statistics. This further weakens the validity of the estimates (Godfrey, 2008). The meaning of this that known statistics tracing the life cycle of EEE through the value, supply and recycling chains in South Africa can either be of two extremes; pessimistic or optimistic, which complicates planning and budgeting for large capital expenditures for e-waste recycling and associated management activities.

The future implementation of the South African Waste information system (SAWIS) has been considered to enforce reporting of the discussed statistics pertaining to waste management (DEA, 2012).

3.6.3. Legislative and regulatory climate

In section 3.2.2 of this discussion, it was noted that the absence of specific legislation documented for e-waste management is the dominant feature characterising the legal and regulatory climate in the country. This loop hole has created a window through which illicit recycling activities proliferate in the nation. Backyard recycling operations are a threat to the environmental sustainability and well being of the affected communities.

The absence of clear policy and legislation complicates the merger of the formal and informal sectors due to lack of regulatory and monitoring measures that are systematic and quantifiable. The informal recycling sector is notorious for poor metals recoveries while accompanied by a significant level of environmental pollution originating from the uncontrolled burning, disposal of scavenged material and disposal of toxic acidic leach solutions (Finlay & Liechti, 2008). It is feared that the existence of illicit recycling operations much like those of Nigeria, Ghana, China and India are in existence in South Africa. It is however uncertain to what extent the challenge is rampant in the country (Furter, 2004).

The existence of restrictive and complex legislative requirements delays the process of regulatory compliance. The application of relevant operating licences is a lengthy process that is counterproductive to recycling operations. Delays in processing documents have culminated in some cases into losses of crucial supply contracts. Some of the legislation has an impact on the applicable technologies that can be selected by e-waste recyclers and this can have a bearing on the profitability of their operations (Dittke, 2007; Godfrey & Oelofse, 2017; Lydall *et al.*, 2017).

It has also been observed that the cost of compliance in the last 30 years has shown an almost proportional growth to the number of instruments being enacted. The result has witnessed the progressive erosion in competitiveness as profit margins are being trimmed at the cost of compliance (Godfrey & Oelofse, 2017).

National and provincial government only oversees waste management of local authorities. They are not active participants in the absence of a well defined e-waste management policy and legislation (Dittke, 2007). This arrangement limits the impact of regulations enforced when an arm's length approach is taken by the government.

3.6.3.1. New Quasi agency conflict

Views regarding the urgency of e-waste management and recycling are varied. As a result, the level of priority accorded to e-waste management for a number of years was vague thereby delaying the structured institutionalization of e-waste management policy and legislation. The present set up of the system has favoured the export of varying degrees of recycled e-waste contrary to government's agenda targeting value addition along the e-waste recycling chain (Godfrey, 2015; Lydall *et al.*, 2017). In the absence of a well defined pricing strategy for e-waste material to be recycled and metals recovered from the treatment operations promotes unhealthy competitive behaviour in a growing business niche.

3.6.3.2. Capacity building

The current e-waste recycling activities in South Africa are characterised by the land filling of plastic material impregnated with flame retardants that are not chemically bound to the plastic matrix. The mobility of these substances into the ecosystem is probable. The fate of some of the batteries removed from EOL EEE it was noted was unknown, though efforts more recently are now underway to recover metals through recycling of batteries (Finlay & Liechti, 2008). Research has also revealed that some recyclers are not ISO 14 0001 compliant. As a result of improper storage of e-waste and failure to adequately treat effluent water, the potential for ground water contamination is significant.

3.6.3.3. Transnational co-operation and responsibility

IT companies on the other hand affiliated to industry associations and bodies have been and continue to be involved in refurbishment projects. The reluctance of some multinational companies to participate in e-waste management drives on the domestic scene is a set back to the establishment of EPR in South Africa. Producer countries are signatories to protocols regulating the life cycle of electrical and electronic goods. Commitment to the mandatory regulations is limited (Finlay & Liechti, 2008). As a result, this has undermined the overall objective of solving the global e-waste challenge in regions of the world without the requisite capacity and infrastructure to sustainably handle e-waste.

3.7.   The key findings – A PESTEL analysis

In order to summarize the key findings made in the study of the South African e-waste management and recycling scene a holistic approach was used to address all the relevant facets of the country's business environment. The PESTEL approach is a tool that is originally used to evaluate the environment of a business as a part of assessing conditions for growth and strategic realignment.

The acronym describes the political, economic, social, technological, environmental (influences to a concern's operating activities and tactical decision making) and legal facets of the business sphere. This tool was adopted in the analysis of the country's e-waste recycling economy assess the status of e-waste management and recycling in South Africa as the operating environment.

The approach enabled a detailed stakeholder relationship to be critically interrogated across the value and supply chain in the e-waste recycling operations. Figure 3.14 illustrates the elements of the PESTEL approach used in this study. The key findings from the study of the South African e-waste scene are summarized using the PESTEL approach.
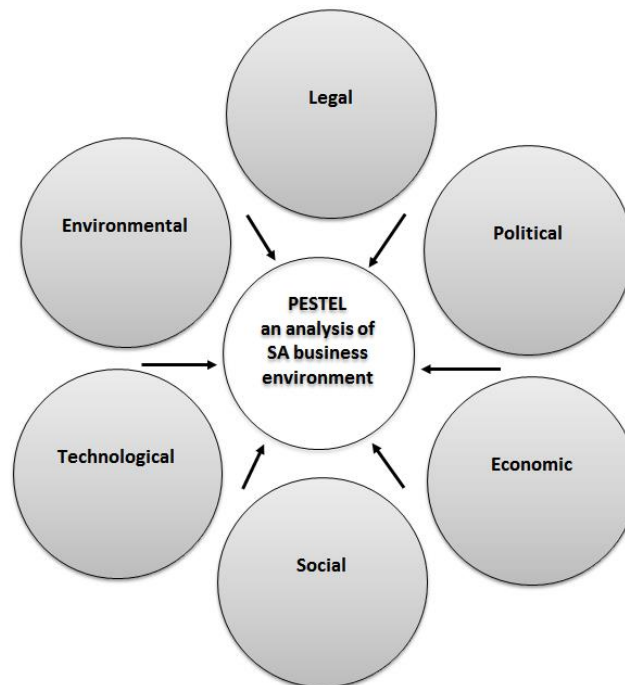
Figure 3.14: The PESTEL approach to analysing e-waste management and recycling in South Africa

3.7.1.   Political

E-waste recycling has been identified as one of the pivotal solutions to alleviating the high unemployment rate particularly in the unskilled population groups. Pilot projects initiated by interest groups in partnership with foreign NGOs were launched in response to the urgent need to create employment through small scale establishments that can easily be set up in the different provinces.

Concerns have been raised at the potential value that is being lost to export markets. The government has increasingly been promoting and prioritising efforts by different research groups in the country towards the increased recycling and value addition along the e-waste recycling chain. The value addition is targeted at the exploitation of all materials associated with e-waste with a strong interest in the recovery of precious and rare earth elements. The traditional disposal of e-waste is being rejected in favour of waste diversion from landfills and value addition through recycling.

In response to the growing urgency to manage the growing volumes of e-waste an association was formed in an attempt to homogenise and regulate e-waste management activities through the formation of e-WASA. Through this body stakeholders' involvement is drawn from across the value and supply chain. Diversion of waste from landfills has also been supported by the efforts leading towards the achievement of a more circular and green economy by the promotion of Extended Producer Responsibility (EPR). This has been evidenced by the changing face of the waste management bills enacted with an effort towards effectively crystallizing the regulation and management of the e-waste recycling sector (Godfrey & Oelofse, 2017).

The need to merge the formal and informal sectors is pivotal in curtailing illicit activities detrimental to human health and the environment, violence and crime. The study has revealed that no association or collective forum exists for the informal sector.

The stringent enforcement of legislation may not be the adequate solution alone: the legal and regulatory instruments must be concise, well structured with measurable and obtainable objectives while identifying the relevant supporting structures to ensure and aid compliance (Schluep, 2014). South Africa is a significant participant in the global economy and therefore must judiciously enforce any legislation with conscious regard to the potential implication of future global trade.

3.7.2. Economic

Asian and European markets have been the major recipients of South African exports of recycled e-waste products. The export of processed e-waste is greater than local sales suggesting that the continued existence of the recyclers is dependent on a predominantly export based business model.

Current activity in the e-waste economy is characterised by the reuse and refurbishment activities that offer superior margins than recycling activities. Research findings revealed that some of the recyclers combine other activities to boost revenues into their businesses.

The market responsiveness is demand driven and this affects the movement of material from the recyclers. This has resulted in a low processing activity for some materials.

A large proportion of e-waste in South Africa is projected to be derived mostly from white goods in the near future (Finlay & Liechti, 2008). Planning for the expansion and large scale diversification of e-waste recycling activities is complicated by the lack of sufficient data. The available data has not been verified and is heavily reliant on subjective estimates.

In earlier remarks it was noted that the quality of recyclable high value e-waste is decreasing and this trend is expected to continue. This has a strong impact on the future dynamics of the e-waste recycling sector in South Africa. The economy in South Africa has not matured to the level that it can absorb large outflows of low value by-product streams. A clear plan for the operations and continued research must be merged to ensure a sustainable and viable contribution from e-waste recycling in coming years.

3.7.3.   Social

The output of informal activities feeds into the more structured settings within the e-waste recycling sector. No concrete statistics exist for the number of participants in the sector. The activities are diverse and range from the collection of special parts or materials like copper for resale to larger collectors who in turn supply even larger collectors. The nature of operations affords greater liberties as a result of being exempt from environmental regulations. Some of the operations are associated with activities that endanger human health and the environment. The participants in this sector are vulnerable and possess little or no bargaining power for the material they salvage for sale.

The high supply risk and cyclic raw material supply forces larger players to employ the bulk of unskilled labour on a contractual basis as opposed to running on permanent labour contracts (Lydall *et al.*, 2017). The majority of participants in the informal e-waste recycling value chain in the country comprise the unskilled segment of the population. The input from this highly unregulated sector forms a crucial feeding point into the country's e-waste value chain. Input from this segment of the population trims the costs associated with the reverse logistics that could be incurred by larger formal recycling companies associated with the collection and dismantling of e-waste in the country.

The successful merger and regulation of informal activities is therefore a priority agenda in the country's broad e-waste management and employment creation initiatives. The success however of small pilot initiatives provides a more permanent solution for absorbing the available unskilled work force. Findings in this study have shown that the greatest job potential when examining waste management activities in South Africa is associated with e-waste recycling (GreenCape, 2017).

3.7.4.   Technological

The country has been making strides towards a more sustainable e-waste management model. While general waste management activities in South Africa focus on waste collection and land filling, e-waste recycling has been on the spotlight for over 13 years.

While there exists a vibrant metals processing industry, attempts to adopt large scale WEEE facilities like the Noranda process in Quebec Canada or Umicore, Hoboken Belgium, command substantial capital outlays with low supply risk operations. To date however, some of the firms have been absorbing output from e-waste pre-processors to recover precious metals like gold and PGMs. Some of the metal processors have signalled intentions to expand their processing capability for handling e-waste feed streams. The operations of most recyclers have however remained conservative, largely due to low and irregular raw material inflows. This has slowed the implantation of capital upgrade and operating expansion projects of most of the recyclers.

South Africa is also supported by a vibrant research community. The continued research and development expertise contribute towards value realisation, operating and extraction efficiencies in the field of e-waste recycling. Further research associated with e-waste management provides relevant material for debottlenecking areas retarding progress in the sector as a whole.

Capacity exists for developing technologies tailored for the needs of local manufacturers in the e-waste recycling business. More established recyclers and pre-processors have developed equipment for the in-house processes. The option for importing equipment from Europe and Asia also exists.

Dismantling and mechanical processing options are the more established technologies in South Africa. The capacity to expand pyrometallurgical processing is restricted by raw material supply risk and the larger capital outlay that must be committed at the onset. To date however, there are no pyrometallurgical facilities solely dedicated to e-waste smelting and refining operations.

In this study it was shown that Hydrometallurgical recycling of e-waste is slowly gaining traction. One registered recycler has been reported to run a small scale operation with a process flow sheet that has been improved for almost two decades. The operations of the hydrometallurgical plant can range between 650 and 1 300 tonnes of WPCBs per annum. The recycler sources feed stock from within South Africa and other SADC States.

3.7.5. Environmental

The e-waste recycling is characterized by a mix of formal and informal activities. The activities of larger formalised recyclers tend to be more concentrated in the economic hubs of the country. The smaller players are more dispersed while concentrating their activities in the region of origin.

It can be seen that the investment climate for exploitation is favourable. Opportunities for investment in e-waste recycling within the e-waste recycling chain are available in the area of research, industry, consultancy in e-waste management, small business development and various training programs.

An analysis of the recycling rate of all waste streams in the country has revealed that e-waste has the lowest rate. High barriers to entry upstream of the recycling chain with respect to the active participants were identified. Low barriers to entry exist downstream of the recycling chain with respect to value adding activities

The prevalent raw material supply risk however has altered the business dynamics across the value chain. The high supply risk and centralized activity of major players has forced entrants and established recyclers to reconfigure their business models to ensure survival.

The recycling rate of e-waste in the country has not exceeded 20% and in comparison with the recycle rates of more established economies, the recycling activity is still in the infantile stages especially further down the e-waste recycling chain. The operations of formal recyclers do not exceed an annual capacity utilization of 50%. The depressed e-waste collection rates have culminated in a low cumulative efficiency and the overall recycling rate.

3.7.6. Legal

A unique legislation for e-waste management and recycling activities in South Africa is lacking. The enforcement of legislation affecting e-waste management and recycling in South Africa lacks homogeneity. It varies along the different government levels. The existence of different municipal by-laws contributes to this variation. Some by-laws do not contribute positively towards e-waste management hindering operating activities (Dittke, 2007).The current legislation does not provide for the protection of stakeholders in the informal sector. Further it does not provide for the regulation and a means for monitoring informal e-waste recycling activities.

There still lacks a means of regulating the flow of and handling of materials in the supply chain. No pricing strategy exists for appropriately valuing e-waste. As a result, irregularities exist across the value chain. Accounting for value realised from e-waste is subjective.

The enforcement of extended producer responsibility as a means of enhancing e-waste management activities is still in its infancy. As yet it is more voluntary than it is compulsory while the active participation by foreign suppliers of white goods remains unsatisfactory.

3.8. The future outlook

Effective e-waste management is achieved through a synergy existing among researchers, industry, consumers and government. It also embraces a balanced marriage of access to the principal raw material, availability of the market for the waste and associated by products of e-waste processing activities, the technical infrastructure and competences to support e-waste management activities and sound economic viability (Widmer *et al.*, 2005; Schluep *et* al., 2009; Schluep, 2014; Mouton & Wichers, 2016, Lydall *et al.*, 2017; GreenCape, 2017; 2018).

The perceived future position of South Africa and e-waste recycling is better understood from a consideration of the findings of the research conducted. It is argued that with the appropriate support and structures in place South Africa will be better positioned to be an efficient e-waste recycler with the potential of being the role model for efficient recycling practices on the continent (Schluep *et al.*, 2009). It has been predicted that as South Africa is positioned to be the recycling hub of the continent, the inflows of PC scrap and mobile phones from neighbouring countries like Zimbabwe, Botswana and Zambia, will be significant contributors to the e-waste volumes destined for recycling in the country (Schluep *et al.*, 2009).

Secondly the existent strong metallurgical process industry supported by vibrant research support networks and analytical capabilities help to equip the country to handle inflows of e-waste.

Thirdly there exists a large unskilled labour market in South Africa that can be absorbed into new mainstream e-waste recycling activities. A strong manpower base supports the long term national e-waste recycling goals.

Efforts to enforce a ban on imports into South Africa of e-waste however, may not be successful as a measure to curtail the illegal influx of e-waste into the country.

3.9. Chapter summary

An analysis of the e-waste recycling economy of South Africa formed the first objective of this study. In this chapter an appreciation of the legislation and other existing instruments influencing e-waste management and recycling was developed. An exposition of the history of e-waste recycling and management was made, by tracking the history of waste management legislation and its impact on the e-waste recycling economy in the country. Other key findings were also discussed in the close of the chapter. The PESTEL approach was used to summarize these findings that included:

- The absence of a unique legislation pertaining to e-waste management and recycling,
- The existence of an underdeveloped market and technological capacity for absorbing complex by-product streams from high volume processing of complex e-waste,
- A clear aggressive thrust towards the attainment of a circular economy through EPR,
- The prevalent low inflows of the key raw material into the mainstream e-waste recycling economy,
- Finally the absence of a concrete statistical data pool for planning purposes.

The weaknesses in the system and constraints affecting the efficient management of e-waste were brought to the fore. The discussion also highlighted the enthusiasm and the active participation of the government and non-governmental interest groups in addressing e-waste management in the country.

Tertiary and other institutions are actively involved in providing a solution towards homogenizing the stakeholder relationships and improving the existent infrastructure in place.

The study also examined the existent market environment and forces dominant on the e-waste landscape. South Africa as a business environment was analysed and found to be in the early stages in comparison to the developed countries with more mature e-waste management and recycling systems. E-waste recycling rates have been constrained a high raw material supply risk derived from the low e-waste inflows into the e-waste recycling chain.

While the drive towards reducing the e-waste reporting to land fill is high the extent to which material volumes within the e-waste value and supply chains is not well accounted for and has presented a challenge when the need to plan for capital investment related to managing the resource arises.

Factors contributing towards the reported low e-waste volumes in circulation were alluded to. Researchers have agreed that a holistic model that has strong representation from all stakeholders, backed by well policed and monitored regulatory measures will contribute to an enhancement in efficient e-waste management. Bearing in mind the need for a strong interplay of key elements discussed previously, the absence of a potentially profitable venture is meaningless.

In chapters 7 and 8 a detailed investigation of the economic feasibility of a hydrometallurgical recycling process designed for copper and gold recovery to be operated in the context of South Africa, was undertaken.

Prior to this study a discussion of the underlying process and economic considerations used in the study is discussed in chapters 4 and 5. The proposed processing capacity was pegged at WPCB recycling rate of 400 tonnes per annum of WPCBs. This implies that 100% of the estimated annual generation of e-waste with WPCBs in South Africa will be absorbed. The development of potential business models around the proposed hydrometallurgical process was discussed in chapter 6. These business models were aligned to the confines of the operating environment and analysed qualitatively and quantitatively.

Chapter 4:   PROCESS DESCRIPTION AND DESIGN CONSIDERATIONS

4.1.   Process description

Four major technologies are available for recycling e-waste and can be used in isolation or in combination with each other. Hydrometallurgy has been identified to be a technology that is scalable, associated with high process kinetics offering a degree of selectivity that made it an attractive technology to explore for the recovery of metals from pre-processed e-waste as a value adding activity (Pyper & Hendrix, 1981; Monhemius, 1987; Yannapoulos, 1991; La-Brooy *et al.,* 1994; Wu *et al.*, 2008; Kamberovič *et al.,* 2011; Jing-Ying *et al.,* 2012; Gurung *et al.,* 2013; Rossouw, 2015; Birloaga, *et al.*, 2016).

A hydrometallurgical process to selectively recover copper and gold from end of life printed circuit boards (WPCBs) rated initially at a recycling capacity of 30 000 tonnes per annum of WPCBs was designed following previous empirical work (Rossouw, 2015; de Waal, 2018). The initial design capacity allowed the researchers an opportunity to compare the economic performance of a hydrometallurgical process against that of a pyrometallurgical process rated at a similar production capacity developed by Ghodrat *et al.,* 2016.

It was established in the current study that the operation at a name plate capacity of 30 000 tonnes per annum of WPCBs would require a consistent supply ranging between 590 000 and 1 770 000 tonnes of unprocessed e-waste based on the proportion of WPCBs in e-waste described by researchers (Widmer *et al.*, 2005; Schluep *et al.*, 2009). A comparison of operations at an established facility, Umicore running at a name plate capacity of 250 000 tonnes per annum of varied secondary metal sources, which includes WPCBs and high value electronics (Hagelüken, 2006), revealed that the success of the proposed hydrometallurgical venture would be affected by the extent to which the capability of logistics function of Umicore could be replicated and even exceeded in South Africa.

A due diligence of different operating capacities was therefore undertaken in the study in light of the existent raw material supply constraint. As evidenced from the analysis, operations in excess of 1 000 tonnes of WPCBs per annum would be burdened by steep reverse logistics costs associated with circumventing the existent raw material supply risk (see appendix E, Table E.1). The consistent supply of high grade WPCBs was noted to be limited and is expected to deteriorate with the active interest in replacing the use of precious metals in electronics manufacture with cheaper alloys that have similar performance characteristics.

It was envisaged that the WPCBs would be recovered from multiple electrical and electronic components. An analysis of the activity and capability of existent recyclers upstream of the South African e-waste recycling chain revealed that an installation with a processing capacity of up to 500 tonnes per annum could be supported. There were indications that a reliable supply of the key raw material could be supplied by the existent recyclers. Using a pessimistic approach the calculations based on inferences made by Widmer *et al.,* 2005, revealed that operating at a name plate capacity of 400 tonnes per annum for example, would require sourcing of approximately 23 530 tonnes of unprocessed e-waste.

The plant was designed to recycle WPCBs at a rate of 400 tonnes per annum. The quality of the feed material was varied so that the plant would process all three grades of WPCBs: low, medium and high grade. The mix of WPCBs reflected the most realistic representation of what is prevalent in the South African e-waste recycling economy. For the scope of this study categories two, three and four of e-waste, as defined by The European Parliament and the council of the European union, were considered for the collection and dismantling section of the operations (see Table 3.1 section 3.1).

Due to the large variation in the age and material specification of the WPCBs composition an assay detailed in literature (Shuey & Taylor, 2004) was used in previous studies for the empirical test work and development of the process flow sheets used in the current studies (Rossouw, 2015; de Waal, 2018). Five major operations were associated with the proposed venture that defines the first business model and these include: collection, dismantling and segregation, size reduction, solder leaching, copper extraction and gold extraction.

Figure 4.1: The generic flow sheet for the WPCB recycling process

Detailed flow sheets showing the equipment for all the processes are shown in appendix D. Figure 4:1 outlines the main operations associated with the proposed venture.

For the study, the collection of e-waste was assumed to be achieved through a dispersed system relying on remittance from all nine provinces in South Africa. It was also assumed that the collection model was dependent on material that was voluntarily surrendered at designated collection points already in existence. Published e-waste recycling rates in South Africa have revealed the need to boost collection volumes through a financed incentive scheme that will support buy back of obsolete equipment and strengthen the engagement of the informal sector.

The collected e-waste was transferred to the central processing plant located in the central commercial hub of the South African e-waste industry namely Gauteng. It was assumed that this was achieved by using a dedicated fleet of vehicles that include the company's own and hired vehicles for the periodic collection of e-waste. The collected material was stored onsite prior to dismantling. Bulk storage of the e-waste onsite was done with the assumption that relevant legislation affecting bulk storage of hazardous material was amended.

The dismantled fractions were directed to a transitory warehouse facility for dispatch to other recyclers in the metals and plastics recycling business. The recovered WPCBs were directed to the pre-treatment plant designed to process at least 1.28 tonnes/day of WPCBs. Pre-treatment entailed dry crushing and screening. Based on the designed process, 45% of the material was redirected to the hammer crusher as oversized material. The underflow reported to the ball mill where the material underwent dry milling. The material exiting the ball mill was channelled by means of a screw conveyer into a buffer storage hopper.

An extractor screw conveyed the milled WPCBs to the buffer feed hopper located in the solder leaching plant. Copper was extracted under oxidative conditions using sulphuric acid and hydrogen peroxide. The pregnant leach solution (PLS) was pumped to the pregnant leach solution holding tank.

The PLS was transferred after 24 hour cycle times to the solvent extraction circuit and electrowinning plant. Copper was recovered as copper cathode plates of LME Grade A quality. The solid residue from the copper leaching circuit underwent cyanide leaching after stripping the remaining base metals. The leached material was then filtered and the PLS directed to the carbon in pulp (CIP) adsorption circuit. The pregnant carbon was then washed with hydrochloric acid before undergoing elution to recover the gold in solution. The basic gold solution was then channelled to the electrowinning circuit where gold was recovered in plated form as sludge.

The mass balances applicable to the copper extraction circuit are specified in appendix A, Table A.3. The solvent and aqueous phase recovered rates were assumed to approximate steady state conditions.

The mass balances applicable to the gold extraction circuit at steady state are specified in Appendix B. The steady state conditions factor in the carbon losses across the circuit, gold losses to tailings derived from carbon losses and barren solution from the adsorption section and the carryover of base metals to the carbon adsorption circuit after filtration. Make up of carbon inventory is factored in at steady state operations.

4.2. Process and design considerations

4.2.1. Collection and dismantling

Collection and dismantling as a separate operation in the business was considered in the study. Literature has reported that the logistical and handling cost components associated with the collection and dismantling of e-waste function are high (Lydall *et al.*, 2017; GreenCape, 2018). During the development of the business models the viability of housing a collection and dismantling business unit was assessed. The detailed assessment was covered in chapter 6. The collection and dismantling of e-waste to recover WPCBs was not included in the scope of the operations of the group 3 and 4 business models.

4.2.2. Shredding and comminution

The mechanical treatment of WPCBs can be done in either dry or wet conditions. The dry shredding and comminution of WPCBs in preparation for subsequent processing steps has however been associated with a notable degree of co-separation of precious metals. This co-separation is increasingly marked at higher degrees of particle size reduction and the resultant metal liberation from the main board matrix.

The high returns associated with recycling high grade WPCBs deteriorates with the use of mechanical methods as a result of higher losses of valuable metals due to the co-separation of precious metals (Goosey & Kellner, 2002; Cui, 2005; Hagelüken, 2006; Kogan, 2006; Hanafi *et al.,* 2012; Khanna *et al.,* 2014; Ghosh *et al.,* 2015). Hagelüken (2006) reported precious metals losses of up to 20%.

For the process under consideration dry shredding and comminution of the WPCBs was selected. It was assumed that the WPCBS were shredded and crushed to a size < 2mm in diameter. The specifications for the size reduction are outlined in Table 4.1 below.

Table 4.1: Operating specifications for the crushing and comminution of WPCBs (Rossouw, 2015)

Main operation	Equipment	Size specification
Crude Crushing	Hammer Mill	1mm
Screen	15mm	20mm
Pulverisation	Ball Mill	<2mm

In the previous studies size classification and removal of the 2mm particles was done intermittently to avoid the generation of fines.

The final product size distribution of shredding and crushing is influenced by the volume feed rates, residence time in the circuit, material composition and moisture. The detailed size distribution for the crushed WPCBs was not provided (Rossouw, 2015).

For the purpose of this study the material size distribution was based on previous work done by He *et al.*, (2015) as detailed in Table 4.2. Based on these results it was assumed that 7.16% of the material was recovered by the dust recovery mechanisms inbuilt into the purchased equipment. Localised temperature elevations occur during the dry shredding of WPCBs, which releases fumes of infused flame retardants (Zhao *et al.,* 2006; Li *et al.,* 2010).

It was assumed that comminution fines would be recovered within the process by use of dust socks built into the purchased equipment while the fugitive dust emissions and the toxic fumes generated were eliminated by the use of fume extractors. Outlined in Figure 4.2 is a schematic representation of the size reduction operation entailing shredding and comminution.

Size reduction (Shredding and comminution)
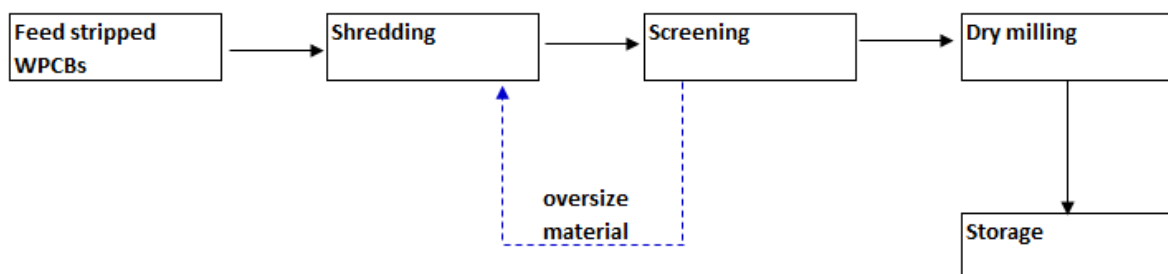
Figure 4.2: Shredding and comminution circuit

Owing to the variability in the composition and physical size of WPCBs no single density can be assigned to all WPCBs. For the study categories 2 to 4 of e-waste formed the source of the WPCBs used in the process.

All grades of WPCBs were included. For the mass and energy balances the WPCB analysis provided in literature was used to calculate an un-compacted bulk density for the milled WPCBs (Shuey & Taylor, 2004). For the study a density of 1 718kg/m^3 was used. It was assumed that the unmilled WPCBs had a voidage of 30% when stored in an un-compacted state so that the bulk density of the unmilled WPCBs was estimated to be 1 203kg/m^3.

4.2.3. Solder leaching

The objective of this step was to remove the lead and tin prior to the extraction of copper. Elevated concentrations of lead and tin are undesirable in the copper and gold recovery circuits. Based on previous studies it was established that lead, tin, nickel, zinc, iron, and small amounts of copper, aluminium and silver were leached under the conditions specified (Rossouw, 2015; de Waal, 2018).

Table 4.2 summarises the proportions of materials that were attacked by the acid based on previous empirical studies (Rossouw, 2015). The data from these studies was used for determining the mass and energy balances and predicting the reactions that were expected during the solder leaching. The ceramics and plastic fractions were not attacked by the acid.

Table 4.2: Comparative profiles based on empirical work and up-rated mass balances of material attacked during the solder leaching step (Rossouw, 2015; de Waal, 2018)

Item	Material	Percentage that reacted (%) Empirical (Rossouw, 2015)	Percentage reacted based on mass balances (de Waal, 2018)
1	Iron	83.80	76.20
2	Copper	0.20	0.19
3	Aluminium	21.40	15.30
4	Tin	15.10	6.99
5	Nickel	43.00	33.60
6	Zinc	2.30	1.69
7	Lead	84.30	97.90
8	Manganese	N/D	N/D
9	Antimony	N/D	N/D
10	Gold	0.00	0.00
11	Silver	2.10	1.31
12	Ceramics (SiO_2, Al_2O_3, alkali and alkaline earth oxides, mica)	0.00	0.00
13	Plastics (Polyethylene, epoxies, PVC, PTFE, polypropylene, polyesters)	0.00	0.00

The hydrogen produced was estimated to be in concentrations low enough to vent into the atmosphere. Two options were incorporated in this process to handle the oxides of nitrogen generated. The options include direct venting to the atmosphere and scrubbing. The first option which entails direct venting to the atmosphere formed the basis of the original mass balances. It was envisaged that the concentration of the oxides of nitrogen vented would be low and the diluting effect of the background atmosphere would contribute towards a low emission level (de Waal, 2018).

Following a review of the process flow sheets a second option for handling the fumes was considered. The second option allowed for the recycling of the fumes to generate weak nitric acid at concentrations not exceeding 10%. The first option for handling the fumes was considered for this study. Figure 4.3 illustrates the main solder leaching process.

Solder leaching
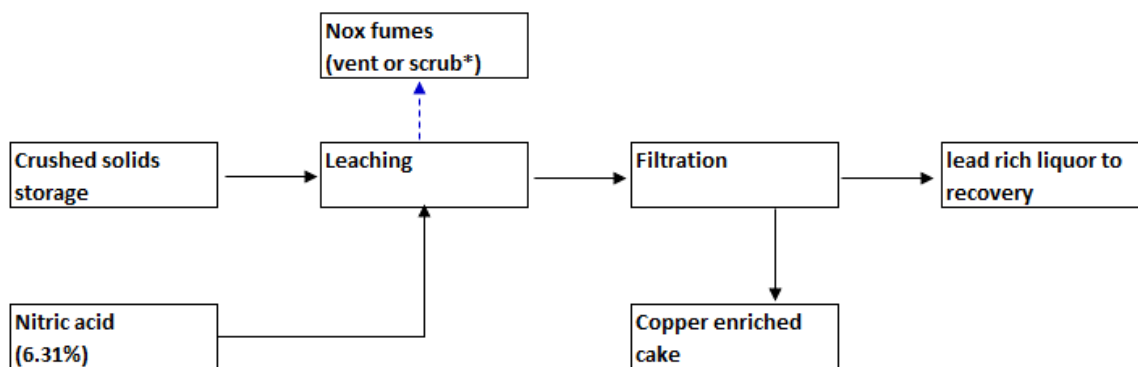
Figure 4.3: The solder leaching process

4.2.4.   Copper extraction

4.2.4.1.   Leaching

The objective of the copper extraction plant was to produce copper of LME Grade A standard through electrowinning of the PLS derived from leaching of the WPCBs. This was achieved by an oxidative leaching step using sulphuric acid and hydrogen peroxide based on the results of leaching and optimisation tests done (Rossouw, 2015). Combining the findings of the test work done it was determined that the solid materials in the feed would react with the sulphuric acid in the proportions listed in Table 4.3.

Table 4.3: Comparative profiles based on empirical work and up-rated mass balances of material attacked during the copper leaching step (Rossouw, 2015; de Waal, 2018)

Item	Material	Percentage that reacted (%) Empirical (Rossouw, 2015)	Percentage reacted based on mass balances (de Waal, 2018)
1	Iron	84.10	84.10
2	Copper	96.46	96.50
3	Aluminium	26.14	18.40
4	Tin	5.83	0.80
5	Nickel	72.01	57.80
6	Zinc	92.20	93.40
7	Lead	0.00	0.00
8	Manganese	N/D	N/D
9	Antimony	N/D	N/D
10	Gold	0.00	0.00
11	Silver	0.00	8.50
12	Ceramics (SiO$_2$, Al$_2$O$_3$, alkali and alkaline earth oxides, mica)	0.00	0.00
13	Plastics (PE, epoxies, PVC, PTFE, PP, polyesters)	0.00	0.00

Polishing of the acidic liquor was achieved in the solvent extraction circuit prior to electrowinning. Empirical data for designing the solvent extraction and electrolysis circuits was not available. Instead, using heuristics from literature combined with standard industrial practice, mass and energy balances around these two circuits were performed, which have been discussed in greater depth in the sections to follow. Figure 4.4 depicts the main unit operations associated with the recovery of copper from WPCBs. For this study it was assumed that no leaching aids were used and would not be necessary.

Copper extraction
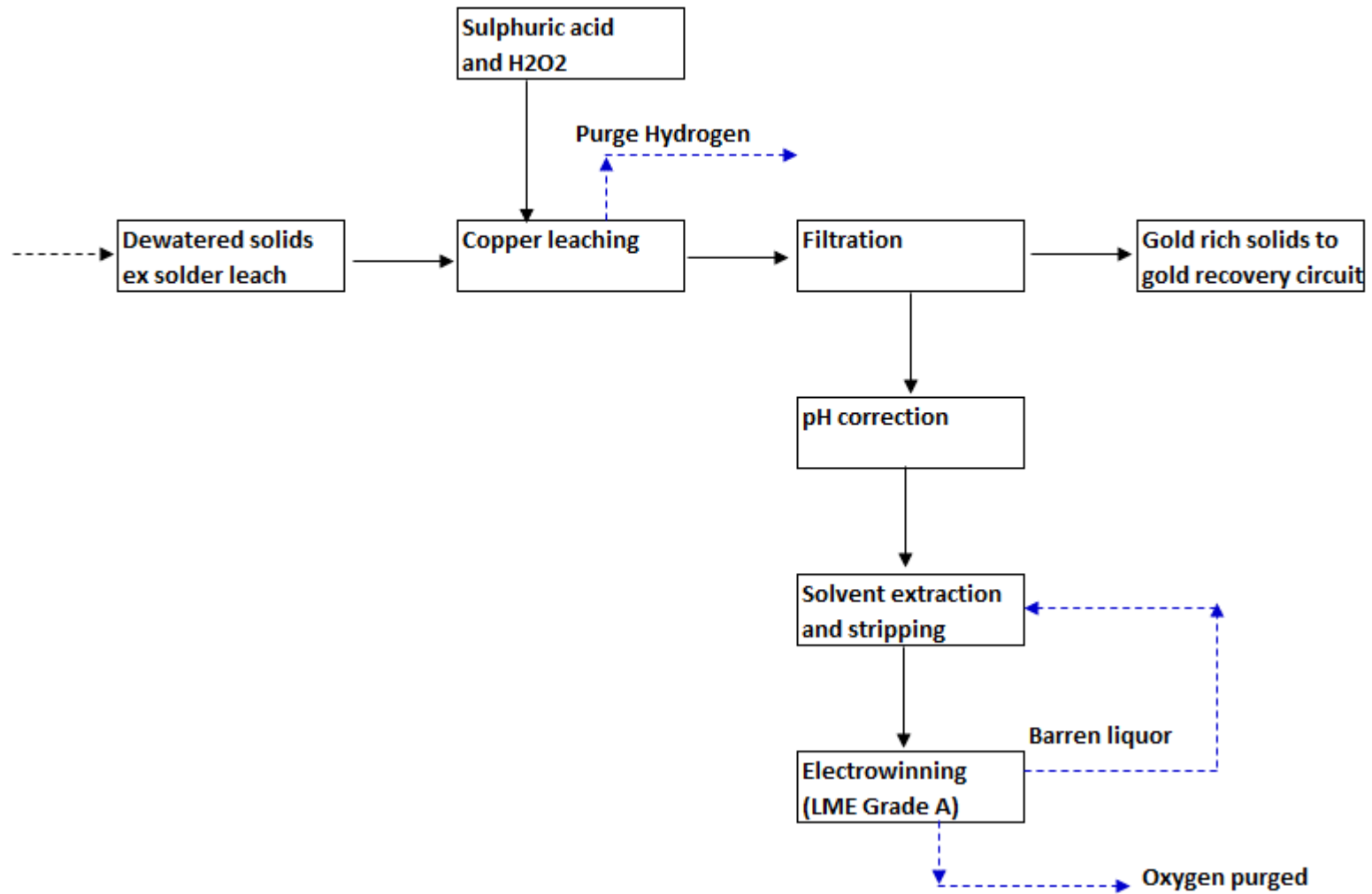
Figure 4.4: The copper recovery circuit

4.2.4.2.   Solvent extraction

Equipment

Contactors can generally be categorised as stage wise or differential. Stage wise contactors are easier to scale up or down. For the process that was developed no empirical studies on a laboratory and plant scale supported the choice of contactor. Industrial experience and literature studies (Ritcey & Ashbrook, 1979) were instead used to support the decision to use the mixer settlers. Mixer settlers were selected as the contactors to be used (de Waal, 2018).

Costs of solvent extraction

The operating costs associated with solvent extraction may be divided into 8 general categories. These costs include: feed liquor preparation, solvent pre-equilibration, scrubbing, stripping, solvent loss and recovery, labour and maintenance costs, energy costs, metal recovery from raffinate streams, chemicals and other consumables. These costs were factored into the economic evaluation of the process under study (Ritcey & Ashbrook, 1979).

Extractants

Three broad categories of extractants exist and include those that form compounds, those involving ion association and those involving solvation of the metal ion. Extractants are expected to be inexpensive, relatively insoluble in the aqueous phase and possess a notable degree of stability after repeated use in the process (Ritcey & Ashbrook, 1979). The extractant must not be prone to the formation of stable emulsions and crud. It is also desirable that the extractant possess good coalescing properties to minimise phase disengagement time. High metal loadings and ease of stripping are also desirable in an extractant (Ritcey & Ashbrook, 1979).

The most widely used extractants in the copper processing industry include aldoximes, oximes and blends of modified aldoximes and ketoximes. Examples of these extractants include ACORGA OPT5510, ACORGA OPT5540, ACORGA M5774, the Shell chemicals (SME series), Kelex series and LIX 984N (Ritcey & Ashbrook, 1979; Maes *et al.,* 2003; Mwale 2009; BASF, 2015; Sole & Tinkler, 2016). These extractants are chelating extractants that fall into the category of the compound forming extractants.

The first LIX extractants were developed to extract copper from copper dump leach solutions. Most LIX extractants can be used without the use of a modifier (Sole *et al.,* 2007). Typical modifiers used in commercial extractants include tridecanol and nonylphenol. Pilot studies were conducted with LIX 860. It was shown revealed that the rate of crud build up using modified extractants was double the rate associated with the use of an unmodified extractant (LIX 680).

This confers an advantage in using LIX extractants over the other extractants containing modifiers or requiring modifiers during use. Mwale (2009) compared the performance of LIX 984NC and ACORGA M5774 and noted that LIX 984N offered the advantage of low copper losses to tailings.

For the process under study LIX 984N was selected as the most appropriate extractant. LIX 984N is comprised of an equimolar blend of LIX 860N-I and LIX84-I. This aldoxime and oxime blend displays superior Cu^{2+} extraction rates when compared to the performance of ketoximes. The extractant requires no addition of a modifier in the process (Ritcey & Ashbrook, 1979; Bacon & Mihaylov, 2002; BASF, 2015).

Typical industrial practice recommends an extractant concentration of between 10% and 40% (Kuipa & Hughes, 2002). A concentration of 32% LIX 984N diluted in kerosene was specified in the process design (de Waal, 2018). An organic to acid ratio of 3.53 was used. Using the specified O/A ratio in the extraction stage 80% copper extraction was assumed that resulted in a copper loading of 9.6g/L. The specified process configuration for the extraction-washing-stripping based on typical industrial practice was 3-1-2 (de Waal, 2018).

The wash stage was included to minimise aqueous entrainment into the loaded organic phase (Ritcey & Ashbrook, 1979; Maki, 1999). The wash stage also served to scrub co-extracted and entrained inorganic contaminants. It was assumed that the wash stage was run in the organic continuous profile. Clean water acidified with sulphuric acid was used for the process. It was assumed that the wash stage was operated at the same capacity as the stripping stage with an O/A ratio of 1.56. It was further assumed that the aqueous phase entrained during the extraction stage was captured in the wash stage and no solvent was entrained in the effluent wash water.

Co – extraction of non - target metals

Metals exhibiting chemical similarity display similar extraction properties at differing or matching rates. The co-extraction of other metals in the feed contributes to an alteration of the chemistry of the solvent phase. Manganese is deleterious to the extractant as will be discussed in later sections. Iron is also co-extracted with copper. The use of a highly selective extractant for this study was assumed to reduce the tenor of iron reporting to the electrowinning circuit.

Cobalt and nickel exhibit chemical similarity to copper and will be co-extracted. Sridhar *et al.*, 2009 investigated the extraction of copper and nickel using LIX 984N. Findings of the research indicated that the selective uptake of copper and nickel is achievable by varying the pH of operation. It was shown that nickel was more likely to be extracted in an alkaline media for the conditions that were studied (Alguacil & Cobo, 1999; Sridhar *et al.*, 2009; Kumar *et al.,* 2013). A comparison of the extraction rates at equimolar concentrations revealed however, that copper extraction was still superior, attaining equilibrium at a faster rate than nickel (Sridhar *et al.*, 2009).

The concentration of the metals present in a solution also acts as a driving force to towards the rate of attainment of equilibrium. In less than 5 minutes of extraction, all the copper was extracted and achieved equilibrium, while only 61% of the nickel extracted attained equilibrium (Sridhar *et al.,* 2009). While the system applies to alkaline conditions the results do give a possible indicator of the strong influence of the concentration of the species and relative stabilities in the specified pH of the medium.

For the process under consideration the ratio of Cu to Ni was found to be 25:1 which was 2.5 times greater when compared to the work of previous researchers (Sridhar *et al.,* 2009). Sridhar *et al.,* (2009) observed that the co-extraction of nickel occurred at a pH of 2. It would thus be expected that in a contact time of less than 60 seconds a negligible amount of nickel would be co-extracted. No empirical data however, was available to support this prediction for the process under study. It therefore was assumed that all the nickel was co-extracted together with cobalt.

Within the scope of this study it was also assumed that in a case where aqueous entrainment (when running in an organic continuous profile) into the organic phase occurred, it was expected that aluminium, zinc, tin, lead, manganese, silver, silica and alumina would report to this phase.

Based on the performance data of LIX 984N for the operating parameters specified and the mass balances calculated, it was established that the maximum iron content to be co-extracted would amount to 0.009mol/hour. For the final processing of the pregnant leach solution no iron was present.

Extractant losses

Solubility data on the LIX and Kelex extractants indicate that these materials are poorly soluble in aqueous media. In copper extraction systems running at pH values in the neighbourhood of 1.5, the reported extractant losses have been noted to range between 15ppm and 140ppm. At levels as high as 140ppm, solvent losses represent a major cost of the solvent extraction process. These losses include both soluble and entrainment losses as determined from inventory calculations (Ritcey & Ashbrook, 1979). Solvent entrainment losses for the LIX systems are estimated to be higher than those of the Kelex systems because of the higher organic/acid (O/A) ratio required but at half the cost per unit mass of reagent required (Ritcey & Ashbrook, 1979).

A review of the LIX 984N solubility data on the material data sheet (BASF, 2015) indicates that this extractant forms insoluble copper complexes in the aqueous phase. Losses for the process under study were deemed not to arise from organic solubility in the aqueous phase.

Entrainment

Solvent entrained in the aqueous raffinate, in the strip liquor or in cruds is the most significant contributor to organic solvent losses. Organic losses to the raffinate have been reported to range between 7ppm and 125ppm (Sole *et al.,* 2013).

Entrainment losses can be attributed to a number of factors. The factors include a combination of one or more of the following: insufficient settling area or inadequate settling time for phase disengagement, poorly designed or operated mixers, a high energy input during the mixing stage, lack of additives to suppress emulsion formation, poor diluent choice, high extractant concentration in the solvent and solids in the aqueous feed.

During the design and revision of the process flow sheets it was assumed that the operating conditions and the selected reagent would not contribute towards solvent losses due to inadequate settling time.

Any losses of the copper complex and solvent would be a result of entrainment of the organic phase in an aqueous continuous system. It was also assumed that with the smaller operating scale and ease of monitoring a smaller installation in comparison to larger facilities the losses of solvent were maintained at an average of 15ppm.

Volatilization

Volatilization of solvent components can be a problem when the system is run at elevated temperatures or in hot climates. The operating conditions specified during the development of the process were below the flash points of the constituents of the solvent and the diluent (de Waal, 2018). Operating in regions close to the flash point of the organic constituents increases the risk of fire. In this study minimal volatilization of the diluent was assumed to occur for the specified conditions of operation, therefore significant losses of diluent were not factored into the economic study.

Reagent stability and degradation

Sufficient data was unavailable to account for the stability of various extractants, modifiers and diluents being used in the solvent extraction process. The degradation of solvent components can arise from various sources. These sources include the oxidising properties of the aqueous phase, high temperatures in the system, aging and oxidation by atmospheric exposure and general instability of the component over repeated cycles of operation (Miller, 1995).

Studies on LIX reagents have shown a tendency towards degradation resulting from the dehydration that occurs between the proximal oximoto and hydroxyl groups within the extractant molecule (Miller, 1995).

In this study it was assumed that the normal degradation rate associated with the nature of the chemical groups and their influence on each other would be the major contributing factor towards solvent degradation.

There are cases where extractants can interact with the diluent resulting in the formation of species that alter the uptake of the metal ion of choice (Ritcey & Ashbrook, 1979; Kuipa & Hughes, 2002).

Diluents with high dielectric constants have been predicted to show a greater interaction with the aqueous phase. Kuipa & Hughes (2002) noted that the capacity of the extractant would be reduced as a result of the potential uptake of water molecules at the liquid-liquid interface between the two liquids. In this study however, it was assumed that this phenomenon occurred to such a low degree that did not influence the performance of the bulk organic phase.

The formation of other species can lead to the generation of third phases that have the ability to contribute towards the formation of cruds. The temperature dependence of third phases has been observed in commercial copper operations. Large scale commercial copper producers have used this knowledge to manage the formation of this phase when the plant is in operation (Ritcey & Ashbrook, 1979). Further research has shown the tendency of aliphatic diluents to form another organic phase. In the scope of this study however, third phase formation was assumed not to occur.

Modifiers are often added to suppress the formation of third phases. The modifier typically constitutes between 2% and 5% of the organic solvent. Carboxylic acid-kerosene and chelating extractant-diluent mixtures do not usually require the addition of a modifier. As with diluents, the extractive properties of a solvent can also be affected by modifiers (Ritcey & Ashbrook, 1979). For the process under study it was assumed that no interaction between kerosene and LIX984N occurred. As detailed earlier with the use of LIX 984N, no modifier was necessary.

Crud formation

Literature reports that crud formation is common to all or most solvent extraction operations in the metals extraction processes with an average of 50% organic in its composition (Bishop *et al.,* 1999; Hartmann & Corbella, 2007).

Solids in the feed are precursors to the formation of crud that act as emulsion stabilizers during the mixing stage. Practical experience however has shown that LIX systems are prone to crud formation. The formation of stable emulsions leads to the formation of cruds. The crud can constitute a significant solvent loss negatively affecting the process economics of the circuit. Several factors have been noted to contribute towards the formation of crud. These factors include: fungal and bacterial growth, degradation products of the solvent, the presence of fine solids, by-products associated with the extractant manufacture, components of the main raw material entering the process, chemical interactions between constituents of solvent phase, high shearing rates during mixing and poor phase disengagement times. The types of crud formed are unique to a process and vary in composition and structure.

The nature of the raw material feed and its handling upstream of the process plays a part in the formation of crud. The presence and concentration of certain cations such as iron (Fe^{3+}), silica (SiO_2), calcium (Ca^{2+}), magnesium (Mg^{2+}), manganese (Mn^{7+}), aluminium (Al^{3+}), phosphorus (P) and sulphate (SO_4^{2-}) can, with sufficient shearing effects during mixing, yield cruds (Ritcey & Ashbrook, 1979, Sole *et al.,* 2018).

Solids present from suspended material form sites that stabilize the emulsion as they exist in the aqueous phase as a dispersed colloid. Solid matter that includes jarosites, dissolved and suspended silica, quartz, micas and gypsum are well known contributors towards the formation of crud in copper solvent extraction systems (Ritcey & Ashbrook, 1979; Readett & Miller, 1995; Sole *et al.,* 2018).

As particles increase in the degree of hydrophobicity, the tendency towards crud formation follows a similar pattern. Normal operations therefore include filtration in the operation to reduce the solids concentration in the rich liquor to 10ppm (Ritcey & Ashrook, 1979). Dissolved silica is well known for polymerizing in aging solutions. The polymerized silica contributes towards elevating the solution viscosity and impedes the transfer of ions across the organic and aqueous interface (Sole & Tinkler, 2016).

Solids responsible for the formation of crud from WPCB feed include, silica and mica fines. For the purposes of this study it was assumed that filtration was able to achieve a solids concentration of up to 10ppm. It was also assumed that silica originally present in the WPCBs was not attacked by acid. No silica gels were formed for the process under consideration. Organic matter, humic acids, dust and low ionic strength, were also considered inapplicable factors contributing to crud formation in this study.

Unreacted chemicals from the manufacture of the extractant or impurities from the containers also contribute to the formation of crud. Their presence can either promote or depress phase separation and enhance or suppress the uptake of the targeted metal (Ritcey & Ashbrook, 1979). The specific effects are usually verified through bench scale and pilot studies. During the development of the process no empirical data was available to consider this factor. It was therefore assumed for the mass balances done (de Waal, 2018) and the scope of this study that the influence of unreacted species was not applicable. Crud formation for the process under study was attributed to the presence of fine solids of SiO_2 and mica, fungal growth and aging of the solutions.

Managing crud formation and solvent recovery

No universal treatment scheme exists that broadly covers the recovery of solvent for all solvent extraction processes. This is attributed to the uniqueness of the process and crud that is formed.

Several options however do exist for breaking down crud and the selected option is also subject to experience in the plants. The options include, settling and skimming, filtration, acidification, neutralization to pH values close to 7 and intense agitation (Ritcey & Ashbrook, 1979).

As the feed grade changes the frequency and nature of the crud formed will also respond in a similar manner. Sole & Tinkler (2016) noted that crud formation can result in the filling of the settler within a one and half to a twelve month period of operation.

Copper recovery circuits using LIX solvents were earlier noted to be notorious for crud formation (Ritcey & Ashbrook, 1979). Test work and full operation of a centrifuge have proved successful in systems with a high degree of fines. In the Bwana Mkubwa solvent extraction plant, that uses LIX 984N, the centrifuge was reported to enable the successful and consistent recovery of at least 95% of the solvent (Mukutuma *et al.,* 2007). It was noted that the centrifuge is capable of handling three phases during operation. A fourth phase was however observed due to the presence of ultrafine solids that stabilised the emulsion. The researchers noted that further work was required to improve the design to manage the fourth phase that formed.

Studies examining the rate of crud formation in a copper pilot plant using a mixer settler were undertaken by Aminian & Bazin (1999). The researchers were able to derive a correlation based on time and thickness of crud formed in the LIX 984N extraction system. The expression, detailed in 4.1 below, revealed the existence of a non linear relationship between the crud formation rate and time.

$$h(t) = 21.39(1- e^{-0.004t}) \tag{4.1}$$

The time 't' is expressed in hours and the thickness 'h' is expressed in centimetres.

Results of the study also revealed that the rate of increase in crud thickness declined with time. The crud was shown to be associated with Al_2O_3, gypsum, silica, and Fe_2O_3. A comparison of the liquor compositions for the process under study and the studies of Aminian & Bazin (1999) are summarized in the Table 4.4:

Table 4.4: Stream analysis for the prediction of crud generation rates in LIX 984N extraction system

Aminian & Bazin (1999)		(Rossouw, 2015; de Waal, 2018)		Comparative proportion
Element	Concentration (g/L)	Element	Concentration (g/L)	
Cu	1.62	Cu	1.41	0.87
Fe	15.60	Fe	2.88	0.19
Al	7.69	Al	0.70	0.09
Mg	2.93	Mg	0.00	0.00
Si	758.50	Si	0.00	0.00
Mean proportion				**0.07**

An analysis of the data revealed that the crud formation rates for the system under study were predicted to be lower in comparison to findings in literature (Aminian & Bazin, 1999).

80

The predicted crud formation rate, using a comparative analysis of the results estimated a rate which was only 7% of the rate predicted by the researchers (Aminian & Bazin, 1999).

In this the study the rate of crud formation was therefore assumed to follow the model derived by Aminian & Bazin, 1999. An additional 3% was added to account for the effects of fungal and bacterial growth, typical of LIX systems. It was therefore assumed that 10% of the calculated crud thickness at the end of 24 hours was the thickness of the crud formed for the process under study. In commercial mixer settler operations crud accumulates towards the discharge end of the settler. The crud in this study was assumed to form in the settler area which was removed at the discharge end of the settler. It was assumed that the settler area occupied 85% of the total surface area available in the mixer-settler. The crud formation data used for this study is summarized in Table 4.5.

Table 4.5: Crud formation data for the process under study

Item	Detail	Data
1	Mixer-settler units	3
2	Available mixer settler area	$7.50m^2$
3	Settler area of crud	$6.40m^2$
4	Operation time	24hours
5	Calculated crud thickness (h_1) at t=24 hrs	1.96cm
6	Crud thickness for the process under study = $0.1h_1$	0.20cm
7	Crud composition (Bishop *et al.,* 1999; Hartmann & Corbella; 2007).	50% v/v organic
8	Crud density	$994.72kg/m^3$
9	Crud volume formed per day	$0.04m^3$
10	Crud mass formed per day	37.34kg
11	Solvent recovery (calculated)	97.25%
12	Solvent recovered	15.41kg/hr
13	Solvent to effluent clay treatment	0.44kg/hr

It was assumed that the rate and nature of the crud formed was fairly consistent since an averaged feed grade of WPCBs was considered for the study. The settlers and skimmers for crud removal and management were considered to have been incorporated in the design of the purchased equipment. Based on the commercial success of the centrifuge reported by Mukutuma *et al.,* (2007) a centrifuge was also included in the main equipment schedule to provide for continuous removal of crud and solvent recovery. The formation of a fourth phase for the process under study was assumed not to occur. Incorporating the use of a centrifuge and clay treatment, it was assumed that at least 97% of the solvent would be recovered and returned to the main solvent extraction circuit.

The aqueous phase recovered from the centrifuge was directed to further clay treatment discussed in section 4.3.2, which was redirected to the leach circuit without organic. It was also assumed that the solids exiting the centrifuge were discharged to the solid waste tailings for specialised disposal.

4.2.4.3.   Electrowinning

Power usage

Over-potential describes the energy required to drive the cationic reduction reaction forward. Diffusion limiting current (DLC) density describes a limiting current beyond which any further increase in the operating current will not enhance the rate of copper deposition. Other factors come into play such as the copper ion concentration at the cathode surface, viscosity of the system, concentration of competing species, and the influence of operating temperatures. An expression for determining the DLC density has been derived that relates the copper ion concentration, Faraday constant and the moles of the target ion to the current:

$$DLC \ = \ i_L \ = n.F.kd \ [Cu^{2+}]_{bulk} \tag{4.2}$$

This value is also validated through empirical study. The DLC density is useful for making plots of voltamograms for optimising the design and operation of the electrowinning cell (Beukes & Badenhorst, 2009). Large copper electrowinning plants have DLC densities ranging between 193 and 285A/m^2. For 22 plants studied the maximum recorded DLC density was reported to be 321A/m^2 (Beukes & Badenhorst, 2009).

Typical industrial practice recommends maintaining the current density in excess of 250A/m^2 to achieve LME Grade A copper cathodes (Ritcey & Ashbrook, 1979; Beukes & Badenhorst, 2009). In the initial development of the process flow sheet, a current density of 300A/m^2 was specified which was assumed to be above the DLC density for the process under study (de Waal, 2018).

Anodes

Lead based alloys are used extensively in copper electrowinning operations. The use of lead based alloys is however associated with the passivation of the anode and migration of lead into the electrolyte solution (Mirza *et al.,* 2016). The corrosion of lead anodes during copper electrowinning is discussed in literature in which weakly adherent and adherent forms of PbO are formed (Mirza *et al.,* 2016). This results in final product contamination (Prengaman & Siegmund, 1999; Mirza *et al.,* 2016). Periodic cleaning of the anodes in use provides for the removal of weakly adherent β-PbO. For this study, a lead-calcium-tin alloy was specified for use as the anodes (de Waal, 2018).

Typical methods for managing lead anode passivation and contamination entail the addition of cobalt and the practice of periodic current reversal. The addition of cobalt is discussed in section 4.2.4.4. Periodic current reversal is often practiced to manage passivation and is done for short periods of time. The practice is well suited to continuous processes (Beukes & Badenhorst, 2009).

Prengaman & Siegmund (1999) in an alternative vein discussed the transient effects of power losses in copper electrowinning circuits and the subsequent reversal of the anodic and cathodic reactions. The researchers noted that an insulating layer of lead II sulphate ($PbSO_4$) would be formed that has a passivating effect on the anodes.

In this study it was assumed that periodic current reversal was practiced and was factored into the calculated energy consumption and associated costs. It was also assumed that the practice of periodic current reversal would not allow sufficient time for the generation of deleterious amounts of $PbSO_4$.

Prengaman & Siegmund (1999) highlighted four major effects of oxygen generation at the anode during copper electrowinning. In addition to the corrosion of lead with the formation of oxides of lead and lead sulphate as intermediates, the researchers noted that the generation of localised oxidising conditions is detrimental to the life of organic froth and acid mist suppressants. The oxidising conditions were found to contribute towards the formation of organic radicals. For the current density specified in this study however, it was assumed that the generation of such organic radicals was in negligible quantities that would not compromise the normal useful service life of the anodes.

The typical life span of anodes ranges between 7 and 10 years (soleconsulting, 2013). The use of anode spacers is reported to increase the service life of the anodes by a 20% margin (Manjengwa, 2018b). The cost of cathode spacers was factored into the cost of the electrowinning cells.

Cathodes

More than two different electrowinning technologies are in use globally. They include a combination of well established processes and technologies tailored to the operating plant's requirements (soleconsulting, 2013). Examples of the technologies include Aisco/Falconbridge, Mesco, WENMEC, T.M. Engineers, Outokumpo, Mesco and Xstrata (Armstrong, 1999; Anderson *et al.,* 2009; soleconsulting, 2013). The technology specified for this study was the ISA process. Key features of the ISA process are summarized in Table 4.6.

Table 4.6: Main features of the ISA Process (Anderson *et al.,* 2009; soleconsulting, 2013)

ISA PROCESS
Copper plated 304 stainless steel hanger bar
U" shaped rectilinear cathode washing and stripping machine design
3.25 mm thick 316L stainless steel cathode plate.
The cathode sheets are separate from each plate.
Typical anode dimensions (Depth, width and thickness) 1.171m x 0.95m x 6.4mm

The wax-less ISA PROCESS was specified for the process, which is better known as the ISA PROCESS 2000, in which stainless steel starter sheets are used in a number of commercial copper electrowinning operations (Beukes & Badenhorst, 2009).

For the process under study stainless steel starter sheets (SS316/304) were specified for the cathodes (Anderson *et al.,* 2009; Beukes & Badenhorst, 2009; de Waal, 2018).

Configuration of electrode pairs

The configuration of the anode and cathode pairs affects the current balances hence the overall process efficiencies (Mirza *et al.*, 2016). Bent sheets associated with beaten copper tend to promote short circuiting (Ritcey & Ashbrook, 1979). Deformities and misalignment associated with hanger bar corrosion also contribute to the distortion of the anode and cathode spacing. This leads to the formation of dendrites, short circuiting and uneven deposition of copper on the cathode surfaces, spalling of PbO_2 and reduced service life of the anodes (Mirza *et al.*, 2016).

For the process under study it was assumed that no short circuits were experienced due to physical deformities associated with the cathode starter sheets. It was also assumed that the inter-electrode spacing was maintained at 10cm as practiced in typical commercial copper electrowinning operations (Mirza *et al.,* 2016).

The copper electrowinning and harvesting considerations specified for the study are summarized in Table 4.7 as follows:

Table 4.7: Copper electrowinning and harvesting process design considerations

Item	Area	Data	Reference
1	Plant design	400 Tonnes WPCBs	
2	Operating time per year	312 days	de Waal (2018)
3	Electrowinning technology	ISA PROCESS 2000	Armstrong (1999)
4	Cathode technology	ISA Single sided	Armstrong (1999)
5	Cathode stripping technology	ISA	
6	Method of cathode stripping	Machine, manual	Armstrong (1999)
7	Cathode active area	$1.21 m^2$	Kafumbila (2017)
8	Harvest cycle	7 days	Kafumbila (2017)
9	Harvest machine hours	3-19/day	Kafumbila (2017)
10	Current efficiency	0.9	de Waal (2018)
11	Operating current density	$300 A/m^2$	de Waal (2018)
12	Operating constant (K)	1.19 g/Amp-hour	Kafumbila (2017)
13	Number of cells	1	Calculated Armstrong (1999); Kafumbila (2017)
14	Cathodes per cell	25	Calculated Armstrong (1999), Anderson *et al.,* (2009)
15	Cathode starter sheets	SS316L Thickness 3.25mm Width 1.04m Height 1.29m	soleconsulting (2013), Kafumbila (2017), de Waal (2018)
16	Anodes per cell	26	Calculated Armstrong (1999) Kafumbila (2017) Soleconsulting (2013)
17	Anodes	Pb, Ca, Sn alloy Thickness 6.4 mm Width 0.95m Height 1.17m	soleconsulting (2013) de Waal (2018)

Recovery of pure copper cathodes

The copper cathode harvest cycles can be determined from the final desired cathode weight or a selected time period of harvesting. Kafumbila (2017) described three methods for determining the harvest cycle of the cathodes. For the process under study a time based cycle of harvesting was used. In commercial electrowinning operations cathodes are normally harvested after 5 to 8 days of operation (Armstrong, 1999; Beukes & Badenhorst, 2009). The time period that was considered adequate for the process under study was 7 days.

A mathematical expression for determining the weight of the copper deposited on the cathode plate was described by Kafumbila (2017) summarized in expression:

WOC(kg) = K x A x OCD x η x 10^{-3} x HC x 12   (4.3)

Where WOC is the weight of the one sided copper deposit, K is a constant equal to 1.19, OCD is the operating current density, A is the cathode active area, η is the current efficiency and HC is the harvest cycle.

In typical large scale commercial electrowinning operations 30 to 36 cathodes are loaded into each cell for a production output ranging between 5 and 15 kilo tonnes of copper per annum (Anderson *et al.*, 2009).

For this study the weight of a one sided copper deposit for each cathode was calculated to be 32.4 kg. The total weight of copper deposited onto the stainless steel cathode was therefore 64.8kg which is a reasonable estimate that closely approximates typical industrial performance data (Beukes & Badenhurst, 2009; Kafumbila, 2017). This translated to a total of approximately 25 cathodes. For the study and scale of operations one electrowinning cell was specified with the dimensions summarized in Table 4.8.

Table 4.8: Dimensions for the electrowinning cell

Description	Cell length conversion factor	Industrial reference data (soleconsulting, 2013) Toquepala, Southern Peru Limited, Peru		Dimensions for the current process	
Dimensions of electrowinning cell	0.403	Width	1.47m	Width	1.47m
		Length	6.39m	Length	2.58m
		Depth	1.54m	Depth	1.54m

It was assumed that LME Grade A copper was the final product recovered for sale as cathode sheets. A manually operated cathode stripping machine was specified for use.

Cathode stripping

Cathode stripping can be achieved manually or automatically depending on the scale of operations. The cathode stripping rate can be calculated from a knowledge of the working hours of the machine, the harvest cycle and the number of cathodes per cell house as shown in equation 4.4 (Kafumbila, 2017).

$$Mc = n/(HC \times WHS) \tag{4.4}$$

Where n   = cathodes per cell house

HC   = Harvest cycle

WHS = Working hours of stripping machine

In this study a stripping rate based on a working time of 1.25 hours was assumed. This meant that at least 3 cathodes were stripped and washed each day for the entire electrowinning cycle during idle periods in the copper extraction circuit. Using this cathode stripping rate it was assumed that two sets of cathode starter sheets were purchased as part of the working capital.

Additives

Four groups of additives defined by their purpose are normally added to copper electrowinning circuits. They include the froth and acid mist suppressants, levelling and smoothing agents, coating agents and grain refiners. Other additives specific to the production of copper cathodes include accelerators, inhibitors and carriers.

Cathodes can be coated with anti-deposition agents at the bottom of the cathode or can be left bare. Coating the base of the cathode reduces contamination due to slime at the base of the tank. For the process under study no coating of the cathodes was included in the initial design (de Waal, 2018). The mass balances suggested the unlikelihood of sludge formation that would form to such an extent that would contaminate the cathode product.

Porous, nodular and dendritic deposits are not desirable. Additives are also used to improve the final quality of the cathode and enhance electro-deposition at the same time. The porous cathodes can entrain impurities. Additives typically added in commercial copper electrowinning operations include levelling agents like glue, oligosaccharides for example, DXG-F7 and guar gum (Ritcey & Ashbrook, 1979). The addition of DXG-F7 was factored in the economic study of the process study.

DXG-F7 has a lower preparation time with a reported solubility in water favouring comparatively high dissolution rates in lower water volumes (Cifuentes *et al.,* 2015). Standard industrial operations normally specify a dosage rate of 150g/tonne to 400g/tonne (Beukes & Badenhorst, 2009; Robinson *et al.,* 2013). For the study the lower dosage rate of 150g per tonne was used.

The third group of additives serve as grain refiners and they include thiourea and chloride ions. For the process under study, sodium chloride was added. Industrial practice recommends maintaining the chloride content between 10ppm and 15ppm (Beukes & Badenhorst, 2009). Mirza *et al.,* (2016) reported a higher chloride average ranging between 20ppm and 30ppm.

It has been noted that chloride concentrations greater than 30ppm are detrimental to the service life of the anodes. A high chloride concentration (100ppm >) not only results in aggressive media that is deleterious to the operation of the anodes, the elevated chloride levels also result in friable copper sheets during the stripping of the copper cathodes and corrosive operating conditions that promotes the attack of the stainless steel cathodes. The corrosion of the stainless steel cathodes leads to a build up in chromium, iron and nickel in the electrolyte that leads to process temperature rise, reduced current efficiency and generation of chlorine gas at the anode (Beukes & Badenhorst, 2009). A concentration of 10ppm of chloride was maintained for the process under study.

Cobalt has been found to be beneficial in copper electrowinning circuits due to the role it plays in passivating the anode thereby minimizing lead contamination. This is reported to be well pronounced in acidic media due to the low reduction potential cobalt has (-0.28V) when compared to hydrogen (0.00V) (Mulaudzi & Kotze, 2013). Hydrogen in low pH media competes with the cobalt due to its larger bulk concentration. A concentration ranging between 100ppm and 150ppm suffices for copper electrowinning circuits (Al Shakarji, 2012; Sole & Tinkler, 2016). Operating above an upper limit of 200ppm, leads to a reduction in the cost/benefit ratio (Mirza *et al.,* 2016). Typically if the feed source contains cobalt there is no need to add cobalt into the process.

The WPCB analysis used in the study indicated the absence of cobalt (Shuey & Taylor, 2004). Based on the mass balances specific to the process it was assumed that cobalt sulphate was dosed into the process as a 24.25% m/m solution and factored into the operating costs of the copper electrowinning circuit. The dosage rate is subject to the results of empirical studies of the system. Within the scope of the development of the process under study no empirical work was done to determine the specific usage ratio of cobalt. A lower dosage rate was however selected since the feed did not contain manganese for the original process that was developed (Shuey & Taylor, 2004; de Waal, 2018).

Sole and Tinkler (2016) have reported a cost contribution ranging between US$3.50 to US$4.00 per tonne copper produced. An average cost contribution of US$3.75 from cobalt usage was specified for the process under study. This translated to a usage ratio of 0.493kg cobalt sulphate/ton copper produced. Mirza *et* al., (2016) noted that cobalt sulphate is a substantial contributor towards copper electrowinning costs.

In this study it was assumed that the co-deposition of cobalt with the copper was marginal. It was also assumed that at start up all the cobalt remained in the spent electrolyte. During bleeding 20% of the total cobalt dosed into the system reported to the leaching stage. No accumulation of cobalt was reported for the process under study. It was further assumed that the cobalt derived from the recycle of electrolyte bleed was washed out in the solvent extraction circuit during the loaded solvent washing stage, since cobalt is co-extracted with copper (Sridhar *et al.*, 2009).

Acid mists and the suppressants

Acid mists are generated during the electrowinning process and are aggravated by elevated process temperatures (Ritcey & Ashbrook, 1979; Al Shakarji, 2012). As oxygen is liberated at the anodes, the collapsing film of the bubbles that rise to the electrolyte surface rapture with the entrainment of acid in the gas phase in the form of an aerosol. There are a number of methods in use to manage acid mist and they include adapted engineering design configurations, physical and chemical means.

The methods include the use of mist suppression foams, floating macromolecular macrospheres, open tank house designs, positive flow ventilation and cell top hoods (Anderson *et al.*, 2009; Al Shakarji, 2012). The methods listed can be used in isolation or as combinations of two or more alternatives.

Al Shakarji (2012) examined the mechanisms of and factors influencing acid mist formation. The research findings showed that the use of FC-1100 reduced the formation of acid mist. The performance of FC-1100 was complemented by the use of macrospheres reducing acid mist generation by a 29% margin.

The generation of the acid aerosol is a function of the site elevation, current density, barometric pressure, ambient temperature, anode condition, electrolyte temperature and the composition of the electrolyte. For the process under study it was assumed that acid mists were managed by use of an acid mist suppressant, 3M acid mist suppressant FC-1100. The action of the acid mist suppressant was complemented by using 20 mm polypropylene macrospheres.

The loading rate of the polypropylene spheres was determined using the available surface area of the electrowinning cells as summarized in Table 4.9.

Table 4.9: Data for determining loading rate of polypropylene macrospheres

Item	Description	Data	Reference for calculations
1	Cross sectional area occupied by anodes	$0.16m^2$	soleconsulting (2013) Kafumbila (2017)
2	Cross sectional area occupied by anodes and cathodes	$0.09m^2$	soleconsulting (2013) Kafumbila (2017)
3	Cross sectional area occupied by the electrodes	$0.24m^2$	soleconsulting (2013) Kafumbila (2017)
4	Diameter of polypropylene macrosphere	20mm	Selected, Kafumbila (2017)
5	Maximum area a polypropylene macrosphere can occupy	$0.0004m^2$	
6	Number of polypropylene macrospheres required	8875 units	Calculated
7	Polypropylene macrospheres/kg bag	2000	Vendor data
8	Bags of spheres required	4.43	Calculated

Based on the performance data sheet of FC-1100 it was assumed that there would be no interference with the extraction kinetics of LIX984N and stripping of loaded organic, if dosed at these points (Robinson *et al.,* 2013; 3M, 2016). Further, the additives would not interfere with the deposition rate of copper and the final cathode quality. Performance test results have been shown to display satisfactory performance at concentrations lower than 30ppm (Al Shakarji, 2012; 3M, 2016). In this study it was also assumed that FC-1100 would be dosed into PLS holding tank prior to pumping the pregnant leach solution to the electrowinning tank house, at a rate of 20ppm (3M, 2016). It was assumed that no foam would be generated during the electrowinning process.

4.2.4.4. Process contaminants

Iron

Iron that reports to the electrowinning circuit occurs as a result of aqueous ingress, slippage during the washing stage or co-extraction and must not exceed a concentration range of between 1 g/L and 3g/L in the electrowinning circuit. Sole &Tinkler (2016) have reported an average of 3.4g/L of iron in some African copper electrowinning circuits. Studies and experience have shown that for every gram increase of iron, a 2 to 3 percent loss in current efficiency is reported (Beukes & Badenhorst, 2009). For the process under study the concentrations of co-extracted iron determined, were deemed sufficiently low enough (0.005g/L) not to affect electrowinning of copper even if a wash stage were to be excluded in the solvent extraction circuit.

Manganese

The ability of manganese to assume several stable oxidation states contributes to an increase in the solution potential, resulting in a reduced current efficiency in the electrowinning operation. It reduces solution potential during copper electrowinning (Miller, 1995). Sole *et al.,* (2007) noted that commercial operations in African and Australian copper extraction circuits have high manganese concentrations in the liquors. The practice in some operations has been to dose the liquor with Fe^{2+} to maintain the manganese in a reduced state (Mn^{2+}). This is illustrated in the reactions 4.5 and 4.6 shown below:

$$Mn^{7+} + 5Fe^{2+} \longrightarrow 5Fe^{3+} + Mn^{2+} \tag{4.5}$$

$$Mn^{4+} + 2Fe^{2+} \longrightarrow 2Fe^{3+} + Mn^{2+} \tag{4.6}$$

While some operations in South America maintain the Fe:Mn ratio at 10:1, research indicates that other operations run their processes at lower ratios without experiencing challenges (Sole *et al.,* 2007).

Manganese is not chemically extracted during the solvent extraction stage but will report downstream of the solvent extraction circuit in the absence of a wash stage to the electrowinning circuit as a result of aqueous entrainment (Sole *et al.,* 2007). Manganese promotes the generation of chlorine gas. The presence of manganese is also associated with a reduction in the conductivity of the solution. It has been shown that as MnO_2 is produced in the system an accelerated degradation of the anodes is observed in the form of PbO (Hakakari, 1995; Miller, 1995; Sole *et al.,* 2007; Zhang, 2007b). As a result PbO_2-MnO_2 is formed that spalls into the bulk electrolyte solution leading to contamination of the copper cathode (Prengaman & Siegmund, 1999).

Recycling of electrowinning bleed solutions and aqueous entrainment into the organic phase during stripping introduces manganese into the organic phase. The presence of manganese in the organic phase exhibits deleterious properties through solvent degradation. The increased solution potential due to manganese is responsible for the observed degradation rates.

The degradation products formed are known to stabilize emulsions thereby increasing the likelihood of aqueous entrainment due to increased phase disengagement times (Miller, 1995). This entrained aqueous phase increases the load of contaminant metals reporting to the electrowinning circuit. The degradation products formed further contribute towards crud formation discussed in section 4.2.4.2.

The tolerable operating range of manganese lies between 10ppm and 500ppm. At concentrations in excess of 500ppm, the degradation of the anodes ensues. Manganese is also a constituent in some WPCBs as noted in the work of researchers in the field (Oishi *et al.*, 2007; Silvana *et al.*, 2013; Behnamfard *et al.*, 2013).

For the process under study, no manganese was however detected for the characteristic sample of WPCBs used for the mass balances in the developed process (Shuey & Taylor, 2004; de Waal, 2018).

Antimony and bismuth

Antimony and bismuth were not factored into the development of the original process flow sheets (de Waal, 2018). Results of the original empirical work did not detect the presence of antimony (Rossouw, 2015). These metals are also constituents of some WPCBs (Sum, 1991; Hagelüken, 2006; Oishi *et al.,* 2007; Hao *et al.,* 2008; Behnamfard *et al.,* 2013). As a result of aqueous entrainment the metals eventually report to the copper electrowinning circuit ultimately reducing the quality of the final product.

Organics

The carryover of organic into the electrowinning circuit is associated with solution corrosion that has been postulated to be a result of the action of free radicals on the exposed anode in direct contact with the electrolyte organic mixture. Mirza *et al.,* (2016) discussed the mechanism of attack by the free radicals. The carryover of organic cannot be ruled out entirely from the process operations of typical copper electrowinning operations. Beukes & Badenhorst (2009) have recommended a split operating configuration to handle the main production stream and the organic laden stream. For the purposes of the current study it was however assumed that organic entrainment was negligible as described in section 4.2.4.2 (under extractant losses).

Overview

From the discussion the effects of some of the constituent metals associated with the acid leaching of WPCBs to extract copper have been summarized below in Table 4.10.

Table 4.10: Summary of the effects of different contaminants and other metals in the copper extraction process (Hakakari, 1995; Miller, 1995; Sole *et al.,* 2007; Zhang, 2007b; Mirza *et al.,* 2016; Sole &Tinkler, 2016).

Metal	Specification	Remarks
Lead (Pb)		Reduces final copper cathode quality.
Manganese (Mn)	General < 40ppm 10ppm-500ppm (process specific)	Accelerates degradation of solvent, promoting aqueous entrainment and eventual contamination of electrolyte. Promotes chlorine formation, corrosion of the anode and reduces current efficiency.
Cobalt (Co)	100ppm-200ppm	Depolarizes the lead anodes resulting in a reduction in the corrosion rate and subsequent contamination of the electrolyte.
Iron (Fe)	1g/L-3 g/L	Reduces final copper cathode quality. Promotes process temperature rise, reduced current efficiency and generation of chlorine gas at the anode.
Nickel (Ni)		Reduces final copper cathode quality. Promotes process temperature rise, reduced current efficiency and generation of chlorine gas at the anode.
Chromium (Cr)		Reduces final copper cathode quality. Promotes process temperature rise, reduced current efficiency and generation of chlorine gas at the anode.
Antimony (Sb)		Reduces final copper cathode quality.
Bismuth (Bi)	100ppm-200ppm	Reduces final copper cathode quality.

Bleeding

Bleeding is practised to reduce the accumulation of additives and other species that have a negative impact on the electrowinning process. Such effects manifest as low current efficiencies, higher operating process temperatures, generation of higher volumes of corrosive vapours, dissolution of the anodes, an increased potential for contamination of the copper cathode and a higher frequency of rejects that must be recycled. The overall result is a higher power usage ratio reflected by lower returns per productive operating cycle time.

The proportion of liquor to be bled off is defined by the characteristic profile of the barren leachate exiting the electrowinning circuit. Major process changes arising from the variability of the feed, as is characteristic of WPCB recycling, destabilises the system. To cater for the variations in feed quality and the effects on the downstream processes, the mass balances across the process flow sheets were undertaken assuming consistent feed grade (de Waal, 2018).

While this is a hypothetical and almost non-existent scenario, provision in further analyses of the process for a reconditioning stage prior to entering the solvent extraction stage was assumed. This would have to be considered in greater detail in further refining of the process flow sheets should the preliminary economic studies prove to be viable. For the purpose and scope of this study however, a consistent feed quality into the solvent extraction circuit and from the solvent extraction circuit into the electrowinning circuit was assumed.

Four destinations exist for the bleed and these include leaching, pregnant leach solution, solvent extraction wash and raffinate (Ritcey & Ashbrook, 1979). Since the bleed carries contaminants (Mn, Pb, Fe and Ni) entering the potential destinations and residual copper, each of the destinations will experience a compositional disturbance from the original stable operating conditions. Through a consideration of the applicable kinetics and equilibrium driving forces at each of the destinations, for the process under study, it was thus opted to recycle the bleed to the leaching circuit (de Waal, 2018).

The movement of copper in the process was traced. In the initial development of the copper process flow sheet, it was assumed that 20% of the spent electrolyte was bled after every electrowinning cycle (de Waal, 2018). For the study it was assumed that a scavenging electrowinning section was included to polish effluent liquor from this section. The barren electrolyte from the scavenging section was bled and the remaining 80% was returned to the process in the stripping section. The detailed handling of the copper recovery is handled in section 7.3.2). A volume equivalent to the bleed from the spent electrolyte was directed to effluent treatment. Mass balances of fresh feed were adjusted to prevent accumulation in the circuit.

4.2.5.   Gold extraction

4.2.5.1.   Process chemistry

An analysis of the in the solids existing the copper extract plant revealed that the base metals content reporting to the gold extraction circuit was high. The treatment of feed material entering the gold extraction circuit or leach solutions reporting to the downstream processes in the gold extraction circuit was therefore considered in this study. The treatment can be achieved prior to cyanide leaching or after cyanide leaching. Two processes were considered for these options. The SART process and the concentrated nitric acid wash.

In the earlier development of the process flow sheets, incorporation of both treatment options was not included (de Waal, 2018). Further revision of the gold circuit incorporated a consideration of either pre cyanide treatment or post cyanide treatment.

A discussion of the process chemistry associated with cyanide leaching, the SART and nitric acid wash processes is outlined in the next section as a background to the analysis and selection of the most appropriate option that was undertaken. The metals considered in this section include all the metals specified in the analysis of WPCBs by Shuey & Taylor (2004). The analysis was used for the development of the process flow sheets and working out the respective mass and energy balances.

Localised galvanic effects contribute to the dissolution and existence of different ionic species in the reacting media. To accurately predict the predominant reacting species in alkaline or acidic systems pourbaix diagrams would have to be specified for handling complex polymetallic feed systems as characterised by WPCBs. For the preliminary studies, use was made of previous research investigations and published pourbaix diagrams (Bergstrom, 1924; Estay *et al.,* 2010; Karimi *et al.,* 2010; Uhlig, 2011; Rossouw, 2015; Webelements, 2018; docbrown, 2018) to predict the expected reacting species and states of the metals for both the cyanide leaching, the SART process and the nitric acid wash. Results from the mass balances were incorporated in evaluating the effectiveness of both treatment options and also in determining material recoveries in the updated gold process flow sheets.

4.2.5.2.　　Cyanide leaching

Typical residence times range between 20 to 40 hours (Stange, 2009). For the process under study it was assumed that a total of 24 hours was committed to the cyanide leaching process using 6 agitated leach tanks (de Waal, 2018).

Cyanide is capable of forming many complexes with a number of transition metals. In this section a number of reactions will be discussed that typically occur in gold cyanide leaching systems. The precipitation of the complexes within the process is governed by the pH, ionic strength, oxygen and cyanide concentrations.

Temperature

Higher temperatures are associated with lower oxygen partial pressures. In such a case the use of pressurized leaching systems with oxygen enrichment becomes necessary (Marsden & House, 1992). While studies have supported the operation at high temperatures, reflected by higher gold dissolution rates, operating the process, at temperatures in excess of 85^OC results in a depletion of dissolved oxygen. Industrial practice recommends a dissolved oxygen concentration greater than 6ppm. Elevated oxygen partial pressures however promote the rate of zinc dissolution in a manner that consumes cyanide and depresses gold dissolution.

Since WPCBs contain gold concentrations exceeding that of native ores (Hagelüken, 2006), oxygen enrichment at 25^OC was applied in the process under study (de Waal, 2018).

Precious metals

While PGMs were not detailed in the process flow sheets originally developed, mention of the fate of PGMs has to be considered. In a more practical scenario dependent on more varied WPCB sources, the existence of PGMs especially platinum and palladium are expected (Iji & Yokoyama, 1997; Zhang & Forrsberg, 1997; Hagelüken, 2006; Ogunniyi & Vermaak, 2007; 2009;).

During cyanidation it is expected that recovery rates of palladium will exceed platinum recovery rates (van Wyk, 2014) so that the residual solid fraction would be directed for further treatment or sale to refiners specialising in platinum recovery. Palladium would report in greater concentrations with gold.

In studies conducted investigating the recovery of gold and PGMs from low grade Platreef ores, it was shown that at the reaction conditions specified, a two stage cyanide leach process would suffice to enhance platinum recovery. It was proposed that the PGMs would be isolated prior to pumping the gold rich solution to the electrowinning circuit (Mpinga, 2012; Schoeman, 2012; Snyders *et al.,* 2013; van Wyk, 2014).

The dissolution of gold, palladium and platinum has been described using Elsner's equation (Adams, 2016):

$$4\,Au + CN^- + O_2 + 2H_2O \longrightarrow 4Au(CN)_2^- + 4OH^- \tag{4.7}$$

$$2Pt + 8CN^- + O_2 + 2H_2O \longrightarrow 2Pt(CN)_4^{2-} + 4OH^- \tag{4.8}$$

$$2Pd + 8CN^- + O_2 + 2H_2O \longrightarrow 2Pd(CN)_4^{2-} + 4OH^- \tag{4.9}$$
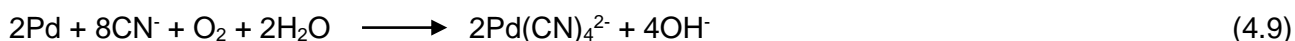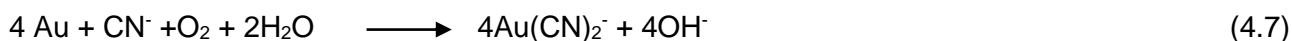
The leaching of palladium and platinum according to reactions 4.8 and 4.9 are noted to be slow at ambient operating conditions.

Tin

Tin was assumed to exist in two forms in this study; as the amphoteric SnO and in its native state. Prior to treatment of the feed material to remove base metals, it was assumed that the finely divided tin was oxidised due to the localised heated generated during shredding and comminution of the WPCBs (Li *et al.,* 2010). Amphoteric SnO was assumed to have been produced.

Following the nitric acid wash under the specified reaction conditions, any of the tin nitrate that was formed was rapidly hydrolysed to metastannic acid. In highly acidic conditions tin exhibits behaviour similar to silisic acid (Manjengwa, 2011), forming a stable colloid gel insoluble in acid or water and is difficult to filter. To improve the filtration characteristics on a more practical note the pH of the stream exiting the acid wash tank should be raised to just above 7.

Empirical verification will be required in further refinement of the process flow sheets in separate studies. For the study it was assumed that the slurry formed did not experience severe pressure drops due to premature blinding of the filter cloths. For the purposes of the study it was also assumed that tin was carried over as metastannic acid to the cyanide leaching stage.

At increasing pH values $Sn(OH)_4$ was formed that dissolved in excess alkali to yield $Sn(OH)_6^{2-}$ and reacted with cyanide and calcium ions. Sodium metastannate and calcium metastannate were the two dominant products formed in the alkaline media. For the study in the presence of calcium ions calcium stannate was assumed to have been precipitated continuously until the metastannate was exhausted. Calcium stannate is insoluble in water at the temperature conditions specified. The solid product was filtered out prior to the process stream reporting to the adsorption train. As a result, calcium hydroxide was added to compensate for this precipitation reaction in the material balances. The precipitation of the tin was assumed to follow the reactions 4.10 and 4.11 described below:

$$H_2SnO_3 + 2OH^-(aq) + H_2O \longrightarrow Sn(OH)_6^{2-}(aq) \tag{4.10}$$

$$Sn(OH)_6^{2-}(aq) + Ca^{2+}(aq)^- \longrightarrow CaSnO_3(s) + 3H_2O \tag{4.11}$$
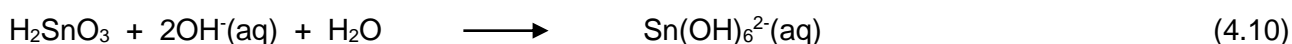
As the conditions become very acidic the stannic acid is reformed. For the study it was assumed that all the tin was eliminated during the filtration of the gold leach slurry.

Lead

Lead (II) ions in cyanide react with gold to yield $AuPb_2$, $AuPb_3$ and metallic lead. Metallic lead on the gold surface has been reported to have a depolarizing effect on the reduction of oxygen (Adams, 2016). Makanza (2006) showed that at a pH close to 7, lead exists as $PbOH^+$ species wherein 20% of the lead used in the investigations existed as Pb^{2+} ions. It is reported that Pb^{2+} does not react directly with cyanide. The predominant species at pH values greater than 10 for gold cyanidation include, $Pb(OH)_3^-$, $Pb(OH)_2^-$, $PbOH^+$.

In the presence of Pb^{2+}, the reactions associated with the dissolution of gold were reported in literature (Adams, 2016) as follows:

$$2Pb(OH)_2 + 5Au(s) + 8CN^-(aq) \longrightarrow AuPb_2(aq) + 4Au(CN)_2^- + 4OH^- \tag{4.12}$$

$$AuPb_2 + 2CN^- + 4OH^- \longrightarrow Au(CN)_2^- + 2PbO + 2H_2O + 5e \tag{4.13}$$

$$2PbO + 5Au + 8CN^- + 2H_2O \longrightarrow AuPb_2 + 4Au(CN)_2^- + 4OH^- \tag{4.14}$$

$$Pb(OH)_2 + 4Au + 4CN^- \longrightarrow Au_2Pb + 2AuPb + 2Au(CN)_2^- + 2OH^- \tag{4.15}$$

$$3Pb(OH)_2 + 7Au + 12CN^- \longrightarrow AuPb_3 + 6Au(CN)_2^- + 6OH^- \tag{4.16}$$
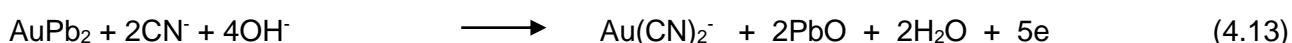
Zinc

In oxygenated alkaline systems zinc forms a complex with cyanide as described by the Elsner reaction and $Zn(OH)_2$. At higher ratios of free zinc in solution gold dissolution can be retarded. Research however reports that at decreasing free cyanide concentrations, the liberation of CN^- ligand for uptake in gold is favoured (Makanza, 2006; Adams, 2016). Several competing reactions exist due to the formation of hydroxyl ions. Zinc undergoes the following reactions in the system:

$$2Zn + 8CN^- + O_2 + 2H_2O \longrightarrow 2Zn(CN)_4^{2-} + 4OH^- \tag{4.17}$$

$$2Zn + 8CN^- + 2H_2O \longrightarrow 2Zn(CN)_4^{2-} + 2(OH)^- + H_2(g) \tag{4.18}$$

$$2Au(CN)_2^- + Zn + 3OH^- \longrightarrow 2Au + HZnO_2^- + 4CN^- + H_2O \tag{4.19}$$

$$Zn + 2OH^- \longrightarrow Zn(OH)_2 \tag{4.20}$$
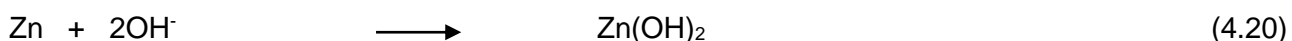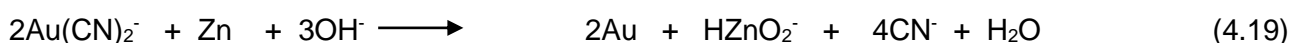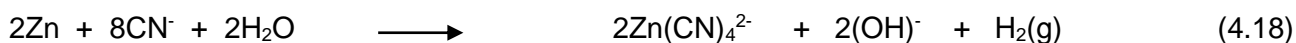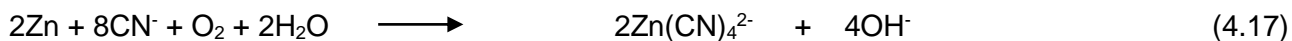
Silver

An analysis of the initial raw material feed concentration showed that the concentration ratio of silver to gold was 1.63:1, which is economically significant. Research has shown that under normal alkaline cyanide gold leaching conditions, the silver dissolution rates are inferior to gold. In the previous studies it was assumed that the complete dissolution of silver occurred under the reaction conditions specified (de Waal, 2018).

Copper

Copper dissolves in alkaline cyanide solutions though the rate is superseded by that of both gold and silver. Solubility has been shown to increase with increasing temperature. Based on the Eh-pH diagram for $Cu-CN-H_2O$ system the species expected to dominate at pH values greater than 10 is $Cu(CN)_3^{2-}$. Other cyanide species of copper, $Cu(CN)_2^-$ and $Cu(CN)_4^{3-}$, have also been reported to exist in the normal gold extraction conditions. This is influenced by the concentration of free cyanide species in the system. The species exhibit similar stability constants under these conditions (Marsden & House, 1992). At high cyanide concentrations $Cu(CN)_4^{3-}$, predominates. This species is not adsorbed as well as $Cu(CN)_3^{2-}$.

Iron

After the nitration of the residue to remove base metals Fe^{2+} is rapidly oxidised to Fe^{3+} in the presence of oxygen, with a resultant mixture of Fe^{2+} and Fe^{3+} cyanide complexes that are stable (Marsden & House, 1992). In the presence of OH^- it is expected that $Fe(CN)_6^{3-}$, $Fe(CN)_6^{4-}$ and $Fe(OH)_3$ will predominate. $Fe(OH)_3$ must be avoided since this reduces the leaching efficiency due to the precipitation of $Fe(OH)_3$ onto the surface of the gold particles. In the presence of cations double salts can be formed (Marsden & House, 1992). Some of the salts expected to be formed during cyanidation are summarized in Table 4.11 below.

Antimony and arsenic

These species do not form cyanide complexes in their native state. Arsenic was not detected in the analysis used for the study (Shuey & Taylor, 2004; Rossouw, 2015). For the process under study antimony was not factored in the development of the flow sheets (de Waal, 2018).

Mercury

Mercury forms complexes with cyanide ligands. It is a competitor with gold for the ligands and exhibits the ability to displace CN^- ligands from zinc.

Table 4.11: The possible species of cyanide complexes during the cyanidation of WPCB leach liquor (Marsden & House, 1992)

Oxidation	Nickel	Copper	Zinc	Silver	Iron	Gold
CN is oxidised	$NiCN^-$	$Cu(CN)_2^-$	$ZnCN^+$	$Ag(CN)(s)$	$Fe(CN)_6^{4-}$	$Au(CN)_2^-$
	$Ni(CN)_4^{2-}$	$Cu(CN)_3^{2-}$	$Zn(CN)_2(aq)$	$Ag(CN)_2^-$	$Fe(CN)_6^{3-}$	$Au(CN)_4^-$
	$NiH(CN)_4^-$	$Cu(CN)_4^{3-}$	$Zn(CN)_3^-$	$Ag(CN)_3^{2-}$	$Fe_2(Fe(CN)_6)(s)$	
	$NiH_2(CN)_4$	$CuCN(s)$	$Zn(CN)_4^{2-}$	$Ag(CN)_4^{3-}$	$Fe(Fe(CN)_6)(aq)$	
	$NiH_3(CN)_4$	$Cu_2(Fe(CN)_6)(s)$	$Zn(CN)_5^{3-}$	$Ag(OH)CN^-$	$Fe_4(Fe(CN)_6)_3(s)$	
	$Ni(CN)_6^{4-}$	$Cu_3(Fe(CN)_6)_2(s)$	$Zn_2(Fe(CN)_6)(s)$	$Ag_4(Fe(CN)_6)(s)$		
	$Ni_2(Fe(CN)_6)(s)$					

Adams (2016) noted that the most important complexes formed during gold cyanidation include: $Au(CN)_2^-$, $Cu(CN)_3^{2-}$, $Pd(CN)_4^{2-}$, $Pt(CN)_4^{2-}$, $Ag(CN)_2^-$, $Zn(CN)_4^{2-}$, $Fe(CN)_6^{4-}$, $Ni(CN)_4^{2-}$, $Co(CN)_6^{3-}$ and $Hg(CN)_4^{2-}$. Mercury cobalt, platinum and palladium were not considered in the previous studies (de Waal, 2018). Based on the analysis by Shuey & Taylor (2004) platinum was not present.

Table 4.12 provides a summary of the expected reactions and observations that were incorporated in the study for the constituent metals associated with the WPCB raw material feed that reported from the copper extraction circuit.

Table 4.12: Expected reactions during cyanide leaching (Bergstrom, 1924; Makanza, 2006; Estay *et al.*, 2010; Karimi *et al.*, 2010; Uhlig, 2011; Rossouw, 2015; Webelements, 2018; docbrown, 2018)

Cyanide leaching	
Metal	**Expected reactions and remarks**
Al	Aluminium is rapidly oxidised in air to form Al_2O_3.
	This reaction was expected during the shredding and comminution of the WPCBs.
	This points to the existence of Al_2O_3 and aluminium metal (protected by the oxide layer) in the feed.
	The reactive form of Al_2O_3 was assumed to react with the base during cyanide leaching to form aluminate.
	It was assumed that metallic aluminium was in negligible quantities; as a result the displacement of other transition metals was considered insignificant.
	It was assumed that there were no cyanide complexes of aluminium since the complex is rapidly hydrolysed to yield a soluble complex of $Al(OH)_6^{3-}$ in strongly alkali media.
Zn	Zinc metal also undergoes cyanidation.
Fe	Formation of ferrous cyanide complexes ($Fe(CN)_6^{4-}$) was assumed for the study though at inferior rates to zinc.
Pb	In the finely divided state it was assumed that lead would react to form plumbates and intermediate species.
Sn	It was assumed that the finely divided tin was oxidised due to the heated generated during the shredding and comminution to generate the amphoteric SnO.
	At increasing pH values $Sn(OH)_4$ was formed that reacted with cyanide and calcium ions prior to the nitric acid wash.
	Calcium stannate was precipitated.
Au	It was assumed that the predominant complex of gold formed was $Au(CN)_2^-$.
Ag	Silver was assumed to form cyanide complexes.
	The most predominant species being $Ag(CN)_2^-$ (Makanza, 2006).
Ni	It was assumed that nickel underwent cyanidation though the rate would be inferior to zinc.
	The predominant complex formed was expected to be $Ni(CN)_4^{2-}$.
Cu	Copper cyanide complexes formed are readily adsorbed onto carbon (Mpinga, 2012)).
	A consideration of the existing species was made through a study of the E-pH diagram for the Cu-H_2O-CN system at 25^OC (Dai *et al.,* 2012).
	The most predominant species assumed to be adsorbed for this study were $Cu(CN)_2^{3-}$ and $Cu(CN)_2^-$.

Cyanide usage

Maintaining a pH above 10 suppresses the hydrolysis of the free cyanide and the dissociation of the complexes formed with base metals. The formation of cyanate is a non essential reaction that is promoted in conditions of high oxygen partial pressures. The result of these competing reactions necessitates dosing the system with NaCN concentrations that maintain the desired gold dissolution rates. The concentrations are adjusted accordingly as contaminant concentrations and subsequent gold dissolution rates drop or increase. This is verified empirically during pilot studies. Cyanide degradation rates have been reported to range between 0.1g/L and 0.3g/L as NaCN.

The in-process decomposition of cyanide, as a result of the catalytic effect of carbon proceeds according to equation 4.21:

$$CN^-(aq) + O_2 + 4H_2O \rightleftharpoons CO_3^{2-}(aq) + 2NH_4^+ \qquad .(4.21)$$

In the presence of Ca^{2+}, $CaCO_3$ (aq) is generated. The $CaCO_3$ (aq) formed tends to precipitate within the pores of the activated carbon. This contributes to a reduction in the activity of the activated carbon and hence gold adsorption efficiency (Marsden & House, 1992).

The concentration of competing metal species in the process under study was assumed to be fixed. Cyanide degeneration was assumed to proceed at a rate of 0.1g/L. A NaCN concentration of 250 ppm (25% excess of the stoichiometric amount) was specified for the process (de Waal, 2018). The excess was considered suitable to compensate for cyanide degeneration on the carbon surfaces, unwanted side reactions associated with the non beneficial consumption of cyanide and the competitive consumption of cyanide as a result of contaminant base metal species. A significant cost driver in cyanide leaching processes is the cyanide usage rate (Marsden & House, 1992). An analysis of the fluctuating rates to minimize the changing concentrations of contaminants was also undertaken.

4.2.5.3.　　Base metals management

Nitric acid wash - Pre cyanide treatment of feed

The objective of this process was to eliminate all base metals that were carried over from the previous processes (size reduction, solder leaching and copper extraction circuits). An analysis of the base metal content in the feed streams revealed that the constituent ratios of the metals under study were high when compared to standard industrial practice. The subsequent accumulation of chloride in the system following hydrochloric acid washing resulted in its rejection as a potential stripping agent. The benefits of washing feed with nitric acid have also been demonstrated by Karimi *et al.,* (2010). The researchers reported an increased gold and silver recovery during cyanide leaching of ores rich in base metals coupled with a reduction in cyanide consumption by 8.5% and 8.4% for the experimental runs. The nitric acid wash circuit was therefore developed. It was added to the original gold process flow sheet and is outlined in the Figure 4.5 below:
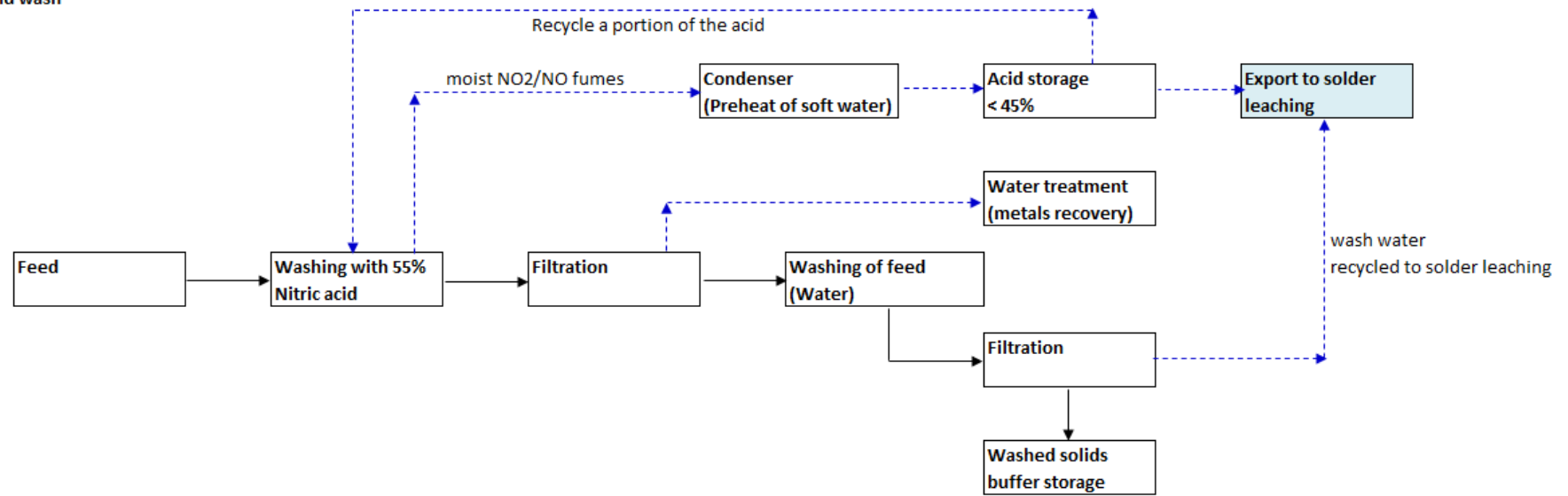
Figure 4.5: The proposed process flow sheet of the nitric acid wash circuit

Table 4.13 provides a summary of the expected reactions and outputs of the nitric acid wash.

Table 4.13: Expected reactions during the nitric acid wash (Kleinschmidt, 1918; Bergstrom, 1924; Driver, 1988; Makanza, 2006; Estay *et al.,* 2010; Karimi *et al.,* 2010; Uhlig, 2011; Rossouw, 2015; Webelements, 2018; docbrown, 2018)

Nitric acid wash (55%)	
Metal	**Expected reactions and remarks**
Al	Aluminium was rapidly oxidised in air to form Al_2O_3.
	This reaction was expected during the shredding and comminution of the WPCBs.
	This points to the existence of Al_2O_3 and aluminium metal (protected by the oxide layer) in the feed. For the study it was assumed that the reactive species of Al_2O_3 was formed.
	The Al_2O_3 was assumed to react with the nitric acid to ultimately form Al^{3+} ions.
Zn	Zinc metal was attacked by nitric acid to form stannic acids (Uhlig, 2011).
Fe	The iron was attacked by the nitric acid.
	Formation of Fe^{3+} was promoted by an oxidising environment.
Pb	Lead in the finely divided state will react to form lead nitrate.
	It was assumed in this study that no lead oxide existed.
	Formation of a passive layer of PbO was expected at temperatures in excess of 600^OC.
	The presence of lead accelerated the attack of aluminium and zinc (Uhlig, 2011).
Sn	It was also assumed that some of the tin formed SnO during the comminution stage due to the heat generated during the size reduction of the WPCBs.
	The reaction system was highly exothermic and for the conditions specified it was assumed that tin would react to form stannic acids.
	Initially a precipitate of metastannic nitrate was formed that underwent rapid hydrolysis to form metastannic acid.
	It was also assumed that the gel precipitate was carried over to the cyanide leaching stage.
	The stable colloid of metastannic acid was destabilized in the alkaline cyanide leach.
	Under strongly alkaline conditions it was assumed that calcium stannate was precipitated.
	Any residual tin was eliminated at the HCl acid pre-soak and wash stage prior to elution of the carbon.
Au	Gold was assumed to remain inert under the reaction conditions.
Ag	It was assumed that silver nitrate was formed.
Ni	It was assumed for the study that nickel reacted with the nitric acid to yield nickel nitrate ($Ni(NO_3)_2$)
Cu	Copper was assumed to react with nitric acid to form copper nitrate ($Cu(NO_3)_2$) with the liberation of nitrogen dioxide.

The SART process – Post treatment of the leach solution

This process was originally developed to achieve a reduction in the concentration of copper reporting to the adsorption and electrowinning circuits from gold heap leach liquors. It also provides a means for achieving the recovery of cyanide as a result of the destruction of weak acid dissociable (WAD) dissociable cyanide complexes from base metals like copper, nickel, zinc, silver and mercury (Estay *et al.,* 2010).

Pregnant leach solution is acidified to a pH of between four and five. This is done using sulphuric acid. The pH drop results in the generation of hydrogen cyanide that is captured in the scrubbing stage downstream. Sodium hydrogen sulphide is added as a source of HS^- that is necessary for the formation of sulphides that precipitate in acidic media (Karbanee *et al.,* 2008).

Rapid settling rates are achieved through the use of a flocculant. A portion of the solids under-flow is recycled in order to accelerate the precipitation process through seeding. The remaining filtered sulphides are dewatered and neutralised using sodium hydroxide prior to exporting to the smelting process to recover the copper.

The leach liquor from the thickening stage is neutralized with hydrated lime to a pH of 11. Most of the hydrogen cyanide (HCN) which was retained in the solution phase reacts with the lime to form calcium cyanide ($Ca(CN)_2$). Due to the presence of sulphate, gypsum dihydrate and gypsum hemihydrate are formed. The conditions of operation and moisture content do not support the formation of the finer anhydrous crystals. The solution is filtered and gypsum with a maximum moisture content of 10% is recovered for storage or sale. The filtrate and settler overflow is then redirected into the main process with a reduced base metal content. Hydrogen sulphide (H_2S) and HCN generated in the process is captured and scrubbed using sodium hydroxide (NaOH) and water (Ford *et al.,* 2008; Estay *et al.,* 2010). The outline of the process is depicted in Figure 4.6.

Figure 4.6: The SART process (Redrawn (Estay *et al.,* 2010)

An attempt to apply the SART process to the current gold recovery under study was done. The assessment of likely by-products aligned with the characteristic profile of the feed material was done and the findings were summarized in Tables 4.14 and 4.15. The analysis revealed that adoption of the SART process within the gold flow sheet would realise a reduction in the lead, copper, tin, iron and silver content reporting to the adsorption section. Aluminium, nickel and zinc would however be carried over.

Table 4.14: Expected reactions during the SART process (Bergstrom, 1924; Makanza, 2006; Manjengwa, 2009b; Estay *et al.,* 2010; Karimi *et al.,* 2010; Uhlig, 2011; van Wyk, 2014; Webelements, 2018; docbrown, 2018)

SART process	
Metal	**Expected reactions and remarks**
Al	It was assumed that the reactive form of Al_2O_3 formed was attacked by sulphuric acid to form aluminium sulphate.
	The existence of aluminium sulphate aids in settling of the precipitate formed after acidification an sulphidzation since it is a coagulant.
	However the agglomerated particles are dispersed and settling is poor in the acidic pH range 4-5 (Manjengwa, 2009b).
	The formation of Al_2S_3 was considered non-existent since the sulphide is rapidly hydrolysed in the presence of water.
Zn	Zinc ions interact with HS^-. Zinc sulphide dissolves in acid to yield zinc oxide.
	In excess base $Zn(OH)_4^{2-}$ was produced and carried over to the adsorption section.
Fe	FeS formed reacts with sulphuric acid to yield FeO and SO_2.
	Expected to precipitate during the neutralisation stage to form $Fe(OH)_3$.
	Iron will be recovered as a solid precipitate along with gypsum.
Pb	Lead will react with NaSH to form PbS precipitate.
Sn	The metallic tin was assumed to react with NaSH to yield SnS_2 which precipitates in acidic media Uhlig (2011).
Au	The gold complex was assumed not to dissociate under the process conditions.
Ag	The silver was assumed to be recovered as AgS along with the sulphides that are insoluble in the acidic media.
Ni	Ni^{2+} ions from the sulphate salt forms sulphides which are sparingly soluble in acidic medium. Nickel was assumed be recovered as NiS and $Ni(H_2O)_4(OH)_2$.
Cu	Copper was assumed to be recovered as the CuS precipitate.

Table 4.15: The expected behaviour and destinations of the by-products of the SART process for the system (Bergstrom, 1924; Estay *et al.,* 2010; Karimi *et al.,* 2010; Uhlig, 2011; Rossouw, 2015)

Metal	Solubility of sulphide in acid	Solubility of sulphide in base	Remarks
Pb	No	-	PbS precipitate formed and removed with CuS.
Cu	No	-	CuS formed and settling is accelerated using flocculent.
Sn	No	-	SnS_2 is formed and reports to settler with PbS and CuS.
Fe	-	-	FeS formed reacts with sulphuric acid to yield FeO and SO_2.
			Expected to precipitate during the neutralisation stage to form $Fe(OH)_3$.
			Iron will be recovered as a solid precipitate along with gypsum.
Ni	-	No	Ni^{2+} ions from the sulphate salt forms sulphides which are sparingly soluble in acidic medium.
			Nickel will be recovered as NiS and $Ni(H_2O)_4(OH)_2$.
Zn	-	No	Zinc sulphide dissolves in acid to yield zinc oxide.
			In excess base $Zn(OH)_4^{2-}$ is produced and carried over to the adsorption section.
Ag	No	No	The sulphide will decompose in very strong acids.
			The silver will be recovered as AgS along with the sulphides precipitated during treatment with H_2S.
Al	-	-	Stable sulphide will not be recovered.
			Undergoes rapid hydrolysis to yield hydrated oxides and hydroxides. In basic media it will exist as aluminate.
			Aluminium will be carried over to the adsorption circuit as $Al(OH)_6^{3-}$.

Comparison of the two options

An analysis of the key attributes of the nitric acid wash and the SART process was undertaken. A prediction of the expected concentrations of the metals reporting to the adsorption section was done. The findings of the analyses are summarized in Table 4.16 and appendix C (Tables C4 and C5).

Table 4.16: A comparative analysis of the two options for managing base metals in gold feed from the copper extraction circuit.

Metrics	Pre-treatment: Nitric acid wash	Post –treatment: SART
Reagents consumption	55% Nitric acid Water	Hydrated lime NaOH NaSH H_2SO_4 Flocculent
By products	Regenerated nitric acid (≈10%) Hot soft water for elution circuit Nitrous oxides, chiefly NO_2	Gypsum Sulphide precipitates (Ag, Cu, Pb, Sn) Recovery of cyanide Hydrogen sulphide Ferric hydroxide
Efficiency	>98%. Gold composition with respect to other metals in feed is quadrupled prior to leaching. Residual copper for the process under study reporting to the adsorption circuit was 0.017ppm. Refer to Appendix D for the detailed analysis.	65-95% (Ford *et al.,* 2008; Estay *et, al.,* 2010) Residual copper for the process under study reporting to the adsorption circuit was 82.14ppm. Refer to Appendix D for the detailed analysis.
Objective features	Removal of competing metals prior to cyanide leaching and adsorption. Enriching the solid feed prior to cyanide leaching.	Removal of copper and other competing metals prior to adsorption and elution. Recovery of copper and other base metals as insoluble sulphides for smelting. Recovery of cyanide
Drawbacks	The need to contain the oxides of nitrogen. Feed rate into the reactor must be monitored carefully. The process is highly exothermic. As the wash cycles increase the acid bled and returned into the wash progressively weakens, increasing acid to effluent treatment. There is need to capture NO_2 exiting the solder leach circuit. High entry gas temperatures do not favour high NO_2 adsorption rates.	HCN generated must be well contained. Generation of H_2S. The need for additional capital investment in flocculation and dewatering equipment. The potential calcium loading to the adsorption circuit is elevated. The process does not always guarantee 100% removal of base metals (Estay *et al.,* 2010). Extensive bench studies required prior to using it in the process. Increased likelihood of $CaCO_3$ crystallizing within the activated carbon. There is a reduction in active sites and pathways within the activated carbon are blocked. A risk of gypsum carryover exists that can be deposited in the macro pores (Estay *et al.,* 2010). Risk of increased carbon losses as $CaCO_3$ during thermal regeneration. Carbon losses are noted to be 6% higher for calcium concentrations increased from 0.10% to 0.5% on the activated carbon. (Marsden & House, 1992) There is an introduction of organics from subsequent flocculation and settling operations that foul the carbon.

Using published process data (Ford *et al.,* 2008) for the SART process, a prediction (applicable to the process under study) of the expected copper recovery was calculated to be at least 66% as copper sulphide.

Based on the comparison of residual copper reporting to the adsorption circuit it was established that the nitric acid wash displayed superior performance results. Further studies are however required to examine the efficiencies of the SART process in the removal of other base metals in the polishing of WPCB leach solutions.

Having considered the merits and drawbacks associated with both options the nitric acid wash circuit was selected as the most suitable measure to reduce the contaminant metal concentrations prior to the PLS reporting to the adsorption train. Outlined in Figure 4.7 is the schematic representation of the proposed gold extraction process flow sheet used in this study.
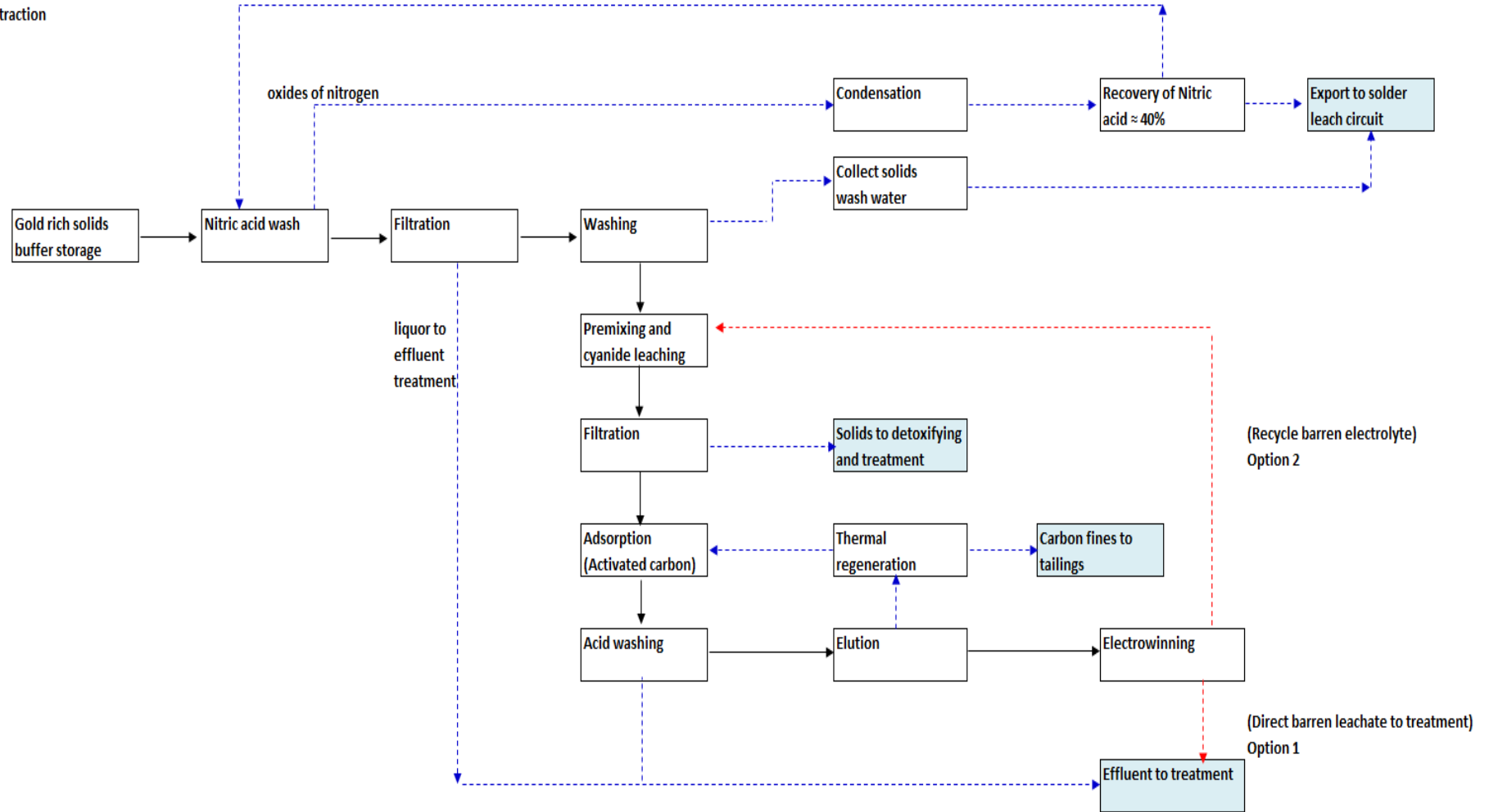
Figure 4.7: The gold recovery circuit

4.2.5.4. Carbon adsorption

For the process developed, the rate and loading of carbon was supported by typical industrial performance data (de Waal, 2018). High carbon loadings result in gold and carbon inventories with the added risk of higher losses of gold due to the higher carbon losses resulting from the effects of attrition. With the incorporation of the nitric acid wash circuit, it was established that the gold head grade to the adsorption circuit with relative to the concentration of other metals was at least doubled. Three options were considered to adapt the process to this change.

The first was option entailed using a lower solids to liquids ratio during the cyanidation step. The second option involved diluting the filtered leach liquor with softened process water to match the original design parameters. This implied a doubling of the volumes handled. The third alternative was to increase the carbon concentration to match the increased gold head grade. A consideration of the operating time frames within the process circuit indicated that a bottle neck existed during the leaching stage. A total residence time of 24 hours was allocated (de Waal, 2018). The adsorption, acid wash and elution stages have a combined operating cycle time of up to 12 hours (Stange, 2009; Rogans, 2012). The second option was selected (refer to appendix C, Tables C.1 and C.2). The liquor buffer storage tank was therefore doubled in size.

A consideration of the losses of gold to tailings indicated that higher losses would be associated with increased carbon concentration coupled with losses in the barren leach solution (Marsden & House, 1992; Snyders *et al.,* 2017). For the study granular activated carbon (coconut shell) at a concentration of 15g/L was specified (de Waal, 2018). It was also assumed that gold to tailings in the barren solution ranged between 0.001ppm and 0.02ppm consistent with standard industrial practice (Stange, 1999).

In typical industrial practice, carbon is replaced after 400 cycles of use if contaminant metal loadings are high (Marsden & House, 1992). Debottlenecking of the entire scope operations (refer to section 5.3.1, Table 5.3) under study revealed that the carbon in the gold section would undergo approximately 27 cycles of use each month, which was estimated to equate to 283 cycles per year. Since the nitric acid wash was included on the original gold process flow sheet it was assumed that contaminant loading would be comparatively lower than typical industrial process leach liquors derived from ores. Based on the calculated cycles of carbon usage it was assumed that carbon replacement would be done after a period of service ranging between 1.5 years and 2 years.

4.2.5.5. Acid washing

Washing of the loaded carbon with hydrochloric acid prior to elution was incorporated in the process. A total residence time of 140 minutes for the acid wash cycle was assumed based on industry practice.

Hydrochloric acid and nitric acid are the two acids typically used for this process (Marsden & House, 1992; Stange, 2009; Rogans, 2012). The use of nitric acid is typically associated with the oxidation of the carbon surfaces. This results in a reduction of the active surface area for adsorption. Nitric acid is however beneficial in the removal of mercury.

AARL acid wash stages were adopted in the specified process under study (de Waal, 2018). Briggs (1983) described the detailed operating conditions and time profiles for a typical AARL acid wash cycle process during the commissioning of the CIP plant at Beisa mine. A similar approach was adopted in the operation of the proposed process in the current study. It was assumed that the cycle consisted of an acid wash cycle that was followed by an acid soak cycle and ended with the acid rinse cycle which has been summarized in Table 4.17.

Table 4.17: AARL acid wash cycle (Briggs, 1983; Steyn, 2010; Rogans, 2012; de Waal, 2018)

Stage	Column loading	Acid wash	Acid soak	Acid rinse	Column drainage
Residence/operation time	15 minutes	20 minutes	40 minutes	90 minutes	15 minutes
Reagents	Loaded carbon from adsorption	3% HCl circulated, 90°C	3% acid solution remains stagnant in the column. 90°C	4 bed volumes of water soft water are used to rinse the carbon. 90°C	Washed carbon

The conditions outlined in Table 4.17 combine typical operating conditions described in literature (Briggs, 1983; Steyn, 2010; Rogans, 2012; de Waal, 2018). A hydrochloric acid concentration of 3% was specified for the process under study, similar to that used in industry. During the acid wash stage other complexes are formed as a result of the presence of the chloride ions. Nickel forms a precipitate within the pores of the carbon. To ensure the effective removal of these base metals and the associated cyano-chloride complexes the washing is achieved at temperatures in excess of 30°C (Rogans, 2012). It was assumed that gold, silver and copper cyanide complexes were not removed. All the residual copper reported to the gold electrowinning circuit (Marsden & House, 1992; Rogans, 2012; Steyn, 2010). A process temperature of 90 ^{O}C under a maximum pressure of 101KPa was specified for the process under study (de Waal, 2018). It was also assumed that the design of the circuit provided for the containment of the liquid and gas phases.

The generated HCN would be extracted and appropriate measures were in place to contain any generated HCN. For the process under study it was also assumed that moderate calcium loadings were expected in a well mixed system and that 95% of the $CaCO_3$ would be removed in this process. Secondly precipitated $Ni(CN)_2$ within the carbon pores was re-dissolved during the acid soak and was washed out with the other base metals. Rogans (2012) noted that higher acid wash temperatures promote the deposition of gold within the carbon and the formation of polymeric aurocyanide.

It was assumed for this study that most of the gold formed in the hot acid wash would remain trapped in the micropores of the carbon together with the polymeric aurocyanide. No gold was assumed to have been lost during the acid wash cycle. Further it was assumed that this gold would be re-dissolved during the elution stage.

The presence of iron in the reduced state facilitates the reduction of stannic compounds to stannous chloride which is soluble (Driver, 1988). For the study it was assumed that all the tin reacted with the hydrochloric acid to form stannic tetrachloride pentahydrate under the reaction conditions. It was assumed that the conditions prevailing would not support the formation of a stable colloid gel of metastannic acid. This chloride of tin is very soluble and it was assumed that all the tin was washed to effluent prior to the stream reporting to the elution stage. The iron present was assumed not to be in comparatively sufficient proportions to yield significant amounts of stannous chloride. The iron complexes of chloride and cyanide ligands that were formed were assumed to have been washed out to the effluent stream.

Removal of organics

Four major sources of organic foulants have been identified. They arise from the raw material or activities within the process. These foulants can be either volatile or non volatile. The non volatile contaminants are eliminated by the use of steam in temperatures exceeding 650°C (Rogans, 2012).

For the process specified it was assumed that the concentration of organic substances in the system is negligible and was therefore not factored into the mass and energy balances and the subsequent economic study. No flocculants were used for the process specified.

4.2.5.6. Elution

Elution refers to the stripping of the loaded carbon. By a reversal of operating conditions the gold is desorbed from the loaded carbon (Stange, 2009). In previous studies (de Waal, 2018) the AARL elution process conditions of operation were selected. It was noted that this process is the most preferred in African and Australian gold processing operations. The major steps associated with the AARL elution are summarized in Table 4.18. The associated residence times were also included in the Table.

Table 4.18: AARL elution process adopted for the designed process under study (Briggs, 1983; Steyn, 2010; Rogans, 2012; de Waal, 2018)

Stage	Carbon loading	Soak	Recycle elution stage	Water elution stage	Carbon cooling and discharge
Residence/operation time	15 minutes	30 minutes	60 minutes	240 minutes	45 minutes
Conditions	90 ^{O}C, 101KPa	110°C, 230KPa	115°C, 230KPa	110°C, 230KPa	25°C, 101KPa
Reagents	Washed carbon	2%NaCN 3% NaOH 1 bed volume	2 bed volumes gold concentration 10% in soft water with less than 300g Na/t in water	8 bed volumes of soft water with 300g Na/t in water	High quality municipal water
Performance		Over 95% gold cyanide complex is weakened.	60% of gold desorbed from carbon	27% of gold desorbed from carbon	

In some commercial gold extraction processes glycols or alcohols are added to promote gold desorption from the carbon. The addition of these organics is beneficial for the removal of organic foulants during high temperature elution. For the scope of this study it was assumed that no additives were used for the elution process.

Typical industrial performance data has shown that the retained gold on the carbon does not exceed 3% of the gold adsorbed onto the carbon (Rogans, 2012). One percent of the gold adsorbed was assumed to have been retained on the stripped carbon which was estimated to be just under 4g/t based on the mass balances (de Waal, 2018). Industrial practitioners have reported that the attainment of a residual gold loading on carbon of up to 50g/t is desirable (Davidson & Schoeman, 1991; Rogans, 2012). Well operated plants have been reported to exceed this performance level (Davidson & Schoeman, 1991).

It was assumed that the base metals were all removed during the acid wash after adsorption and only gold, copper and silver would report to the elution and electrolysis circuits (Marsden & House, 1992; Rao & Misra, 2004; Steyn, 2010; Rogans, 2012). The lead chloride fines were considered to have been washed out to effluent during the acid washing of the loaded carbon. No lead reported to the elution circuit.

4.2.5.7.   Carbon reactivation

Fouling

Foulants can either be organic and inorganic in nature. The inorganic foulants originate from the process water used, the raw materials, iron particles derived from the grinding media and calcium salts.

Organic contaminants originate from the flocculants that may be used in upstream processes and leaks associated with lubricant oils used on moving parts of the process equipment upstream of the adsorption circuit.

Inorganic contaminants adsorbed or deposited onto the carbon are removed during the acid washing stage. The organic foulants however do not respond to the acid washing. They are removed during the thermal reactivation stage of the stripped carbon.

Reactivation

Three options exist for the operating configurations associated with the reactivation of activated carbon. These are outlined below in Table 4.19 with the attendant advantages and disadvantages (Marsden & House, 1992).

Table 4.19: Operating configurations available for reactivating the stripped carbon (Marsden & House, 1992)

Operating configuration	1	2	3
Profile	Acid wash Desorption Thermal reactivation	Desorption Acid wash Thermal reactivation	Desorption Thermal reactivation Acid wash
Advantages	Removal of inorganic foulants. Removal of calcium to minimise carbon loss as $CaCO_3$ during thermal reactivation at 750^OC. Reducing scaling and deposition in the circuit. A cleaner PLS reports to the electrowinning circuit.	Avoids corrosion effects in the desorption circuit. Removal of inorganic contaminants and excess calcium before thermal reactivation.	Avoids corrosion of elution column. Avoids corrosion of thermal reactivation equipment from the presence of residual Cl^- ions.
Disadvantages	Corrosive environment in the elution circuit Residual Cl^- attacks thermal reactivation equipment downstream. Risk of chloride carryover to the electrowinning circuit.	Higher load of base metal complexes are carried over to the electrowinning circuit. This profile is unsuitable for feed sources with a high base metals concentration. Residual Cl^- ions attack thermal reactivation equipment downstream.	Higher load of base metal complexes is carried over to the electrowinning circuit. This profile is unsuitable for feed sources with a high base metals concentration. Higher carbon losses as calcium carbonate $CaCO_3$.

For the process under study the feed was noted to have a high concentration of copper as Cu^{2+}. WPCBs have higher base and precious metal concentrations as well as heavy metals than their native ores (Hagelüken, 2006). The first operating configuration for carbon reactivation was selected for the process under study. It was also assumed that the total residence time of the carbon in the reactivation circuit would amount to 85 minutes. This time was distributed as follows in Table 4.20:

Table 4.20: Carbon reactivation time profiles (adsorption resource handbook)*

Operation	Dewatering eluted carbon, pre-drying and charging	Thermal reactivation (hot combustion air)	Quenching and discharge to main adsorption circuit
Time (minutes)	45	15	25*

Carbon losses

Six major sources of carbon losses are associated with gold leaching operations (Marsden & House, 1992). These sources include: Carbon-solid attrition in slurry systems, carbon attrition, carbon-steel attrition, breakage associated with carbon transfer, breakage due to thermal shock and chemical shock also associated with the removal of carbon-inorganic composites.

The losses and proportions of losses associated with each of these mechanisms are unique to a plant (Marsden & House, 1992). Experiential evidence however suggests that the bulk of the carbon losses are associated with regions of high mechanical activity and the effects associated with thermal processes (Marsden & House, 1992). The precise measurement of carbon losses has been a difficult task dependent on continuous monitoring and processing. No unique relationship can be ascribed to a unit operation and the rate of carbon loss.

Davidson & Schoeman (1991) reported losses that varied between 23g/t and 263g/t. These findings were condensed to an average of 69+/-59 g/t in other studies (Davidson & Schoeman,1991). The researchers concluded that fresh carbon directed to make up ranged between 3% and 5% of the total carbon loading required.

Carbon losses are strongly influenced by the carbon concentration used in the adsorption circuit (Stange, 2009). Other researchers have used a linear correlation based on experiential data from industry and empirical work. Using a linear loss rate of 0.02% resulted in estimated losses of between 30g/t and 70 g/t for the processes studied by van Wyk (2014). Marsden & House (2006) noted that carbon losses in CIS circuits were less than those observed in CIP circuits by a margin ranging between 30% and 40%. The losses have been quantified and have been found to range between 35g/t and 100g/t for the different gold adsorption circuits in industry (Marsden & House, 1992; Rao & Misra, 2004). A study of one process showed that gold losses from the adsorption circuits, interstage carbon transfer and elution process cumulated to 53% of the total losses observed.

The process under study specified adsorption of gold from the filtered PLS (CIS). An industry average of 50g/t was reported (de Waal, 2018). It was assumed that 60% of this average would amount to the losses associated with the process under study. A loss of 30g/t was arrived at, which is at the lower end of observed industrial average. Considering that the adsorption was undertaken in solution the estimate seemed fairly reasonable. It was also assumed, using findings of previous industrial work that 53% of these losses would have gold that reported to tailings.

The remaining 47% were associated with carbon losses in the regeneration circuit. Previous research has shown that as the fines move down the adsorption train, changes in the equilibrium occur due to increasingly lower gold tenors (Schubert *et al.*, 1993). The result is a preferential desorption into solution though at rates inferior to those expected during elution. Due to co-current movement of the fines it was assumed that pseudo-equilibrium loading would be less than that observed for the bulk carbon moving up the carbon train. For the study it was therefore assumed that the retained gold on the fines reporting to the barren solution would not exceed 2%. The losses of gold at the carbon regeneration circuit were derived from the gold that remained on the carbon after elution which was assumed to be 1% of the total gold adsorbed (Marsden & House, 1992; Rogans, 2012).

4.2.5.8.   Electrowinning

Reactions proceeding

Four major reactions proceeding during the electrowinning of gold have been reported in literature (Marsden & House, 1992; Steyn, 2004). The reactions are outlined as follows:

The cathode reactions:

$$Au(CN)^{2-}(aq) + e^- \longrightarrow Au(s) + CN^- (aq) \tag{4.22}$$

$$2H_2O (l) + 2e^- \longrightarrow H_2(g) + 2OH^- (aq) \tag{4.23}$$

$$O_2(g) + 2H_2O (l) + 4e^- \longrightarrow 4OH^-(aq) \tag{4.24}$$
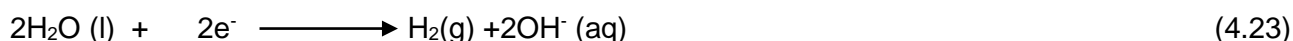
The anodic reaction:

$$2 H_2O(l) \longrightarrow O_2(g) + 4H^+(aq) + 4e^- \tag{4.25}$$
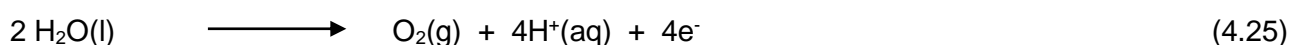
At overpotentials close to or exceeding the limiting diffusion current density the generation of hydrogen is noted to increase (Steyn & Sandenbergh, 2004). It has been noted that reactions 4.22 and 4.23 occur simultaneously.

For the purposes of the study it was assumed that hydrogen evolution did not occur in such a manner as to be detrimental to the deposition of gold. The system was not operated in excess of the limiting diffusion current density. It was assumed that oxygen was evolved at the anode according to reaction described in equation 4.25.

Co-deposition

Co-deposition of silver, mercury and lead occurs during electrowinning of gold. If the silver has a composition greater than 50% of the total metal composition in the pregnant leach solution (PLS) and is greater than the gold concentration, it is preferentially deposited. The gold deposition rate increases as the silver in solution is depleted. In the presence of copper, nickel, zinc and iron, co-deposition will also occur (Marsden & House, 1992; Steyn & Sandenbergh, 2004).

The rate of deposition of base metals increases as time progresses due to lower concentrations of the aurocyanide complex (Steyn & Sandenbergh, 2004). This reduction is followed by a cathodic shift in the reduction potential of the gold half cell reaction (Steyn & Sandenberg, 2004), resulting in the prioritization of base metals removal. Summarised in Table 4.21 are the permitted concentrations of some metals in the gold electrowinning process.

Table 4.21: Specifications for base metals in the gold electrowinning process (Marsden & House, 1992; Steyn and Sandenbergh, 2004)

Metal	Specification	Remarks
Lead (Pb)	<1ppm	Co-deposits with the gold and reduces dore quality of the final product. In high concentrations the complex will be electrodeposited in preference to gold.
Copper (Cu)	<100ppm	Reduces current efficiency. Co-deposits with the gold and reduces dore quality of the final product. The gold to copper ratio must be maintained below 1:5.
Mercury (Hg)		Produces effluent that is complex to handle. In high concentrations the complex will be electrodeposited in preference to gold.
Iron (Fe)		Co-deposits with the gold and reduces dore quality of the final product. Refineries require Fe^{2+}/Fe^{3+} concentrations to be less than 2%.
Nickel (Ni)		Co-deposits with the gold and reduces dore quality of the final product.
Chromium (Cr)	5ppm	Co-deposits with the gold and reduces dore quality of the final product. At 100ppm or more, gold deposition ceases.
Silver		In high concentrations silver will be electrodeposited in preference to gold.

The nitric acid wash achieved the enrichment of the gold composition relative to that of the other metals in WPCB solids. This led to an increased gold concentration in the PLS after cyanide leaching and filtration of the liquor. The mass balances performed around the nitric acid wash circuit showed that the ratio of gold to silver was not less than 8600:1 (Appendix F). The concentration of the silver relative to other metals in the PLS, was less than 1%. The base metals that reported into the cyanide leaching circuit were in the form of soluble nitrates derived from the retained moisture after filtration of the wash feed.

The residual concentrations of these metals formed weak acid (WAD) cyanide complexes. It was assumed that the base metals were all removed during the acid wash after adsorption and only gold, copper and silver would report to the elution and electrolysis circuits (Marsden & House, 1992; Rao & Misra, 2004; Rogans, 2012; Steyn, 2010). Since no mercury was present in the original analysis of the WPCBs this was not considered a problem in the process.

Research has shown that the gold to copper ratio of 1:40 favours the preferential deposition of copper over gold (Steyn & Sandenbergh, 2004). Gold to copper ratios of at least 1:5 were shown to produce highly adherent gold cathode deposits. This would result in significant gold lockup in the electrowinning circuit (Steyn & Sandenbergh, 2004).

Spent electrolyte

The spent electrolyte liquor can be handled in three ways: disposal after detoxification, recycle to leach or recycle to elution. An inspection of the mass balances around the electrowinning circuit revealed that recycling of the spent electrolyte would reduce the fresh sodium cyanide usage in leaching by 75%. The spent electrolyte could also be reused in leaching thereby reducing fresh water feed by a margin of 100%. The sodium load would however be increased in manner favouring the existence of $NaAu(CN)_2$ species as opposed to $Ca(Au(CN)_2)_2$. The third option provided for the recycling of the spent electrolyte to the elution section. The spent electrolyte contained 2.61ppm of the gold complex (Refer to mass the balances in appendix A, Table A.4). The spent electrolyte was polished in the scavenging cells prior to reuse in the process (see section 7.3.2) for the detailed handling of the gold recovery streams). For the process under study it was assumed that the spent electrolyte was recycled to the elution section. Streams of spent electrolyte not recycled join the gold recovery circuit.

Product quality

The quality of the product obtained is also governed by the applied cathode over-potential. High over-potentials are desirable in cases where impurity concentrations that co-deposit with gold are high. The sludge produced can then be refined further. Low over-potentials produce gold from solutions with lower contaminant concentrations. The product tends to have more compact gold precipitates (Marsden & House, 1992; Steyn & Sandenbergh, 2004). For the process under study the attainment of a less compact weakly adherent product was desirable. It was assumed that the combined effect of the applied overpotential and high gold to copper ratio (8600:1) would yield a loosely adherent product. Varying the over-potentials applied contributes to the varied recovery of the gold in the form of solid gold cathode or sludge. It was also assumed that very large overpotentials were avoided to suppress the generation of hydrogen and the generation of hydroxyl ions at the cathode. The production of hydrogen at the cathode would retard the gold deposition rates (Steyn & Sandenbergh, 2004).

Two major reactions were assumed to predominate for the system under study, namely: the reduction of gold at the cathode and the oxidation of hydroxyl to oxygen at the anode (de Waal, 2018). Having considered the high gold to silver ratio that was in the electrolysis circuit, a lower over-potential was specified for the process.

Based on the mass balances done in a separate study, during electrowinning an overall efficiency of 99% was assumed for both gold and silver (de Waal, 2018). It was also assumed that the final product generated from the electrowinning of the PLS was rich in gold with a copper, silver and PGMs concentration of less than 2%. The bulk of the product was therefore assumed to be recovered in the form of a rich gold cathode with a minimum purity of 98%.

While calculations the based on the mass balances indicate a product purity of 99.98%, 98% has however been assumed for the economic study to factor in the presence of palladium and platinum which are associated with diverse WPCB feedstocks for practical purposes.

Two major options were considered for handling the final product. The options are outlined in Table 4.22.

Table 4.22: Final handling of the gold cathode produced

Option and wash method	1 (Pyrometallurgical smelting)	2 (Electrorefining)
A (Acid wash)	1A = acid wash and smelt	2A = acid wash and electro-refine
B (Water wash)	1B = water wash and smelt	2B =water wash and electro-refine

For the process under study it was assumed that the major contaminants were silver, copper and PCMs in concentrations not exceeding 2%. A water wash prior to smelting the gold cathode (Option 1B) was selected. The product produced was assumed to be of a grade suitable enough to be sold to Rand Refinery.

Solution face velocities

Typical solution face velocities are maintained in the range of $250L/m^2$ to $500L/m^2$ at the cathode (Marsden & House, 1992). Industrial experience has recommended running the electrowinning of gold cyanide solutions at pH values greater than 12.5 (Marsden & House, 1992). For the process under study a pH of 13 was specified (de Waal, 2018). At this pH it is expected that the generation of hydrogen is suppressed. Van Wyk (2014) developed process flow sheets for the recovery of PGMs and gold from low grade ores.

A residence time of 5 hours was used for the simulation studies (van Wyk, 2014). It was reported that high gold recoveries in a non cyanide electrolyte are achievable in electrowinning times in excess of 4 hours (van Tonder & Sole, 2006). Briggs (1983) reported a residence time of 12 hours for test work done at Beisa mine using a Mintek cell. It was noted that this electrowinning time translated to approximately 4 passes to achieve a gold recovery in excess of 95%.

For the process under study it was assumed that a solution flow rate of $500L/m^2$ was used at the cathode. In the previous studies a total of 6 cathodes per cell were specified (de Waal, 2018). A single pass efficiency of 60% was assumed (de Waal, 2018).   A residence time of 5 hours in the electrowinning section was used for the purposes of this study. This residence time is equivalent to two passes of the electrolyte solution. Using cathodes with an area of $0.31m^2$/cathode (Briggs, 1983) and a face velocity of $500L/m^2$ it was estimated that 6 cells would be used for the electrowinning process.

4.3.   Effluent, emissions and solid waste management

The mass balances and detailed process flow diagrams for effluent management were not included within the scope of previous studies (Rossouw, 2015; de Waal, 2018). The process however is associated with different unit operations generating varied effluent and solid waste streams that could be handled as by-products for sale, reuse in the main process, toxic materials in need of pre-treatment and special storage and objectionable materials in need of conditioning prior to disposal in the main municipal discharge stream.

A consideration of the varied destinations of the output of effluent handling therefore necessitated a brief discussion pertaining to the handling, characterisation and management of the different effluent and solid waste streams associated with the operations under study.

4.3.1.   Emissions management

Following an inspection of the preliminary mass balances, a comparison against recommended emissions and occupational health and safety guidelines was undertaken. Using the results of the comparative inspections, refinement of the original process was done by addition of the relevant equipment to handle the emissions generated. This is reflected in the equipment flow sheets included in appendix E. All the emissions predicted to be within safe operating levels with respect to the recommended guidelines were vented, as reflected in the original process flow sheets (de Waal, 2018). Summarized in Table 4.23 is the summary of the expected emissions generated from the operations and the applied management method.

Table 4.23: Emissions management in the respective plants

Plant/ section	Emissions	Key aspect	Volumes/ mass generated	Source for calculations	Recommended guidelines (MSDS sheets for materials, USA OSHA, EU (2005))	Intervention
Size reduction (Crushing plant)	Fugitive dust	Ceramic, plastic particulates with metals	43.71kg/hr	He *et al.,* (2015) de Waal (2018)	Permissible levels 350µg/m^3 per hour.	Equipment specified included dust socks to capture the fines generated. It was assumed that the dust recovered from the socks was shaken into the main solids hopper. Within the building further polishing of the operating air space was achieved through the use of dust extractors. It was also assumed that the formation of stable aerosols was not applicable to the process under study.
Solder leaching	Noxious fumes	NO	2.06kg/hr	Rossouw (2015) de Waal (2018)	30mg/m^3	Extraction fan included. Vented to atmosphere.
		NO$_2$	0.59kg/hr	Rossouw (2015) de Waal (2018)	50µg/m^3 over 24 hours	Extraction fan included. Vented to atmosphere.
		H$_2$	0.01kg/hr	Rossouw (2015) de Waal (2018)	40 000ppm lower flammability limit	Extraction fan included. Vented to atmosphere.
Copper extraction	Gas from leaching and electrowinning, volatile fumes from solvent in the solvent extraction circuit.	H$_2$	0.0001kg/hr	Rossouw (2015) de Waal (2018)	40 000ppm lower flammability limit	Vented to atmosphere.
		O$_2$	ND	Rossouw (2015) de Waal (2018)	NA	Vented to atmosphere.
		Acid vapours	< 1mg/m^3	CS	1-3mg/m^3	An additive was used to suppress acid mist formation. This was included in the economic evaluations.
		Kerosene fumes	Up to 18667ppm at 293K	CS	7000 ppm – 50000ppm Occupational exposure limit 10ppm.	It was assumed that the installed facilities incorporated adequate ventilation to manage ambient concentrations. Excessive air currents in the operating area were avoided.

Table 4.23: Emissions management in the respective plants con't

Plant/ section	Emissions	Key aspect	Volumes/ mass generated	Source for calculations	Recommended guidelines (SDS sheets for materials, USA OSHA, EU (2005))	Intervention
Gold extraction	Noxious fumes Hot process vapours WAD cyanide Hot acidic process gases	HCN	At least 0.011kg/hr	Estay (2010) Adams (2016)	11mg/m^3	A minimum generation rate was calculated based on the WAD cyanide expected to be formed. It was assumed that an extraction system with a scrubber would be factored into the detailed design. This was not shown on the original flow sheets (de Waal, 2018).
		HCl	NQ-Variable	Marsden & House (1979) Rogans (2012)	7mg/m^3	Extraction fan included. Vented to atmosphere after scrubbing.
		CO	NQ-Variable	Marsden & House (1979) Rogans (2012)	55mg/m^3 ((PEL) TWA)	Extraction fan included. Vented to atmosphere after scrubbing.
		CO$_2$	NQ-Variable	Marsden & House (1979) Rogans (2012)	9150 mg/m^3 (TWA): EH40 WEL	Extraction fan included. Vented to atmosphere after scrubbing.
		NH$_3$	NQ-Variable	Marsden & House (1979) Rogans (2012)	35 mg/m^3 (TWA)	Extraction fan included. Vented to atmosphere after scrubbing.
		NO	0.009kg/hr	CS	30mg/m^3	Generated during the nitric acid wash. Included an extraction fan and a condenser. Gases not absorbed were vented to the atmosphere.
		NO$_2$	2.20kg/hr	CS	50µg/m^3 over 24 hours	Included a scrubbing system to recover the fumes as weak nitric acid (10%).
		Hot acidic vapours	26.36kg/hr	CS	2 ppm (TWA)	Generated during the nitric acid wash. Included an extraction fan and a condenser. The option for preheating the acid wash water and elution water was considered.
		O$_2$	0.008 kg/hr	de Waal (2018)	NA	Vented to atmosphere.
		H$_2$	0.0005 kg/hr	de Waal (2018)	40 000ppm lower flammability limit	Vented to atmosphere.

4.3.2. Effluent management

The effluent streams from the different sections of the plant and the respective business models derive from the original process flow sheets, were analysed and summarized on an annual basis. The distribution of the effluent flows for each business model is depicted in appendix F, Table F.26.

4.3.2.1. Municipal and general housekeeping

The effluent derived from this source is largely associated with typical municipal waste water and general housekeeping activities. The housekeeping actives include the cleaning of offices, changing rooms and canteens. Effluent from this source is usually disposed of in the regular municipal effluent reticulation system. No material balances were incorporated for this effluent. In the study however no Figures were captured for this effluent stream since the volumes vary in a manner that was not captured in the material balances.

4.3.2.2. Size reduction

Any effluent from this section of the plant was assumed to be associated with discharges derived from general plant housekeeping. It was expected that this effluent would be associated with clay particles, lubrication oils, heavy metals, base and precious metals of varying and minute quantities relative to the composition in the feed source. The source of the metals and non-metallic fractions were assumed to arise from extremely fine particles capable of forming dust aerosols. It was also assumed that the formation of stable dust aerosols was negligible.

The main objectionable features of this stream were the potential presence of lubrication oils and heavy metal dust. The possible inclusion of an oil sump to capture the oils prior to exporting the water to the main water treatment plant was assumed. In the study however no figures were captured for this effluent stream since the volumes vary in a manner that was not captured in the material balances.

4.3.2.3. Solder leaching process

Effluent streams generated from the solder leaching processes were characterized by low pH, high TDS, high lead and iron concentrations. As a result of recycling the wash water from the gold extraction plant, the nitrate, silver and base metals content increased in this stream. Potential exists for recovering these metals and silver through precipitation as sulphides or concentration using ion exchange. Neutralization of this effluent stream with calcium hydroxide would result in the precipitation of hydroxides of the base metals. Following the thickening of the solids, the supernatant could be recycled in the other areas of the factory for general plant use.

4.3.2.4.　　　　Copper extraction process

The effluent from this circuit was expected to be acidic with organics derived from the solvent extraction circuit. It was assumed that at least 97% of the solvent would be recovered by use of a centrifuge and clay treatment. The remaining 3% was assumed to be locked up in clay solids to be disposed of in tailings the pond. Solvent losses that reported to the effluent stream would be treated by carbon adsorption prior to joining the main effluent water discharge flow.

Literature reports that spent electrolyte is evaporated in commercial operations to crystallize nickel and iron sulphide. Neutralization of the effluent, reduction of TDS and scavenging of copper, was also considered a key objective in the treatment of the copper extraction effluent streams. The main effluent streams associated with the copper extraction circuit were the spent electrolyte from the electrolysis bank and an acidic aqueous phase.

The aqueous phase recovered from the centrifuge was assumed to carry the residual solvent which was redirected into the main process. It was assumed that this stream was treated with bentonite clay to adsorb and immobilise the solvent. Wang (2005) described in detail the destabilizing effect bentonite clay has on the organic phase of crud. Industrial practice recommends a clay concentration averaging between 2wt% and 5 wt% in the stream being treated (Crane *et al.*, 2009). A clay concentration of 2 wt% was specified for the process. It was assumed that all the organic was immobilised and the polished stream was redirected to the leach circuit.

Potential exists for the installation of a unique electrowinning circuit to recover the copper in the bleed using EMEW cell technology suitable for low copper feed concentrations of 0.1g/L (Roux *et al.*, 2007). The barren leach solution based on mass balances was calculated to have a copper concentration greater than 30g/L. For the study EMEW cell technology was assumed to be incorporated for the polishing of copper rich effluent streams.

4.3.2.5.　　　　Gold extraction process

Effluent streams from the gold extraction operations under study were derived from the in process spillages and outflows of water to the plant drainage from general housekeeping activities. Spent leach solutions, used acid wash solutions, filtrates, process blow downs were also assumed to be the chief sources of effluent flows from the plant. It was expected that these flows were characterized by high total dissolved solids (TDS), nitrate, calcium, sodium, chloride, cyanide, hydroxyl, nickel, iron, copper, silver and lead. The pH of the streams was assumed to gravitate between two extremes of high acidity and high alkalinity, arising from the hydrochloric acid wash prior to elution and the electrolysis banks respectively. The data has been summarized in appendix F, Table F.26.

The general nature of the effluent from the gold process also revealed that there would be need to reduce the TDS levels, potentially recover the base metals, correct the pH and reduce or destroy the cyanide present. The presence of cyanide was noted to be the most objectionable feature of this effluent stream. It was therefore necessary reduce the cyanide levels in the effluent or else consider options for returning barren leach solutions into the process circuit. The major flow of cyanide was derived from 75% of the spent electrolyte that was stored for further leaching campaigns. A consideration of the generation rates of spent electrolyte at steady state operations revealed that the inventory would increase faster than its reuse.

A number of options exist for handling cyanide effluent from the process, described in depth by Marsden & House (1992). The options include:

- Reuse of some of the barren solutions and tailings
- Acidify, volatilize and re-neutralize (AVR)
- The use of ion exchange resins to recover soluble metal complexes
- Electrolytic treatment
- Ion precipitation flotation

Detoxification

Dilution- fresh solvent can be added to the spent liquor prior to recycling it back into the process. A portion of the bled effluent is redirected into the process. This quantity must not interfere with the performance of the process. Marsden & House (1992) have also discussed options that include the treatment of effluent for disposal, treatment of effluent for reuse in the process, treatment for reuse and bleeding a portion followed by blending with fresh solvent.

Detoxification- this can be achieved by the formation of cyanite. Research indicates that 10% to 15% of the cyanite undergoes hydrolysis to form NH^{4+} and CO_3^{2-}. Three methods exist for the detoxification of cyanide laden streams and include cyanide removal, oxidation to less toxic cyanate species and the complexation to less toxic cyanide complexes species of (Fe^{2+}). A detailed description of each option is available in literature (Marsden & House, 1992)

A likely option for the process under study would be recycling some of the effluent streams with high TDS values to precipitate double salts and to pass the effluent through an activated carbon bed to catalyse the oxidation to cyanite. Alternatively, the detoxification and potential evaporation of the liquor can to be considered as an option to reduce the volumes being handled prior to returning the spent electrolyte into the process.

The options presented for handling the effluent streams above can be explored in greater detail as the process is refined in separate studies. For the study RenoCell technology was applied to electrolytically recover the residual metal values in the tailings and effluent streams.

4.3.3. Solid waste management

Table 4.24 details the expected solid waste outflows originating directly from the process. A summary of the key features and destinations of the solid material is outlined in the Table. The volumes and characteristics of the solid waste outflows are summarized in appendix F, Table F.27

Table 4.24: Solid waste streams generated in each section

Section	Solids outflows	By product feed for next process	By product for potential exploitation	Waste for special treatment and disposal
Municipal and general house keeping	Canteen waste Office waste Dunnaging and packaging material from bulk handling of raw materials and chemicals used in the process.	N/A	Yes- Plastic, styrofoam, steel, cardboard for recycling	Yes – some of the material will contain toxic or corrosive substances like caustic soda flakes, sodium cyanide, cobalt sulphate.
Size reduction	Crushed WPCBs	Yes	N/A	No
Solder leaching	Desoldered WPCBs dewatered cake	Yes- raw material for copper extraction circuit	N/A	No
Copper extraction	Dewatered WPCBs stripped of copper	Yes- raw material for gold extraction circuit	N/A	No
	Spent clay	No	No	Yes – detoxify. The clay is impregnated with acidic solvent.
Gold extraction	Dewatered WPCBs stripped of gold (Palladium and platinum are also stripped though this was not included in the scope of previous and the current studies)	N/A – no provision for recovery of REE in this study	Yes- Contains plastic fractions that can be treated as feedstock for other recyclers. Ceramics can be used as feedstock for construction material. Feedstock for recovery of REE and other PGMs	Yes – At least 22kg of gold can be recovered each year from the solids. Washing of solids to recover diaurocyanide is required. Further detoxification and neutralization of solid cake prior to disposal or use for further processing will follow.
	Carbon fines from attrition losses in the gold circuit.		These fines were assumed to have 1% of the gold adsorbed on them. These fines can potentially be blended with the leached solids from the gold leaching plant that contain unleached PGMs	Fines in need of detoxification.

4.4.   Chapter summary

A detailed discussion of the operating conditions and process parameters affecting the performance of the proposed design was undertaken. An outline of the assumptions and data used to derive the mass and energy balances for each plant was provided. The output of the mass and energy balances built from previous studies (Rossouw, 2015; de Waal, 2018), was used to feed into the development of the production plan and sizing of the equipment in order to carry out an evaluation of the capital and operating expenditures for the detailed economic study covered in chapters 7 and 8.

An analysis of the characteristic feed stream reporting to the gold section was undertaken in which it was established that the base metal composition was high. An evaluation of possible measures to manage the concentrations of the base metals reporting to the gold electrowinning circuit was done. Two treatment options were analysed in detail. The options studied were the SART process (post cyanide leaching treatment) and the nitric acid wash process (pre-cyanide leaching treatment).

Pre-treatment of the feed prior to cyanide leaching was selected as the most suitable option for the scope of this study. This resulted in a revision of the gold extraction process flow sheet and the material balances around the gold circuit. Following this intervention it was established that the gold head grade to the adsorption circuit was increased. This increase necessitated a consideration of three options to cater for the increase namely: reducing the solids to liquids ratio in the leaching circuit, increasing the carbon concentration or diluting the liquor reporting to the adsorption circuit to match the original process design conditions. Having considered the operating constraints within the gold extraction circuit upstream and downstream of the adsorption circuit and the potential gold losses associated with increased carbon concentration, dilution was selected as the more feasible option.

The output of this chapter also created a foundation for predicting and characterizing the nature of the effluent flows, emissions and solid waste streams associated with the plants. The detailed study around the emissions, effluent and solid waste management was not included in the scope of the study. Heuristics based on previous studies were instead used to estimate the costs associated with the management of these streams in chapter eight (Turton *et al.,* 2009; van Wyk, 2014).

Other areas for further study were also outlined in this section and include the development of detailed Pourbaix diagrams for multi-component systems associated with WPCB leach chemistry, the use of EMEW electrowinning technology to recover the copper in the bleed using EMEW cell technology suitable for low copper feed concentrations of 0.1g/L (Roux *et al.*, 2007). Further upgrading of the process should be considered in the flow sheets should the project economics be found to be favourable.

Chapter 5:   CONSIDERATIONS FOR PROJECT ECONOMIC EVALUATIONS

5.1.   Purpose of economic evaluations

Economic evaluations are useful for the assessment of the economic impact of any business concern such as recycling operations, diagnosing bottlenecks in the entire recycling operation and characterising the financial attributes of a proposed or existing process. By making use of the output of the evaluations, relevant adjustments can be made in tandem with recommendations derived from environmental and sociological impact assessments.

Economic evaluations allow crucial investment decisions to be made. They provide a broader picture of the financial direction a venture will be likely to assume in the short or long term. The responsiveness of a minor or major alteration to the configuration of a production concern can be predicted in a manner that incorporates the detailed assessment of the impact of changes in the market.

Market dynamics are a part of macroeconomic factors that behave as negative or positive catalysts to making investment decisions. The degree of financial risk can be predicted so that appropriate contingencies can be incorporated in the planning stages to provide a cushion against unforeseen shocks emanating within a concern or the macroeconomic environment.

It must be noted however, that while rigorous analysis of the operating environment is undertaken through economic evaluations, it is not a fool proof measure of the certainty against severe changes within the business environment. The shocks associated with the operating environment are normally addressed in an analysis of strengths, weaknesses, opportunities and threats (SWOT) or other environmental assessment tools. Such shocks can include adverse commodity price dips or increases.

5.2.   Important considerations for the evaluations

5.2.1.   Raw material procurement

In the scope of this study categories two, three and four electronic waste as defined by the EU Directive guidelines for categorization of e-waste (see Table 3.1) were used as the raw material supply. The gold content in WPCBs averages between 110g and 250 g per tonne for the e-waste categories specified for this study (Hagelüken, 2006; Lydall *et al.*, 2017). Pricing and grading of WPCBs is not standardized. Typically it is sold in the raw unprocessed form to maximise returns. Table 5.1 outlines the typical price structures prevailing for whole WPBC boards.

Table 5.1: WPCB grading and pricing (Lydall *et al.,* 2017)

Grade of WPCB	Low	Medium	High
Description	High copper content, low precious metals derived from large household appliances.	Pins, edge connectors and capacitors contribute to the value of these boards.	Highly populated boards from IT equipment. Highest grades are sourced from older electronic equipment
Price (US$/kg)	0.10-1.02	1.36-2.04	2.72-5.78
Competition for raw materials	Sale to agents and metal merchants for recovery in Asia and Europe. Global co-processors namely Belgian, Swedish and Canadian concerns.	Domestic pre-processors Global co-processors namely Belgian, Swedish and Canadian concerns.	Domestic pre-processors

5.2.2.   Philosophy of calculations

The profitability and all financial calculations and production rates were determined using a pessimistic calculation to incorporate risk factors within the micro and macroeconomic environments.

5.2.3.   Key raw material supply data

The calculations were done using a WPCB recycling rate of 400tonnes per annum. In an earlier discussion (section 3.6.1), a major supply risk due to low e-waste inflows into the e-waste recycling economy was identified. The risk has potential to creep adversely into the operations of existing downstream players who are likely to respond by tightening supply contracts to protect their niche.

It was assumed for practical purposes, that raw material to be accepted included a combination of white and brown goods. These provide WPCBs that vary in grade. Discrimination of the source of the WPCBs was not done due to the perceived supply risk.

In chapter 3 again it was noted that 3% to 5% of the e-waste weight is comprised of WPCBs (Schluep *et al.*, 2009). Widmer *et al.,* (2005) reported an average composition of 1.7% of WPCBs in e-waste. It was also noted that WPCBs from end of life vehicles contribute on average 0.11% of the total weight (Cuchiella *et al.,* 2016).

According to literature the proportion of recycled waste streams in South Africa comprising of e-waste is 12% (DEA, 2012; GreenCape, 2017). For the purposes of the study the maximum average of WPCBs weight in raw e-waste discussed by Widmer *et al.*, (2005) was used in the calculations. Based on estimates of the proportion of WPCBs in e-waste the requisite tonnage of WEEE to be committed to this plant per annum is equal to 23 530 tonnes (Widmer *et al.,* 2005). In the study it was established that almost 25% of known registered e-waste recyclers process e-waste each year.

The tonnage of e-waste required to recover 400 tonnes of WPCBs each year was estimated to be 23 530 tonnes. This means that the major e-waste recyclers can supply up to 64% of the material requirements of the proposed venture. Thirty six percent of the required e-waste would have to be furnished by the informal sector and the remaining 75% of formally registered recyclers. The possibility of additional costs of acquiring WPCBs cannot be ruled out a scenario where the remaining 36% cannot be supplied consistently.

The collection of materials also has inherent business models depending on the prevailing dynamics and influences associated with the efficiency of guaranteeing adequate material volumes for operations.

Three have been identified and include:

- Incentivised collection (full cost of reverse logistics wherein the recycler incurs the cost of buying back and collection of the e-waste),
- Secondly controlled incentive (recycler collects and offers a scrap value for the items being released for recycling),
- Finally the non-incentivised scheme where the recycler enters into a strategic business partnership with a known proprietor that acts as a collection point for e-waste.

The major cost is associated with the collection of the bins and paying agreed levies for the bins. The last collection model has the advantage of advertising itself due to non e-waste related activities which draw consumers, such as shopping. For the purposes of this study it was been assumed that e-waste would be collected from major collection points or delivered by small collectors to the central collection points.

5.2.4.   Tax considerations

The tax rate applicable to the scale of business operations used. Three major tax bands have been specified by the South African Revenue Authority: large corporate, venture capitalists and small to medium scale enterprises. In the study a tax rate of 28% was used applicable to small for medium scale enterprises.

5.2.5.   Depreciation

Depreciation can be regarded as a tax allowance, cost of operation or measure of the declining value of an asset or a mechanism to build up an equipment replacement fund (Peters & Timmerhaus, 1991). For preliminary cost evaluations straight line depreciation is often used. The choice of depreciation method used in later stages of the project is subject to the relevant tax regulation authority within the country. An average useful service life of 10 years was used for each of the business models analysed.

5.2.6. Interest rate (cost of capital)

Power (2004) in a presentation at the OECD development seminar defined the cost of capital as being the function of an organization's capital base, the equity and debt. An expression can be derived for determining a weighted average cost of capital (WACC). This is achieved by taking into account a beta value for determining the contribution of equity to the cost of capital and specifying the equity risk premium. This WACC combines risk free lending rate, lending margin and the equity risk premium and prevailing tax rate. The weighted average cost of capital can be regarded as a determinant of the true economic value of a project wherein discounted cash flows at the end of a project life at a specified internal rate of return (IRR) are targeted to be positive.

If the IRR is less than the hurdle rate, the difference between the returns to capital and cost of capital will be less thereby yielding an unprofitable investment. The WACC is a reflection of the extent of gearing. The hurdle rate is the price of capital and is that discount rate used to determine the financial viability of a project, commonly the interest rate defined 'i'. The risk adjusted hurdle rate using capital asset pricing model is calculated as follows:

WACC = equity% of capital base (risk free rate) + (Beta x ERP*(Equity risk premium)) + debt% of capital base (risk free rate + lending margin) x (1- tax rate)
(5.1)

As illustrated in equation 5.1 it can be seen that as the tax rates for venture capital in South Africa increase and the degree of gearing increases, the WACC decreases. The risk of the project is however increased as the project is more prone to disturbances in the macro-economic environment. Such a scenario motivates lenders to add a premium to cushion against increased risk through the lending rates (Power, 2004). In South Africa an ungeared company has a WACC of 14.9%.

Presently it has been shown that large concerns that can be potential investors listed on the London Stock Exchange (LSE) as opposed to the Johannesburg Stock Exchange (JSE), have lower discount rates. This leaves two options; foreign direct investment or targeting local companies with foreign listings in capital markets that have a deeper appetite for risk and on a supply side a greater breadth (a diverse plethora of investment options).

Depending on the company's asset base and listing, the prevalent WACC in major South African companies is reported to range between 9.5% and 15%. Ghodrat *et al.,* (2016) conducted an economic feasibility study of the large scale pyrometallurgical processing of WPCBs through the black copper smelting route within an Australian context and made use of a comparatively lower cost of capital of only 10% compared to South Africa's average of 12%.

According to the South African reserve bank statistics, the prime lending rate by private banks has been pegged at an average of 10.25% since December 2012 (SA Reserve Bank Statistics, 2018).

It has been assumed that the cost of capital in South Africa due to equity is on the higher side. Using a debt to equity ratio of one, a hurdle rate of 12.6% was specified for the economic study. The arithmetic mean of the ungeared and current prime lending rates was used.

Depending on the degree of risk associated with the project, the hurdle rate can be adjusted upwards or downwards. It is envisaged that low lending rates can be expected from strong concerns with a strong leverage to finance their activities using debt capital or concerns with off shore holdings. The higher cost of capital may be expected in regions where high risk is expected with the project or from financiers with a lower appetite to carry risk. The hurdle rate specified was also assumed to remain constant for the entire project life. Adverse changes in the business environment were assumed not to apply within the scope of the study.

5.2.7.   Adjustments for inflation

The degree of accuracy of project profitability estimations is subject to the reliability of considerations made in the calculations that include inflation, interest on capital, prevailing tax regulations and provisions for depreciation. Sell (1997) described the major components that embody inflation accounting that are necessary for producing the balance sheet. Inflation accounting takes into account the stock valuation methodology, depreciation method (typically based on the historical costs of the assets) and taxation of the inflation premium.

In seeking project finance, development banks and other finance institutes require this level of detail. Sell (1997) has argued that investment appraisal in this light is independent of inflation accounting.

Two types of inflation exist that are the output of political, global economic changes and government (domestic and foreign policies). Perry *et al.*, (1997) described two distinct types that include differential and general inflation.

General inflation is characterised by a uniform increase in prices and all costs. Differential inflation shows a variation in the rate at which costs and prices increase. This is a result of different factors exerting varied influences on pricing structures such as government controls. Inflation does not remain constant every year.

Perry *et al.*, (1997) has noted that the trend has been to neglect the effects of inflation, a posture that echoes remarks advanced by Sell (1997). Sell (1997) noted that project appraisal in regions burdened by high inflation rates are undertaken by use of real and constant prices. It has however been shown that both the general and differential rates influence the outcome of the payback period (PBP), net present value (NPV) and discounted cash flow rate of return (DCFRR). Failure to account for inflation during project investment appraisals manifests in an exaggerated profitability being reported (Perry *et al.*, 1997).

Further, the tax reported for remittance to the government is erroneous. The inherent perceived benefit of differential inflation where goods and services are inflated at a fixed cost of capital an increase in NPV is realised. This is the inherent perceived benefit of differential inflation as is the case of general inflation (Perry *et al.*, 1997). Higher profit margins are associated with higher DCFRR and shorter payback periods (PBPs) that are misleading.

The South African reserve bank announced that the benchmark repo rate has been pegged at 6.75%. The result is that the average 2017 reported inflation rate was recorded at a rate of 5.3%. The 2018 Figure declined to 4.9% while the 2019 forecast registered a climb to 5.4%. This reports an average of 5.2% over the three years.

A mean inflation rate for the period 2012 to 2022 was used in this study. Statistica.com (2018) reported a mean inflation rate of 5.56% for the aforementioned 11 year period embracing historical and future estimates. The discounted cash flows did not include inflation. The effects of inflation were considered in the sensitivity analysis. It has also been assumed that significant hyperinflationary effects do not exist,

5.2.8.   Utilities

Standard prices for municipal water and power for purchased utilities were used. Specific usage rates were evaluated from the usage ratios determined from the mass and energy balances (de Waal, 2018). The costing of soft water and saturated steam at 2.70 bars was done based on the output of calculations used for rating the water softener unit and evaluating the need for a boiler. The detailed boiler and softener sizing and material requirements are discussed in section 5.3.2.2.

5.2.9.   Metal prices

London metal stock exchange and Infomine prices were used to evaluate the expected revenues from the sale of the finished products and ascribe a value to intermediate products pending further processing (Infomine, 2017; LME, 2018; 2018b).

5.2.10.   Material properties and losses

For the purposes of the preliminary evaluations it has been assumed that no alloys of copper exist in the feed material. Standard losses of material and reagents have been determined from experiential calculations based on industrial heuristics described in previous and current studies alluded to in chapter 4. The average analysis of WPCBs used for the development of the process flow sheets in a previous and separate study was based on the work of Shuey & Taylor (2004).The gold content was revised downwards to 250 ppm to match the prevalent averaged supply of medium grade WPCBs in the country.

Palladium was not captured in the original studies which have formed the basis for this study (Rossouw, 2015; de Waal, 2018). The contribution towards potential revenue from palladium was however considered in chapter 7.

5.3.   Operations management

5.3.1.   Equipment sizing and planning

Equipment sizing was done based on the output of the mass and energy balances that were developed in a separate study (de Waal, 2018). Equipment and building facilities were assumed to have been sized to 120% the design capacity. All bunding for liquid storage tanks has been sized to accommodate 110% of the maximum volume contained in the storage vessel. These were updated to incorporate the relevant recycle streams. Refer to appendix A for the mass balances performed for the process plants. Appendix B provides details of the mass balances used for developing the nitric acid wash circuit.

Results from previous empirical work and industrial performance data were used to define the operating time profiles for each section. The schedule of residence times used in identifying process bottlenecks is summarized in Table 5.2 extracted from the main equipment list schedule detailed in appendix F, Table F.7. The time profiles also factored in the vessel change over, material transfer times and vessel preparation times.

Table 5.2: Process debottlenecking (extracted from Appendix F, Tables F.7)

Item	Plant	Maximum running time (hrs)	Batch runs/ day	Section	References
1	Size reduction	Continuous	1	Crushing and milling	Allocated based on longest cycle time.
2	Solder leaching	8 hours leaching 1 hour material transfer	2	Leaching	Rossouw (2015) de Waal (2018)
3	Copper extraction	168 hours electrowinning 1 hour material transfer	0.006	Electrowinning	Steyn (2004)
4	Nitric acid wash	5 hour acid wash, 1 hour Material transfer, 1 hour sluicing 2.25hours material transfer 1 hr wash filtration	2 2	Acid wash Water sluicing	As developed. Operation inherited from solids output from copper leaching.
5	Gold extraction	24 hours leaching, 1 hour material preparation and loading, 0.5hours conditioning 1 hour discharge and filtration	1	Leaching	As developed.

Table 5.3 summarizes the equipment sizing considerations drawn from the profile of the process debottlenecking outlined in Table 5.2 above.

Table 5.3: Equipment sizing considerations

Item	Plant	Maximum running time (hrs)	Batch runs/day	Monthly output	Sizing guide lines
1	Size reduction	24	1	30 production cycles	Sized section to produce solids for leaching that covered tonnage of copper to be harvested after 7 days of electrowinning. Simultaneous transfer of solids to the solder leaching plant occurs.
2	Solder leaching	9	2	80 production cycles	Sized main equipment to leach all solids from the size reduction plant within 24 hours.
3	Copper extraction	169	0.006	4.26 production Cycles	Sized leach tanks to absorb all solids from the solder leach within 12 hours. Storage capacity of the filtered leach liquor to handle all the liquor from leach section. The inventory capacity was equivalent to 7 days of continuous production. The SX circuit was sized to handle all the output of the leaching section in 12 hours. The inventory capacity of the leaching circulation tank was equivalent to the PLS liquor storage tank. Volume of liquor handled in electrowinning cells was equivalent to 7 days of continuous production.
4	Nitric acid wash	6.00 4.25	2 2	77.83 production cycles	The central nitric acid storage tank was located in this section. (Sized to feed solder leaching plant at start up and running of the nitric acid wash circuit). Solids buffer hoppers and wash vessels sized to run two batches within 24 hours.
5	Gold extraction	26.50	1	27.17 production cycles at 1000ppm of gold in WPCB feed. 6.79 production cycles at 250ppm of gold in WPCB feed.	Feed hopper sized to handle solids output from the nitric acid wash circuit produced in 24 hours.

These time profiles were used to identify bottlenecks and specify material movements across the plants. The effluent and solid waste flows were discussed in detail in section 4.3. Sizing was of equipment was done with objective of nullifying the influence of the most significant bottleneck. The bottleneck was identified to be the copper electrowinning process.

Based on the time profiles of the different sections, semi batch operations were identified as the predominant operating configurations. To minimize idle times in the respective sections, sizing of the storage vessels was done to maximize continuity of operations and ensure smooth change over while reducing the frequency of plant moth balling. This ensures that the cost of labour per tonne of copper and gold produced is reduced. In order to achieve this objective an analysis of possible production operating configurations was undertaken. Three operating configurations influencing sizing of equipment upstream of the copper plant and downstream of the copper plant were identified.

The first configuration was based on sizing the equipment upstream and downstream of the copper extraction circuit to run 4.26 cycles per month. In this configuration, equipment was sized to the largest capacity possible on both ends of the copper extraction circuit.

The second configuration that could be adopted was associated with sizing the vessels to run at rates designed to accumulate 7 days' worth of process inventory prior to feeding the copper extraction circuit. A similar sizing criterion was applied to the processes downstream of the copper extraction plant.

The third configuration entailed the maximization of equipment sizing upstream of the copper extraction circuit and the minimisation of equipment sizing downstream of the copper extraction circuit. The schedules of production operations in each plant based on the operating configurations are depicted schematically in appendix F, Figures F.1, F.2 and F.3.

For all production configurations it was assumed that the lead time for purchasing all relevant raw materials that are locally available would not exceed one week. Imported raw materials or additives were assumed to have a lead time of one month. Stocking of imported materials was assumed to support a month's operations. Local purchased materials were stocked at inventory levels supporting a week's production. The portfolio of products derived from the operations was based on the business models developed and is discussed in detail in section 6.1.2. The production plans specified for the respective business models are discussed in section 6.2.6. Table 5.4 provides a summary of the implications of each of the three production operating configurations.

Table 5.4: Production operating configurations available

Production operating configuration	Configuration 1	Configuration 2	Configuration 3
Equipment sizing	Based on 4.26 cycles a month	Based on maximum cycles each section will run as specified in Table 5.3.	Based on 4.26 cycles a month
Start up operations	Solder leaching plant goes on line 24 hours after the WPCBs size reduction plant. Copper electrowinning goes on line 179.75 hours after hot commissioning of all upstream sections. The nitric wash acid plant goes on line 179.75 hours after hot commissioning all upstream sections. The gold extraction section goes online 190 hours after hot commissioning of the sections upstream.	Solder leaching plant goes on line 24 hours after the WPCBs size reduction plant. Copper electrowinning goes on line 59.75 hours after hot commissioning of all upstream production plants. The nitric wash acid plant goes on line 59.75 hours after hot commissioning all upstream sections. The gold extraction section goes online 70 hours after hot commissioning of the sections upstream.	Solder leaching plant goes on line 24 hours after the WPCBs size reduction plant. Copper electrowinning goes on line 179.75 hours after hot commissioning of all upstream sections. The nitric wash acid plant goes on line 179.75 hours after hot commissioning all upstream sections. The gold extraction section goes online 190 hours after hot commissioning of the sections upstream.
Steady state running times	Size reduction 1 day a week Solder leaching 1 day a week Copper extraction 1 day a week except electrowinning Copper electrowinning 7 days a week (one cycle) Nitric acid wash plant 1 day a week Gold extraction 2 days a week.	Size reduction 7 days a week Solder leaching 7 days a week Copper extraction 1 day a week except electrowinning Copper electrowinning 7 days a week (one cycle) Nitric acid wash plant 7 days a week. Gold extraction 7 days a week.	Size reduction 1 day a week Solder leaching 1 day a week Copper extraction 1 day a week except electrowinning Copper electrowinning 7 days a week (one cycle) Nitric acid wash plant 7 days a week. Gold extraction 7 days a week.
Raw material inventory management	Minimum inventory 1 week for locally available materials. Minimum inventory 1 month for materials that are not available in South Africa.	Minimum inventory 1 week for locally available materials. Minimum inventory 1 month for materials that are not available in South Africa.	Minimum inventory 1 week for locally available materials. Minimum inventory 1 month for materials that are not available in South Africa.
Man power utilization	Permanent staff. Versatility training for crusher operators to be able to run the solder leaching plant on a three shift roaster. Permanent staff for the copper plant. 3 shift roaster. Permanent staff. Versatility training for nitric acid wash plant operators to be able to run the gold extraction plant can be considered. Process superintendent	Permanent staff. Running Three shift roaster. Permanent staff for the copper plant. Four shift roaster. Permanent staff. Versatility training for Nitric acid wash plant operators to be able to run the gold extraction plant can be considered. Process superintendent and shift foremen	Permanent staff. Versatility training for crusher operators to be able to run the solder leaching plant on a three shift roaster. Permanent staff for the copper plant. 3 shift roaster. Permanent staff. Versatility training for nitric acid wash plant operators to be able to run the gold extraction plant can be considered. Process superintendent and shift foremen

5.3.2.   Utilities

5.3.2.1.   Water softening

Softened water was used in the gold extraction circuit. The determination of the water softener capacity was done using knowledge of the average water quality water assays that were pumped into the main process plant. A crucial parameter in specifying the operational capacity of the softener is the total hardness based on total magnesium and calcium available. Factors were used to convert the calcium ion and magnesium ion concentrations to an equivalent calcium carbonate concentration (Purolite, 2014). Factors of 2.5 for Ca^{2+} ion concentration and 4.18 for Mg^{2+} ion concentration have been specified for use in industry (Purolite, 2014). Based on the calcium carbonate hardness, the usage ratio of salt for both the softening of water and regeneration of the exhausted resin was calculated.

For this study the usage ratio of salt was determined by evaluating the equivalent hardness as $CaCO_3$. It was also assumed that the initial set up would be installed in the Johannesburg metropolitan area. An average water hardness of typical municipal water in the Johannesburg metropolitan area was used to specify the salt usage ratios for softening the water and regenerating the resin (Mohotsi, 2016).

The performance data was compared against the performance of known operations for installed softeners at Zimbabwe Phosphate Industries (ZimPhos) (Msuku, 2012; Manjengwa, 2014). The typical water hardness for the ZimPhos feed water ranges between 40ppm and 110ppm. During the commissioning of the SPD 1500 water softener at ZimPhos the intake water hardness averaged 40 ppm (Msuku, 2012). Using the water quality report for the Johannesburg metropolitan area a water hardness based on equivalent $CaCO_3$ was determined to be 74.70ppm which approximated to the mean water hardness of Zimphos feed water (Msuku, 2012; Mohotsi, 2016). Using the proximal similarity in total water hardness, the sizing of the water softener and specification of reagents associated with softening water was done using the ZimPhos water softening operations as a guide. The water softening reactions can be summarized as follows:

Softening:

$$RNa_2(s) \; + \; Ca(HCO_3)_2 \,(aq) \longrightarrow CaR \; + \quad 2NaHCO_3 \tag{5.2}$$

Resin regeneration:

$$CaR(s) + NaCl \,(aq) \,(excess) \longrightarrow RNa_2(s) \; + NaCl \,(aq) \,(excess) + \; CaCl_2 \,(aq) \tag{5.3}$$
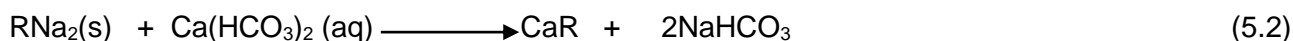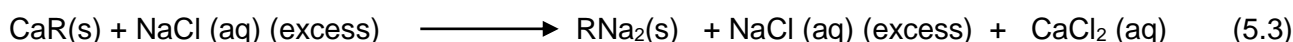
Resin regeneration

The regeneration of water softener resins differs slightly in time periods and ranges between 1.5 and 2 hours. The standard procedure for resin regeneration of softeners consists of four different stages that also differ in length and are subject to the uniqueness of the operations of the plant and the feed water quality. The stages are summarized in Table 5.5. Typical operational data was drawn from plant data during the commissioning of the SPD 1500 water softener at ZimPhos and the DOW water conditioning manual and data sheet (DOW Water Solutions (not dated); DOW, 1999; Msuku, 2012).

It was assumed for this study that the regeneration cycle would replicate a similar profile for the operating times. For the process under study a strong acid cationic resin was selected namely, DOWEX HCR. A brine solution ranging between 8% and 12% is recommended for the brine injection stage (DOW Water Solutions (not dated); DOW, 1999). It was assumed that a 10% brine solution and a flow through velocity of $0.296m^3$/hour was calculated for the process under study. A detailed operational data sheet was generated for the water softening parameters for the process under study. This data sheet was used as the base case for all the business models (see appendix F, Tables F.1 and F.2). Respective data sheets were generated based on the specific water requirements of each business model.

Table 5.5: Regeneration of softening resin (extracted from the main operational data sheet appendix F, Tables F.1 and F.2)

Stage	Backwash	Brine injection	Rapid rinse	Brine refill
Time (minutes)	10	60	12	18
Salt demand (kg/day)	0	0.83	0	0.83
Water usage (m^3)	0.047	0.281	0.056	0.068
Flow destination	Effluent	Effluent	Effluent	Process

Resin replacement

Resin performance is governed by the operating conditions to which it is subjected. The damage results from swelling and weakening of the divinylbenzene (DVB) cross linking within the resin structure. Feed water streams with high chlorine and iron cause extensive damage to the resin cross-linking structure and swelling as a result of the iron that is oxidised while within the resin. The normal operating life for water softener resin bed in urban areas can extend from 10 to 15 years. For this study it was assumed that the resin bed was replenished once for the life of the project. It was also assumed that topping of the resin bed volume was not be done since there were no losses of resin to drain or carryover into the main process.

5.3.2.2.   Steam generation and process heating

Steam was not included as the main heat source in the process. The determination of the total energy requirements from previous studies were used to evaluate the steam demand. The steam requirements were used to size the boiler for the operations. For this process the tonnage of steam at $130^{O}C$ and 2.70 bars required per day was found to be just under one tonne (0.029tonnes/hour). The volume of steam required did not justify the installation of a boiler and the auxiliary equipment. Electric heat exchangers and compressed air were instead used for the process. Compressed air was assumed to be used for maintaining pressurized operating conditions in the elution column and fluidizing the slurry bed in the nitric acid wash circuit.

5.3.2.3.   Compressed air

The compressor was sized based on the requirements of the respective plants. The actual compressor size was 1.5 times the compressed air requirements to factor in pressure drop along the compressed air piping system. A reciprocating compressor with its receiving cylinder was included in the list of major equipment.

The compressed air requirements for the plants are summarized in Table 5.6. The specific compressed air requirements were updated for each business model in the economic studies. The requirements summarized in Table 5.6 were based on the original basis (production of metallic copper and gold).

Table 5.6: Compressed air requirements for the plants (original basis business model 3a)

Plant	Size reduction	Solder leaching	Copper extraction	Nitric acid wash	Gold extraction
Pressurizing (m^3)	0.00	0.00	0.00	0.00	11.25
Running time (hours)	0.00	0.00	0.00	0.00	12.00
Process (m^3)	0.00	0.00	0.00	0.00	0.05
Running time (hours)	0.00	0.00	0.00	0.00	24.00
Agitation/fluidization (m^3)	0.00	0.00	0.00	3.75	0.00
Running time (hours)	0.00	0.00	0.00	4.00	0.00
Total compressed air (m^3)	0.00	0.00	0.00	**3.75**	**11.31**

The costing of the compressed air was done based on the running time, load, operating efficiency and its availability according to the formula specified as follows (Hydraulicspneumatics, 2018):

Total cost = [(hp_b) x (0.746) x (operating hours) x ($/KWh) x (% available time) x (%full load hp)] / (Motor efficiency)   (5.4)

Typically, the efficiency of compressors ranges between 10% and 15%. For the process under study a compressor efficiency of 15% was assumed and the required power was 6.67 times the rated power for the compressor motor (Hydraulicspneumatics, 2018).

The running configurations were specified based on the compressed air requirements in the process. For this study it was found that the compressor would run on full load for 13 hours each day. For the remaining 11 hours it was assumed that the compressor would run on 25% of the full load. It was also assumed that the compressor motor efficiency when running on full load was 95% and 90% when running at 25% of the full load.

5.3.3. Plant maintenance

Routine equipment plant maintenance for the plant was assumed to be fitted into the operations of the plant requiring between one to two hours per session. For the study it was also assumed that during steady state operations no major process interruptions would occur to prolong unplanned downtime. Routine maintenance in each plant was assumed to be done once on a weekly basis. Sections that were offline during material discharge or material preparation were subject to preventative maintenance inspections.

For the study it was assumed that no anti-scalant was dosed in line in all of the heat exchangers. It was assumed that the operation of the softeners and municipal feed water quality would not generate scale at rates warranting frequent suspension of operations to remove scale. The heat transfer efficiency of the heat exchangers remained satisfactory throughout the production year. Descaling would be done annually during major plant servicing on a major plant shutdown campaign. Major shutdown work was scheduled for the close of the production year between November through December.

5.3.4. Organizational and operational structure

The proposed organograms for each of the three operating configurations are depicted in Figures 5.1 and 5.2. Supporting departments were included in the structure. The proposed organograms support the inclusion of all major activities, namely size reduction, solder leaching, copper electrowinning, nitric acid washing and gold extraction. The costs associated with the manpower complement and supporting operations of the other departments used in the economic study were estimated using typical labour prices from available online and supporting heuristics available in literature (Peters & Timmerhaus, 1991; Turton *et al.,* 2009; Indeed, 2018; 2018b; Indeed, 2018; 2018b).Independent engineering, supplies logistics, quality assurance control and safety departments were not specified for the defined scale of operations. The roles were assumed to be handled by appropriate skilled personnel supported by external service providers for more complex functions. The security function was assumed to be managed in house by the administration department with the assistance of an external security company. External maintenance engineers were assumed to be outsourced at defined periods. Three resident support artisans housed under the production department would be employed to support operations in the plant.

Figure 5.1: Proposed organogram for production operating configurations 1 and 3

Proposed organogram for production operating configuration 2
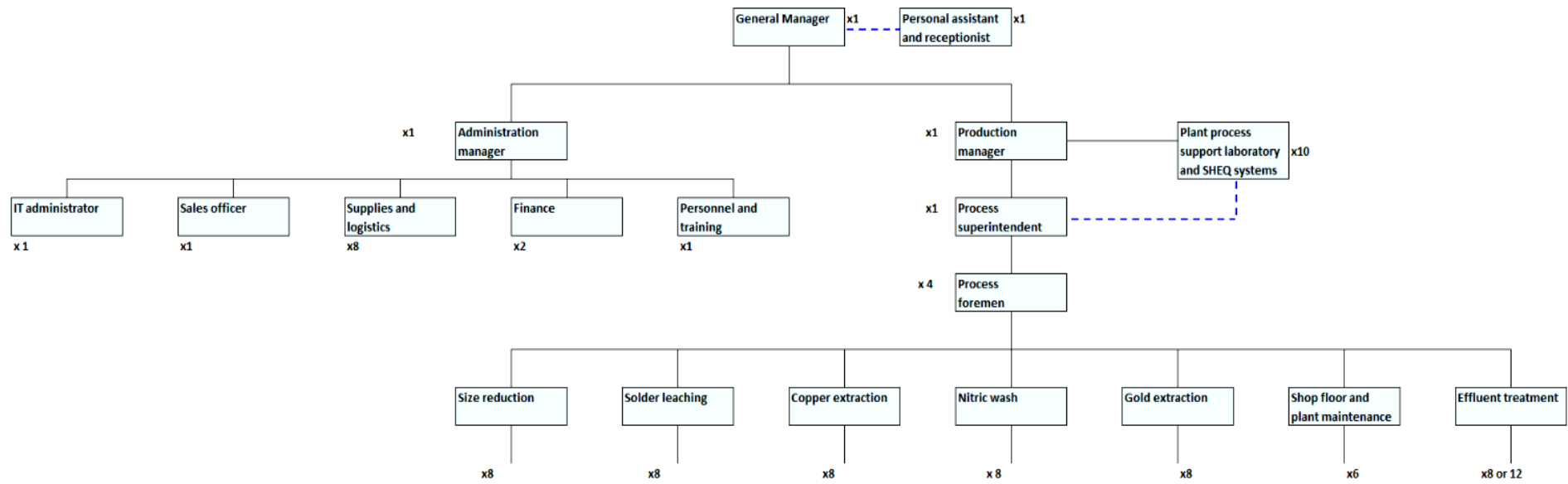
Figure 5.2: Proposed organogram for production operating configuration 2

Table 5.7 summarizes the proposed distribution of the manpower required for the entire operation in the configurations depicted in the organograms. The manpower requirements are not static and were subject to change as the scale, complexity of operations and business model changes.

Table 5.7: Specific manpower requirements for each of the production operating configurations

Department	Configuration 1	Configuration 2	Configuration 3	Skill level
Executive management				
	General manager	General manager	General manager	Specialised
Administration department				
	Administration manager	Administration manager	Administration manager	Specialised
	Suppliers and logistics officer	Suppliers and logistics officer	Suppliers and logistics officer	Skilled
	Warehouse clerk	Warehouse clerk	Warehouse clerk	Semiskilled
	Buyer	Buyer	Buyer	Skilled
	Accountant	Accountant	Accountant	Skilled
	Personnel officer	Personnel officer	Personnel officer	Skilled
	Shift drivers x4	Shift drivers x4	Shift drivers x4	Unskilled
	Fork lift driver	Fork lift driver	Fork lift driver	Unskilled
	Finance and salaries administrator	Finance and salaries administrator	Finance and salaries administrator	Skilled
	IT administrator x1	IT administrator x1	IT administrator x1	Skilled
	Sales officer	Sales officer	Sales officer	Skilled
	Receptionist/Personal assistant	Receptionist/Personal assistant	Receptionist/Personal assistant	Semi skilled
Production department				
	Production manager	Production manager	Production manager	Specialised
	Process superintendent x2	Process superintendent x2	Process superintendent x2	Skilled
	Shift foreman x4	Shift foreman x4	Shift foreman x4	Skilled
	Analytical chemist	Analytical chemist	Analytical chemist	Specialised
	SHE systems officer	SHE systems officer	SHE systems officer	Skilled
	Shift laboratory technicians x8	Shift laboratory technicians x8	Shift laboratory technicians x8	Skilled
	Size reduction plant floor operators x4	Size reduction plant floor operators x4	Size reduction plant floor operators x4	Semi skilled
	Solder leaching plant floor operators x4	Solder leaching plant floor operators x4	Solder leaching plant floor operators x4	Semi skilled
	Copper extraction plant operators x4	Copper Sulphate extraction plant operators x4	Copper Sulphate extraction plant operators x4	Semi skilled
	Nitric acid wash plant floor operators x4	Nitric acid wash plant floor operators x4	Nitric acid wash plant floor operators x4	Semi skilled
	Gold extraction plant floor operators x4	Gold extraction plant floor operators x4	Gold extraction plant floor operators x4	Semi skilled
	Effluent and solid waste plant x12	Effluent and solid waste plant x12	Effluent and solid waste plant x12	Semi skilled
	Shift artisans x4	Shift artisans x4	Shift artisans x4	Skilled
	General hands x3	General hands x3	General hands x3	Unskilled

5.3.5.   Selection of the operating configuration

The selection of the most appropriate production operating configuration was done by making use of a matrix outlined in Table 5.8. Five different metrics were defined for the selection and are listed as follows:

- Initial capital investment
- Equipment utilization
- Potential for employment creation
- Potential for stable production rates and equipment utilization in constrained operating environments
- Strategic and tactical flexibility in conditions of perpetual uncertainty
- Ease of including inter-organisational lateral diversification

By ranking the metrics in the order of priority a scale of 1 to 3 was used for scoring each of the metrics under consideration. The highest score out of 3 for each metric was awarded for the most favourable option. Similarly the least favourable option scored 1 out of a maximum score of 3. Based on the sum of the scores, the operating configuration that had the highest overall score out of a maximum of 18 was selected.

Table 5.8: Matrix for selection of the best production operating configuration

Production operating configuration/Metric	Configuration 1	Configuration 2	Configuration 3
Lowest initial capital investment	1	3	2
Equipment utilization	1	3	2
Potential for employment creation	1	3	2
Potential for stable production rates and equipment utilization in constrained operating environments	1	3	2
Strategic and tactic flexibility in conditions of perpetual uncertainty	3	1	2
Ease of including inter-organisational lateral diversification	1	3	2
Score	**8**	**15**	**12**

All equipment sizing was done based on the production operating configuration 2, which had the highest score of 15 out of 18. Further costing and economic evaluations of all the remaining business models were based on the second production operating configuration. The detailed schedule of equipment sizes and operating conditions is included in appendix F, Table F.7.

5.4.   Chapter summary

This chapter described in detail the key considerations that were factored during the economic study. The first part of the study provided a discussion of the factors that affect the calculation of capital and operating costs and the overall profitability analysis.

The bases of the assumptions used were discussed in the early part of the chapter. This section was followed by a detailed discussion of the aspects related to operations management around the production plant.

A discussion of the main supporting plant services was incorporated namely, utilities that include soft water supply, compressed air and plant maintenance. The continuous, weekly and annual maintenance schedules were described in this chapter. The service sections were also factored in the detailed economic studies in chapters 7 and 8 of this study. A description of the generic inventory plan affecting raw material supply and handling was also included.

Potential production plans were specified drawn from a detailed process debottlenecking of each section. Three possible configurations were described which formed the basis of equipment sizing. Each configuration was accompanied by a supporting organogram that listed the operating, plant supervision and other personnel that supported activities associated with the proposed venture. It was assumed that there was no independent engineering, supplies & logistics, quality assurance control and safety departments. These functions were housed either under the production department or the administration department as depicted in the organograms. The roles were assumed to be handled by appropriate skilled personnel supported by external service providers for more complex functions.

Six metrics were used to evaluate each of the operating configurations. The metrics included the initial capital investment associated with the configuration, equipment utilization, potential for employment creation, potential for stable production rates and equipment utilization in constrained operating environments, strategic and tactical flexibility in conditions of perpetual uncertainty and the ease of including inter-organisational lateral diversification. Following the analysis of each operating configuration based on the metrics, the second operating configuration was selected. This option minimized the sizing of all equipment while maximizing plant and time utilization. The second operating configuration was used for all the remaining business models for the economic study covered in chapters 7 and 8.

Chapter 6:   FIRST LEVEL SCREENING OF BUSINESS MODELS

6.1.   Development of business models

6.1.1. Business models

A business model can be described as the philosophy behind an organization, concern or individual's ability to generate revenue. Revenue can be generated within the life cycle of a service or product through the provision of related services associated with product realization or through the actual generation of the product of interest.

The selection of a business model is a function of prevailing macro-economic factors that define the need for the product and legal frameworks governing the activities of the business, threats in the environment that will nullify the need for the product and hence terminate the revenue stream. Business models possess interdependencies. This becomes especially defined in established economies with vibrant market activity.

The selection of a business model depends on the economics and revenues realisable. The profitability embraces the financial aspect and the measure to which risk is shouldered by the concern. A combination of these types of profitability influences the selection of a business model. Contemporary business environments however embrace the relevance of additional metrics such as the sociological and environmental impacts and the concept of energy preservation on the selection and adoption of a business model.

In the absence of existent or supporting structures for a concern's activities, opportunities exist for the establishment of businesses to support their activities. The permanency however is affected by the size and turnover of the parent concern from which the other businesses were birthed and the likelihood of the emergence of parallel ventures emerging in need of their services. The absence of the support services to a company are what become the business models of the new ventures established around the activities of the parent company in producing its scope of products.

If these new businesses are unviable they will either collapse or else realign their activities to generate better or additional revenue streams. In the case where alternative revenue streams are defined, the concern's business model is said to have evolved. Figure 6.1 illustrates the basic anatomy of a business model. As noted in the latter remarks, when the need is removed from the chain the immediate result is death unless the model is reinvented through alternative strategic intervention.
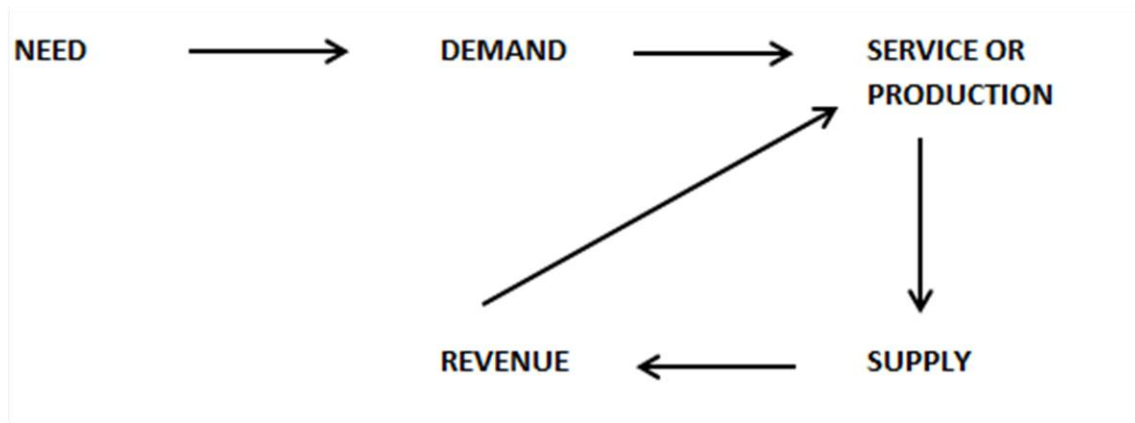
Figure 6.1: An illustration of the anatomy of business models

A regular analysis of a concern's competitive advantage and business environment is therefore essential for the architects of the business to ascertain the feasibility, the practical life span and risks associated with the adoption of a business model.

Within the framework of this study, two major factors were considered in the development of business models from the proposed hydrometallurgical process and are discussed in the following section.

6.1.2. Criteria for defining the business models

Two criteria were combined to define the business models for the hydrometallurgical operations named HYDROMET WPCB Inc. The first criterion used was based on the final products produced for sale. In the scope of this discussion the main products of interest are the production of gold and copper based products. Four classes of the business models were defined. The classes are illustrated by Figures 6.2 through to Figure 6.5

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Product based classification A

HYDROMET WPCB Inc.

Collection and Dismantling

Size reduction

Solder leaching

Copper extraction

Nitric acid washing

Gold extraction

Copper cathodes

Gold sludge

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA
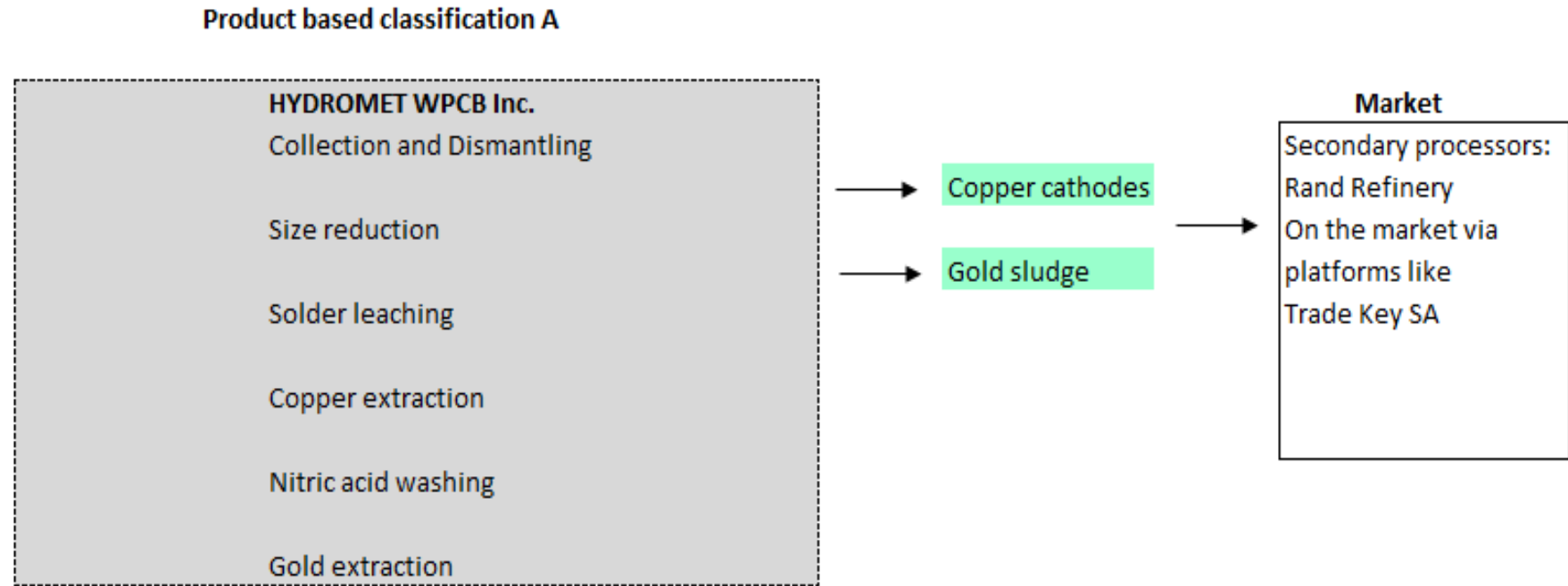
Figure 6.2: Product classification A - production of copper and gold metal

Product based classification B

Market

Precious metals intermediate chemical producers & traders
Trade Key South Africa, SA Precious Metals

Copper liquor Gold liquor

HYDROMET WPCB Inc.

Collection and Dismantling

Size reduction

Solder leaching

Copper extraction

Nitric acid washing

Gold extraction
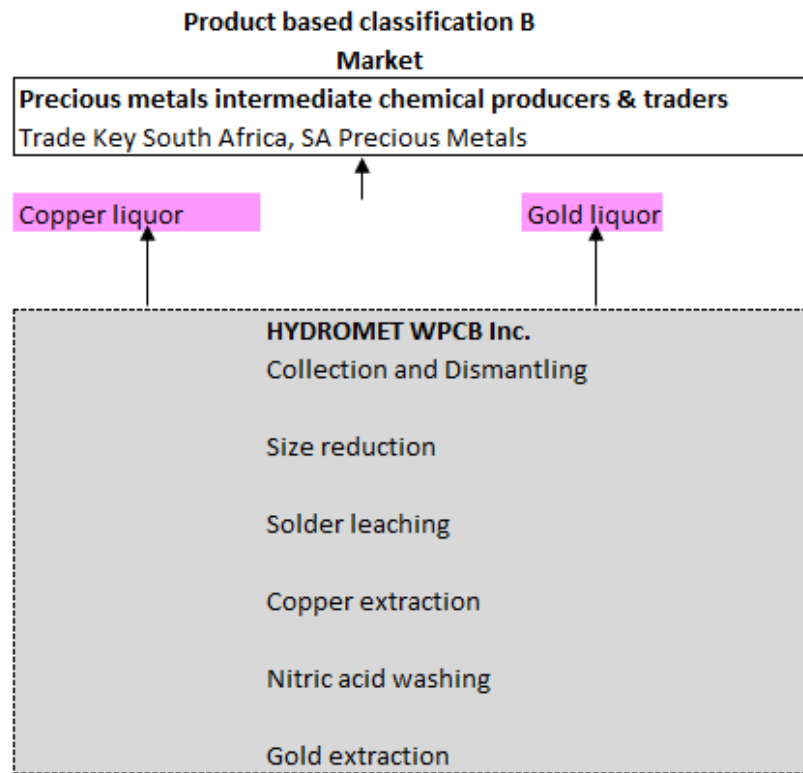
Figure 6.3: Product classification B - production of copper and gold liquors
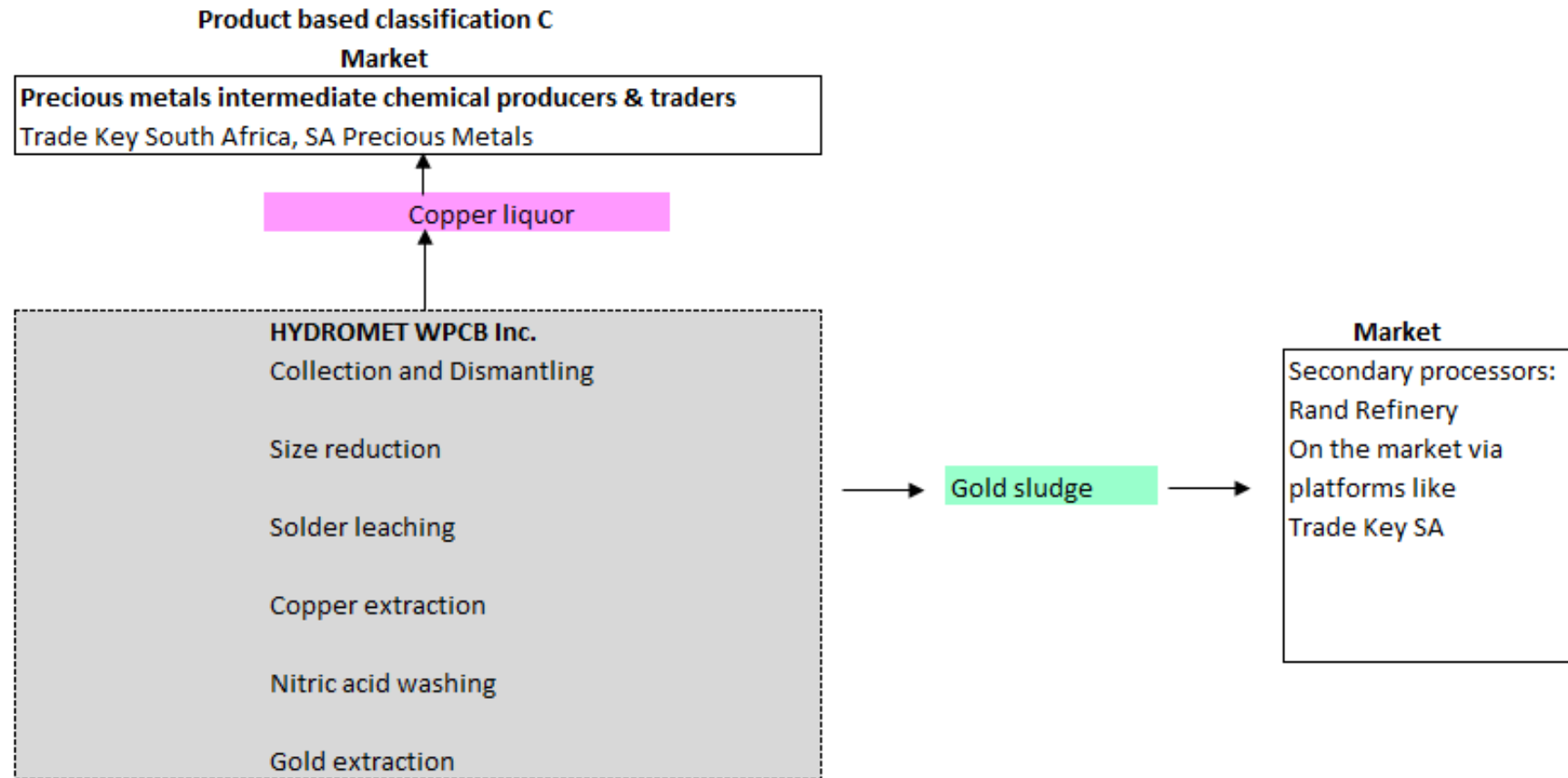
Figure 6.4: Product classification C - production of copper liquor and gold metal

Product based classification D

Market

Precious metals intermediate chemical producers & traders

Trade Key South Africa, SA Precious Metals

Gold liquor

HYDROMET WPCB Inc.

Collection and Dismantling

Size reduction

Solder leaching

Copper extraction

Nitric acid washing

Gold extraction

Copper cathodes

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA

Figure 6.5: Product classification D - production of copper metal and gold liquor

The second criteria used was based on a consideration of the activities that could either be incorporated as an in-house function in the entire scope of operations or could be outsourced. A consideration of the existent stakeholders in the South African e-waste recycling and value chain enabled the identification of operations that could be outsourced. The chief activities considered included collection and dismantling, crushing and shredding (size reduction), the extraction of copper and gold, purification of copper, purification of gold and selling of the output.

The analysis of the South African value chain revealed that the well established collection and dismantling and pre-treatment activities could be exploited. This meant that the collection and dismantling of e-waste and the size reduction functions could potentially be outsourced. Four alternative classifications of business models were then defined based on this criterion. The classifications labelled 1 through to 4, are illustrated schematically in Figures 6.6 through to 6.9.

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Operations based classification 1

Market

| Precious metals intermediate chemical producers & traders |
| Trade Key South Africa, SA Precious Metals |

Products (liquors)

Outsouced

None

Inhouse operations

HYDROMET WPCB Inc.
Collection and Dismantling

Size reduction

Solder leaching

Copper extraction

Nitric acid washing

Gold extraction

(Products)

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA

Figure 6.6: Operations based classification 1 - all operations done in-house

Operations based classification 2

Market

Precious metals intermediate chemical producers & traders
Trade Key South Africa, SA Precious Metals

Products (liquors)

Outsouced

Size reduction
Eg. Sindawonye Granulators & Processors
Universal Recycling Company
New Reclamation Group
Sims Recycling

Inhouse operations

HYDROMET WPCB Inc.
Collection and Dismantling

Solder leaching

Copper extraction

Nitric acid washing

Gold extraction

(Products)

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA

Figure 6.7: Operations based classification 2 - only size reduction is outsourced

Operations based classification 3

Market

Precious metals intermediate chemical producers & traders
Trade Key South Africa, SA Precious Metals

Products (liquors)

Outsouced

Collection and dismantling
eg. Desco Recycling
RecLite
Africa E-Waste
E-Waste Africa
Sibanye Recycling Ltd
Sindawonye Granulators & Processors
Waste Plan

Inhouse operations

HYDROMET WPCB Inc.
Size reduction
Solder leaching
Copper extraction
Nitric acid washing
Gold extraction

(Products)

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA

Figure 6.8: Operations based classification 3 - only collection and dismantling is outsourced

Operations based classification 4

Market

Precious metals intermediate chemical producers & traders
Trade Key South Africa, SA Precious Metals

↑

Products (liquors)

Outsouced

Collection and dismantling
eg. Desco Recycling
RecLite
Africa E-Waste
E-Waste Africa
Sibanye Recycling Ltd
Sindawonye Granulators & Processors
Waste Plan
Size reduction
Eg. Sindawonye Granulators & Processors
Universal Recycling Company
New Reclamation Group
Sims Recycling

→

Inhouse operations

HYDROMET WPCB Inc.
Solder leaching

Copper extraction

Nitric acid washing

Gold extraction

→ (Products) →

Market

Secondary processors:
Rand Refinery
On the market via
platforms like
Trade Key SA

Figure 6.9: Operations based classification 4 - collection & dismantling and size reduction are outsourced

Combining the two criteria a 4 x 4 matrix for specifying different combinations was generated. The combinations are outlined in the matrix in Table 6.1:

Table 6.1: Matrix for defining combinations to specify business models

CRITERIA	Product based classification	a	b	c	d
Operations based classification	Matrix of possible combinations				
1		1a	1b	1c	1d
2		2a	2b	2c	2d
3		3a	3b	3c	4c
4		4a	4b	4d	4d

Sixteen business models were therefore defined from these combinations and are depicted in Figures 6.10 and 6.11 below. The blue boxes depict production activities that are done in-house. The pink boxes represent operations that are outsourced or are associated with the production of chemical intermediates. In the context of this discussion the chemical intermediates were gold liquor (aurocyanide solution 142.52 ppm gold content) and copper liquor (copper sulphate solution, 3.71% Cu).

BUSINESS MODEL	1a	1b	1c	1d	2a	2b	2c	2d
Collect and dismantle								
Shred and crush								
Solder leaching								
Copper extraction								
Nitric acid wash								
Gold extraction								
Primary products	Gold sludge Copper cathodes	Gold liquor CuSO4 liquor	Gold sludge CuSO4 liquor	Gold liquor Copper cathodes	Gold sludge Copper cathodes	Gold liquor CuSO4 liquor	Gold sludge CuSO4 liquor	Gold liquor Copper cathodes
By -products	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue

Figure 6.10: The first eight business models, groups 1 and 2 characterised by in-house collection and dismantling of WPCBs

BUSINESS MODEL	3a	3b	3c	3d	4a	4b	4c	4d
Collect and dismantle								
Shred and crush								
Solder leaching								
Copper extraction								
Nitric acid wash								
Gold extraction								
Primary products	Gold sludge Copper cathodes	Gold liquor CuSO4 liquor	Gold sludge CuSO4 liquor	Gold liquor Copper cathodes	Gold sludge Copper cathodes	Gold liquor CuSO4 liquor	Gold sludge CuSO4 liquor	Gold liquor Copper cathodes
By -products	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue	Precious and base metal liquors Non metallics REE residue

Figure 6.11: The last eight business models, groups 3 and 4 characterised by outsourced collection and dismantling

6.2. First level screening and selection

Supply chain logistics costs

The costs associated with the manufactured product being moved from its point of origin to the end user are what make up the costs of supply chain logistics. Figure 6.12 illustrates the typical cost components of the supply chain of electrical and electronic goods.
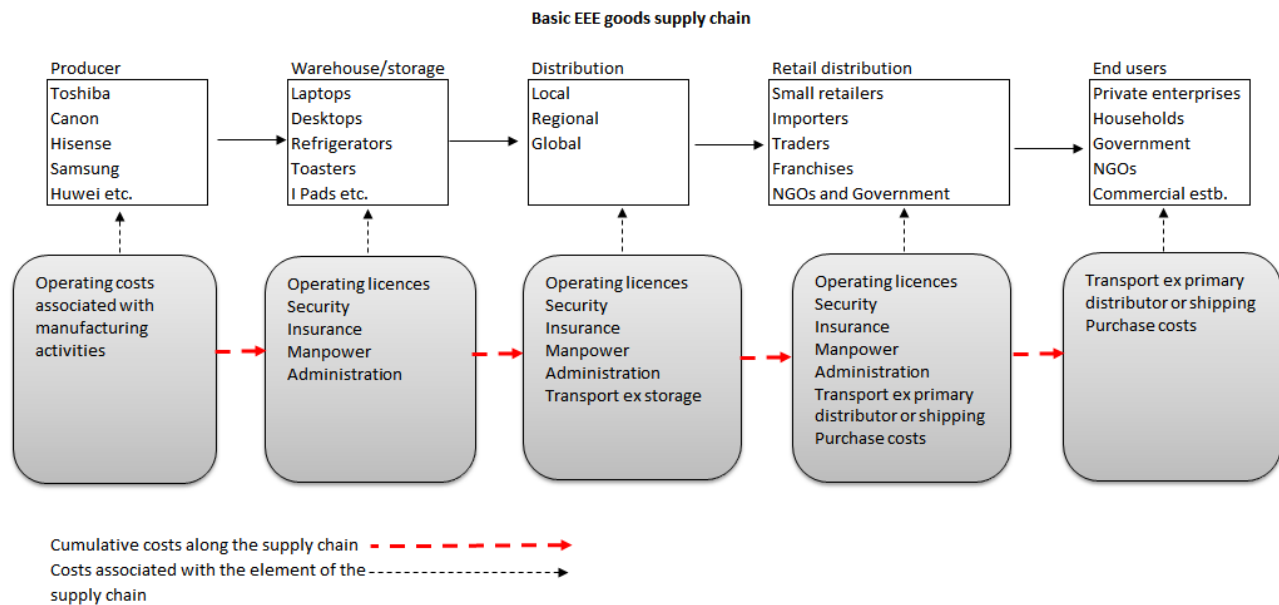
Figure 6.12: Basic EEE goods supply chain and components of the logistical costs

The reverse logistics costs describe the cumulative costs along the supply chain typically associated with reuse or recycling industries. The structure of the reversed supply chain for the good at the end of its useful service life destined for recycling often does not terminate at the original manufacturer alone. There are three major termination points associated with the reverse supply chain. Depending on the business model associated with the recovery of the good in question, more than one termination point can exist as illustrated in Figure 6.13.

The first termination point (Point 1) associated with the original producer characterises the fundamental attribute of extended producer responsibility (EPR). In such a case the producer houses a business unit in the broad portfolio of the concern's operating activities capable of reusing, refurbishing or recycling the whole or part of the product and reintegrating the output into the normal production activities.

The second termination point (Point 2) is peculiar in its ability to potentially be extended beyond the original manufacturer closing the reversed supply chain without returning to the point of origin. When the termination of the reverse supply chain extends beyond the closed loop characterizing an EPR system, the cost components cumulating to the overall cost of the reverse logistics increases.

The reversed supply chain is extended to independent recyclers who incur both costs of warehousing, transfer and collection from the central supplier in addition to paying premiums charged on the elements of the reversed supply chain. The effect of the higher logistics costs associated with this supply chain is magnified when the capacity to add value is low, the market absorption capacity for by-products is very low and the regulatory climate is unconducive to supporting the operations of the enterprise. This can also be seen when the reverse supply chain terminates at point 3, disassembly and material recycling.

As in the case of the prevalent e-waste recycling economy of South Africa, the supply chain has assumed a two pronged and highly branched configuration, characterised by a high reverse logistics cost component (both hidden and direct). A highly branched reverse supply chain oftentimes can be characterised by varied and increasing costs along the supply chain due to the increased frequency of premiums charged onto the desired product of interest: in this case e-waste.

The hidden costs such as inhomogeneous purchase prices and material quality increase with the increasing complexity of the EOL EEE supply chain and is aggravated in conditions of scarce supply of the key material, EOL EEE.
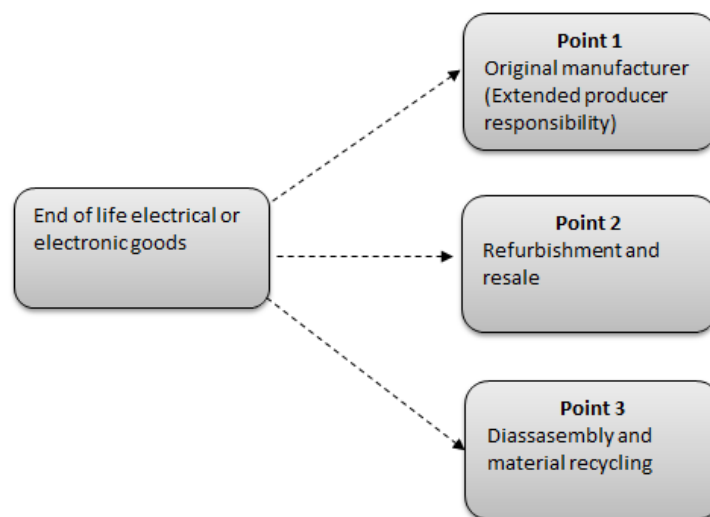
Figure 6.13: Three termination points for the reverse supply chain for EOL electrical and electronic goods

Selection of the business models

The selection of the business models was conducted on two levels illustrated in Figure 6.14. The first level entailed an assessment of the internal and external capacity to manage the reverse logistics function. Results of the analysis conducted on the first level created a platform for the elimination of some of the business models as depicted in Figure 6.14. Refer to section 6.2.5 for the output of the first level screening used for eliminating some of the business models from further study. Level 2 selection was described briefly in section 6.3.

Level 1 selection

Assessment of internal and
external capacity to handle
impact of operations

- - - - - - Screen and eliminate

Rejected models

Models with inhouse
collection and dismantling
models 1a to 2d
See section (6.2.5)

Level 2 selection

Analysis of profitability of each
model
Evaluation of maximum revenue
realisable *(See section 6.3 & Chapter 7)*

Screen and eliminate

Best fit

Undergoes sensitivity analysis

Figure 6.14: Screening criteria for business models with output for first level analysis (refer to sections 6.2.5 and 6.3)

An understanding of the proposed venture's projected competitive strategic positioning in the existent e-waste recycling economy in South Africa was established by means of a SWOT analysis. Porter's five force model was not used in this stage of the analysis since the proposed undertaking was only at the preliminary stage of development. The output of the SWOT analysis is detailed in Table 6.2.

Table 6.2: SWOT analysis of the proposed project undertaking

Attribute	Findings
Strengths	Possesses expertise in base metals management within the gold circuit. There is leverage for diversification of the business models for potential scale up of operations and expansion of product portfolio. Process was designed to maximise water utilization and lower internal inventory of hazardous effluent.
Opportunities	The existence of a vibrant and established collection and dismantling industry. Established organizations specializing in pre-treatment of WPCBs. Diversion of capital and operating risk associated with outsourcing collection and dismantling and e-waste pre-treatment operations.
Weaknesses	Operating capacity may disadvantage the business in strained macro-economic conditions of raw material supply. No access and exposure to detailed market intelligence on the aurocyanide liquor market. Unhealthy dependence on a sole proprietor for market intelligence with very similar operations. Overdependence on a sole chief revenue base. Susceptible to commodities market risk factors for both copper and gold businesses. Has no operating leverage to use predatory pricing as defence strategy. Does not possess the capacity to use penetration pricing as an aggressive entry strategy. Unable to retain a healthy market share in the face uncompetitive business operations or saturation of similar or parallel value adding activities downstream of the value chain. Gold liquor business heavily dependent on strong, reliable client/supplier collaborations.
Threats	Low and inconsistent e-waste inflows into the e-waste recycling economy. The existence of a pseudo monopolistic business climate in the distribution of large e-waste consignments. Tightening regulations and legislative requirements on handling of effluent discharge. Risk of operating at a capacity of not more than 65% per year.

In this study five metrics critical to maximising competitive advantage of the proposed venture, were therefore considered in the first screening level. These metrics included an analysis of the impact of the proposed operations in relation to:

- The storage space requirements,
- Raw material supply within South Africa,
- Manpower and wage bill requirements,
- By-product and waste projections
- The potential revenue sources and streams from the associated by-products.

Calculations for all the analyses done in the first level screening were based on the assumptions summarized in Table 6.3. The output of the calculations used for the analysis is included in appendix E, Table E.1.

Table 6.3: Assumptions used for first level analysis of business models

Parameter	Assumptions	Reference
Bulk density of raw e-waste	210 kg/m^3	SITA waste calculator
Proportion by weight of WPCBs in raw e-waste	1.7%	Widmer *et al.,* (2005)
Maximum dismantling rate per person.	3.5 Tonnes/month	Blaser & Schluep (2012)
Projected tonnages of e-waste in storage (RSA)	2 108 000 Tonnes	Finley & Lietchi (2008)
Projected annual growth rate of e-waste	322 000 Tonnes/year	GreenCape (2018)
Minimum wage as at November 2017 for a 40 hour working week	R3 500/month US$238/month	(*fin24,* 2018:23)
Minimum wage as at November 2017 for a 45 hour working week	R3 900/month US$265.20/month	(*fin24,* 2018:23)

6.2.1.   Storage space requirements

The existent handling and storage capacity for large volumes of unprocessed e-waste was noted to be large, requiring extensive land space as revealed in the analysis (Appendix E, Table E.1). This was also accompanied by a significant input towards security and monitoring associated with the storage spaces. Existing recycling and dismantling operations at present have been reported to operate below the installed capacity of the plants (Lydall *et al.*, 2017). Recent reports indicated that the low and inconsistent inflows of e-waste within the SADC region and South Africa have contributed towards the conservative operating capacities of most established recyclers (Lydall *et al.*, 2017; GreenCape, 2017; 2018).

Most of the recyclers in the formal sector receive e-waste that has undergone some degree of recycling from the smaller suppliers (Lydall *et al.*, 2017). Typically, the informal sector feeds into the formal sector. The smaller recyclers in the formal sector forward their output to the larger recyclers. It would appear that the contribution of the informal sector helps to alleviate the burden of handling and processing large volumes of what may be regarded non-essential material within the scope of the formal recyclers' operations. The case does not hold true for all recyclers however. One recycler was reported to having an inventory of plastics derived from e-waste recycling of up to 2 tonnes (Lydall *et al.*, 2017). It was noted that failure to secure a market to absorb these plastics led to the build up of inventory on the recycler's operating premises.

The nature of operations of each of the recyclers in the formal and informal sectors is unique. They are strongly market driven, dictated largely by the specific supplier client agreements within and outside of the South Africa.

In the absence of such collaborations and agreements in the South African e-waste economy, the likelihood of the businesses operating with meaningful returns to support operating activities declines. The alternative options will then entail exporting, land filling or stockpiling of the fractions.

Earlier findings of the research (section 3.6) revealed that the existing legislation and the municipal by-laws in place are varied in their application, definition and specification across the different regions in South Africa (Dittke, 2007). Research findings further revealed that the existent legislation would have to be revised to allow for specialised long term storage of complex hazardous waste and by-products associated with e-waste recycling. At very high operating capacities, typically associated with field installations, handling of large unprocessed e-waste inventories becomes unsustainable.

Research findings made by Zeng *et al.,* (2015) revealed that WPCB recycling plants tend to be characterised by low margins for both field and mobile (modular) plants. As a measure to curtail the operational risk that lead to the observed low margins (Zeng *et al.,* 2015), it was necessary to review options available to reduce the potential operational capital outlay associated with bulk handling and storage of unprocessed e-waste, enhancing the security of the unprocessed material and paying the fees associated with environmental and other regulatory instruments. Consideration of the existing legislative constraints and restrictive land spaces required for material storage was also done. It became apparent that the inclusion of collection and dismantling as a business unit operation in the scope of operations of the proposed process was an unfavourable undertaking.

6.2.2.　　Raw material supply within South Africa

Based on the projections of e-waste growth rates advanced by researchers (Finlay & Lietchi, 2008; GreenCape, 2017), sufficient volumes exist to sustain operations at full capacity in the plant. While projections of e-waste growth rates favour the expansion of existing e-waste recycling operations, the existent raw material supply risk was identified as one of the threats to the viability and growth of e-waste recycling in South Africa.

The due diligence of different operating capacities was undertaken in light of the existent raw material supply constraint earlier described in section 4.1. It was noted that operations in excess of an annual recycling rate of 1 000 tonnes of WPCBs per annum would be burdened by steep reverse logistics costs associated with circumventing the existent raw material supply risk. The consistent supply of high grade WPCBs into the main e-waste recycling stream was noted to be limited and is expected to deteriorate with the active interest in replacing the use of precious metals in electronics manufacture with cheaper alloys that have similar performance characteristics.

The entire scope of operations initially pegged at a processing capacity of 30 000 tonnes per annum of WPCBs would require a consistent supply of approximately 1 770 000 tonnes of unprocessed e-waste. Based on the inference of previous researchers it has been estimated that the current formal and informal recyclers will only be able to furnish 7.7% of the material requirements of the proposed venture. At this operating capacity the proposed venture will collapse in less than a year of operations as a result of failure to finance operating activities.

A comparison of operations at an established facility, Umicore running at a name plate capacity of 250 000 tonnes per annum of varied secondary metal sources, which includes WPCBs and high value electronics (Hagelüken, 2006), revealed that the success of the proposed hydrometallurgical venture would be affected by the extent to which the capability of logistics function of Umicore could be replicated and even exceeded in South Africa.

Remarks made by current formal recyclers showed that collection and dismantling activities, in isolation, within the South African environment is a specialised business function dependent on extensive raw material supplier networks and collaborative partnerships across the supply and value chains (Mouton & Wichers, 2016; Lydall *et al.*, 2017).

Managing the effects associated with the prevalent raw material supply risk on the proposed hydrometallurgical installation would be dependent on the adoption of such collaborations, characterised by strong lateral integration across the e-waste recycling supply and value chains. As the proposed operating capacity of the recycling campaign increases the dependence on the reliability of such collaborations within the supply and value chains has to increase. These business relationships would have to be sustained for the long term.

Research findings also revealed that there is need to introduce frameworks and working models to promote the release of the e-waste into the main recycling chain to support ongoing e-waste recycling activities and therefore enhance plant capacity utilization of established recyclers (Mouton & Wichers, 2016). The existence of the supply risk is evidenced by the consistently low capacity utilization observed in the formal recycling sector (Lydall *et al.*, 2017). Operations of these recyclers have been reported not to exceed a capacity utilization of 60%.

An initial investigation into the feasibility of handling 30 000 tonnes/annum of WPCBs also exposed the potential existence of unfavourable economics of the reverse logistics associated with collection and dismantling activities. This posed a threat to the economic viability of the undertaking at increasing operating capacities. The interrogation of the proposed operating capacity revealed that a clear plan which forms a specialised function for handling large volumes of non-metallic material would also be required. Without the existence of such a plan, the government objectives, set out to divert larger waste material volumes from landfill would be retarded (Godfrey & Oelofse, 2017).

This however, would place significant pressure on the focus of operations and cost structure of the core activities with the risk of the emergence of protective behaviours by established e-waste recyclers.

In light of these findings the due diligence of the proposed initial operating capacity exposed a likely failure to operate beyond a capacity utilization of 10%. This revealed that the installation of a large scale field installation of a hydrometallurgical plant to recover copper and gold is at present not feasible under the current operating environment.

The operating capacity (name plate) was revised downwards to match the existing raw material supply in South Africa. The operating capacity of Desco recyclers was used as the closest benchmark for further studies. The proposed operating capacity as a result was trimmed by a factor of 75 to give a recycle rate of 400 tonnes per annum of WPCBs. A capacity of 400 tonnes per annum potentially assumes a configuration which is more suited to geographically dispersed raw material supplies in South Africa (Mouton & Wichers, 2016; Lydall *et al.*, 2017).

Table 6.4 summarizes the results of the analysis of the unprocessed WPCB purchase cost against the maximum dismantling rate of a worker.

Table 6.4: Analysis of the raw WPCB purchase cost against maximum dismantling rate of a worker.

WPCB grade	Cost per kg (US$)	WPCBs kg/(per month per person)	Purchase cost of equivalent amount of WPCBs	Minimum wage for 40hr working week	Variance of purchase cost against minimum wage (US$)	Remarks
Low mean Lydall *et al.*, (2017)	0.54	59.50	32.37	238	**-205.63**	Buy WPCBs
Medium mean Lydall *et al.*, (2017)	1.70	59.50	101.15	238	**-136.85**	Buy WPCBs
Average of all grades Lydall *et al.*, (2017)	2.49	59.50	128.78	238	**-109.22**	Buy WPCBs
High mean Lydall *et al.*, (2017)	4.25	59.50	252.88	238	**14.88**	Consider buying WPCBs.
Maximum purchase cost of WPCBs Lydall *et al.*, (2017)	5.78	59.50	343.91	238	**105.91**	Buying is still an option when handling and logistics costs are factored.

An inspection of the results in Table 6.4 revealed that the purchase of low grade, medium grade and an average of all grades of WPCBs was a more viable option compared to in-house dismantling.

The purchase cost of the equivalent weight of WPCBs dismantled was always lower than the minimum wages paid to a worker to dismantle the e-waste in order to recover an equivalent quantity of WPCBs.

In the case of high grade WPCBs the purchase cost of WPCBs exceeded the required minimum wage by a 6% margin. It was expected that the effect of the inflation on the purchase cost of WPCBs would be overshadowed by the additional costs associated with operations dependent entirely on manual labour.

An analysis of the comparative cost of labour versus the highest purchase price of WPCBs was also undertaken. The results of the analysis revealed that wages will always be 31% less than the WPCB purchase costs. At a glance it would appear more attractive to conduct in house collection and dismantling activities. Using a similar analogy to that presented for high mean WPCB purchase cost, the combined influence of other costs associated with reverse logistics was expected to dilute the perceived benefit associated with in-house collection and dismantling of e-waste.

The consistent supply of high grade WPCB is at present also limited and is expected to deteriorate with the active interest in replacing precious metals in electronics WPCBs with cheaper alloys that have similar performance characteristics.

The recovery of WPCBs requires deep dismantling of e-waste to avoid premature losses of metals associated with the liberation of crushed WPCBs still attached to the body of the gadget. Deep dismantling activities however, are more specific. Correspondence with a supplier of e-waste recycling machines noted that the more specialised e-waste recycling equipment is designed for small scale operations. The discussion with the supplier revealed that the deep dismantling of e-waste normally achieved by use of manual labour cannot be replicated at present by the machines designed for e-waste dismantling (Manjengwa, 2018). Large scale recycling operations rely on e-waste that has undergone some degree of dismantling and pre-processing (Hagelüken, 2006).

An increase, therefore in the tonnages of WPCBs to be handled would be associated with an increase in the manpower requirement to achieve the desired deep dismantling (see appendix E, Table E.1). This would be accompanied by a congruent inflation in the wage bill and a reduction on the overall profitability of the operations.

Combining the inflated wage bill with the costs associated with the reverse logistics of handling raw e-waste, the direct purchase of WPCBs remained a more viable option.

6.2.3. By-product and waste projections

Collect–a-can in the 1970s went on a massive drive to collect cans. While successful at building large inventories of cans for recycling, the existent consumer industries and players were not at an operating and technical capacity geared to absorb the sudden influx of raw material. A similar challenge is foreseen with the current status of the South African e-waste recycling climate.

In remarks made in chapter 3, findings of the investigations revealed that the present absorption capacity of e-waste into the main stream e-waste recycling chain has not surpassed the set target of 20% in South Africa (GreenCape, 2017). A maximum of 12% was reported and for the study a cap of 15% was used. The implication of the calculations done suggests a need to investigate the specific material and product requirements of the existing potential consumers in the value chain. Based on the results of study it was shown that a detailed cost benefit analysis must be undertaken if one is to consider expanding the portfolio of operations as an alternative to exporting. This expansion would be achieved as a result of additional processing of by-products and waste streams.

The low off take rates of some by product streams like plastics in the current operating environment was noted to be cyclic in nature (GreenCape, 2017). The analysis predicted the burgeoning of large volumes of waste and by-product streams (appendix E, Table E.1).

The status of the existent market has not developed to such an extent to absorb this influx. In a similar vein the existing infrastructure is also yet to mature to a level to handle the safe processing of large volumes of complex polymetallic effluent and solid waste streams.

A disposal plan for any material not absorbed must therefore be considered carefully. Key operations stakeholders must take responsibility for the land space required to dump the unprocessed by-products. The calculations based on research data suggested that an estimated 1390 acres per month would be taken up as a result of the collection and dismantling activities. This land space requirement is contrary to the thrust towards diversion of waste from landfills. The by-products and waste material also require specialised disposal and storage that prevents the contamination of ground water sources from the progressive seepage of flame retardants and heavy metals.

6.2.4. Additional revenues from by-products

The potential revenue streams associated with the collection and dismantling of e-waste showed that value could be derived from the additional processing of substantial volumes of plastic, glass and metallic fractions. The analysis of the collection and dismantling activities however revealed that the current absorption capacity of the by-product fractions was not expected to exceed 15%. The projection was backed by the reported e-waste recycling rate of 12% (GreenCape, 2017).

In earlier discussions of this study (section 3.4.3.1, Figure 3.11), it was shown that certain by-product fractions ultimately reported to landfill. The recycling of glass, phosphor powders and other fractions in comparison to other streams was noted to be low. The observed trend was aggravated by a low market demand for these streams (Lydall *et al.*, 2017, GreenCape, 2017; 2018). The market for specialised plastic fractions was also noted to be low and turnovers from this niche of e-waste by-products have remained largely cyclic.

To date well established formal recyclers have noted that the export of metallic and non-metallic fractions from e-waste recycling is at present a more lucrative business model (Lydall *et al.*, 2017). The climate in the main export regions for e-waste and the associated by-products has witnessed a shifting trend towards the rigid control of material inflows as evidenced by tightened e-waste legislation in China. Any future consideration for exporting material must be undertaken with an understanding of the need to upgrade material quality to secure customer confidence in cases where the economic viability for local processing is unfavourable.

In a broader light it was seen that the successful and sustained exploitation of by-product streams was and is threatened by the prevalent domestic and emergent export market risk factors. The risk factors were characterised by both the disproportionate material generation and absorption rates.

6.2.5.   Output of first level screening

The findings of the first level selection based on the five metrics interrogated revealed that the in house collection and dismantling business unit function was not a favourable undertaking in the prevailing South African context. The key features of the analysis revealed the existence of:

- High raw material supply risk
- High domestic market supply risk for by product streams
- Emergent export market supply risk for by-product streams
- Incongruent technological and infrastructural capacity to manage large outflows of complex hazardous materials
- Commitment of large tracts of land for disposing of unsalable by-products
- High costs associated with reverse logistics
- Comparatively higher man power costs for large scale deep dismantling of e-waste

On the basis of these findings the first eight business models (1a to 2d) as indicated in Figure 6.10, hosting the in-house collection and dismantling business unit function, were eliminated from the study as depicted by Figure 6.12. The last eight business models (3a to 4d), highlighted in Figure 6.11 were subjected to further interrogation following capital and economic evaluations of the size reduction, solder leaching, copper extraction, nitric acid wash and gold extraction circuits.

6.2.6. Production planning

Combining the output of the operations study covered in section 5 and the development and selection of the business models, production plans were specified for the operations. The outputs for each of the plants that included the key products, by products and wastes were characterised. The results are summarized in Table 6.5.

Table 6.5: Characterization of output streams from each of the plants for business models 3a to 4d

Material/product	Material specifications
Crushing and milling or outsource	
WPCBs	60% non metals 40% metals
Solder leaching	
Desoldered solids	2-15% lead as percentage of metals
Solder leach	Pb 0.13%, Sn 0.03%, Ni 0.04%, Fe 0.39%, Cu 26ppm, Ag 3.4ppm, Al 0.019%
Copper Extraction	
Filtered solids	1.03%Cu, 2.1% Al, Fe 1.95%, Co 0%, Sn 5.5%, Ni 0,84%, Zn 0.096%, Pb 0.072%, Sb 0.55ppm, Au 0.149%, Ag 0.26%
Copper Liquor	3.71% Cu
Copper (cathode sheets)	LME Grade A > 99%
Nitric acid wash	
Stripped solids	Au 0.16%, 1.44% as H_2SnO_3, at least 94.10% non-metallics
Silver rich liquor	Cu 0.21%, Ag 0.55%, Ni 0.17%, Al 0.42%, Fe 0.4%, 93,61% H_2O 93.61%
Gold extraction	
Gold liquor	pH 12.68, Au 142.52ppm, Ag 0.093ppm, $Ca(OH)_2$ 0.35%, NaCN 98.39ppm, Fe 54.10%, Cu 0.39%, Al 0.77ppm, Sn 0ppm, Ni 0.31ppm, Zn 0.036 ppm, Pb 0.00081ppm
Gold sludge	> 98% Au, <120 ppm Ag
Non-metallic solids	3.76% Sn, 50ppm Au, 0.00044ppm Ag

The outputs derived from each of the plants for the respective business models were quantified on an hourly, weekly, monthly and annual basis and the results are detailed in appendix F, Tables F.3 through to F.6 for the remaining eight business models 3a to 4d. Based on the second operating configuration selected in section 5.3.5 and the results of the output analysis, generic production plans for the group 3 and 4 business models were specified. The production plans assumed steady state operating conditions after commissioning the plants. It was also assumed that all products produced are sold and there is no inventory build up. The annual production plans and raw material schedules are summarised in appendix E, Tables E.2 to E.5 and appendix F, Table F.11.

6.3.   The second level screening and selection

The second level screening and selection was done using the results generated from the detailed economic studies of the remaining eight business models, 3a through to 4d described in chapter 7. The final selection of the business models selected for further study by means of a sensitivity analysis is detailed in chapter 8.

6.4.   Chapter summary

Within this chapter a detailed discussion of the development of the business models around the proposed process was undertaken. Based on selected core activities within the business units 16 business models, clustered into groups 1 through to 4, were defined which also bear an inter-relationship to the current business environment and e-waste recycling scene in South Africa. The screening of the business models was considered on two levels. In this chapter the first level screening was discussed in detail. The first level screening entailed an initial assessment of the first business unit of all the business models. The existence of a high raw material supply risk due to low e-waste inflows into the recycling economy was established. This current situation would not allow for very large scale installations of hydrometallurgical operations. Additional observations noted included the potential domestic market supply risk for by product streams with low and seasonal off-takes, emergent export market supply risk for by-product streams, incongruent technological and infrastructural capacity to manage large outflows of complex hazardous materials.

It was established that incorporating the collection and dismantling function within the entire scope of operations would be a capitally and operationally intensive undertaking. Large scale operations would require a large manpower complement to carry out deep dismantling activities that are very specific in nature.

A consideration of the current value chain revealed that sufficient capacity exists to support the proposed hydrometallurgical process. This was supported from findings of the SWOT analysis done for the proposed venture. An examination of the potential capability of effectively circumventing current raw material supply constraints and the steep costs associated with the reverse logistics of collecting e-waste was undertaken. Metrics were specified for conducting the initial evaluation. These metrics covered the key risks, technological capabilities, resource requirements and labour requirements.

Results of the analysis also revealed that the business activity associated with the collection and dismantling function is a specialised function that is capitally intensive and would best be handled by independent proprietors specialising in this role in both the existent informal and formal recycling sectors.

There is potential to also eliminate the size reduction function from the entire scope of operations of the proposed hydrometallurgical process. Registered formal recyclers were identified in the discussion that will serve as suppliers of whole or crushed WPCBs. The attendant raw material, market and technological risk factors also rendered collection and dismantling an unattractive undertaking to be handled as an internalized operation. It was on this basis that the first eight business models, groups 1 and 2, associated with in-house collection and dismantling of e-waste were eliminated. The second level screening of the remaining business models 3a through to 4d was handled in chapter 7.

Chapter 7: PROJECT COSTING AND EVALUATION

7.1. Capital cost estimation

The detailed costing and evaluation of the proposed process flow is covered in this chapter. Costing and evaluation of the project profitability has been done by considering all the remaining eight business models. In this study it was assumed that all the costs were determined at full operating capacity with the assumption that all the material produced was sold.

7.1.1. Capital cost method

Different methods exist for determining capital expenditure (CAPEX). A combination of a summation of actual vendor prices and estimates adjusted using plant cost indices available in literature can be used. The accuracy of the estimates declines as the estimates depend more on adjusted prices. Five types of estimates were described by the American Association of Cost Engineers in 1958 (Perry *et al.*, 1997). The methods are listed in the Table 7.1 as follows:

Table 7.1: Capital cost estimation methods

Item	Estimate	Accuracy
1	Order of magnitude/ratio estimate (For similar types of plant)	10-50%
2	Study estimate (factored estimate) requires knowledge of major equipment.	Up to 30%
3	Preliminary estimate (budget authorisation) more detailed than the study estimate.	Up to 20%
4	Definitive estimate (Project control estimate). Based on considerable data prior to drawings.	Up to 10%
5	Detailed estimate (Firm or contractors estimate) requires completed drawings, specifications and site surveys.	Within 5%

For the purposes of the preliminary economic feasibility study, the study estimate method was used. A combination of available vendor prices data from literature was also used for the capital cost estimates. Bare module cost estimation and CAPCOST software was also employed for estimating equipment costs not readily available or accessible (Perry *et al.*, 1997; Turton *et al.,* 2009). Estimates were also made using the six tenths rule and a factor for reduction based on the difference in capacity of operation and material of construction (Peters & Timmerhaus, 1991; Perry *et al.*, 1997; Turton *et al.,* 2009). The conversion was also corrected using the applicable chemical engineering cost index. Van Wyk (2014) described a similar set up for the recovery of precious metals from low grade ores through cyanide leaching and specified equipment identified in the process under study. Some estimates used in this study using the six tenths rule were derived from the similar design proposed by Van Wyk (2014).

The detailed equipment list for all the remaining business models was based on the equipment list prepared for business model 3a. The schedule is included in appendix F, Table F.7 and F.8. The equipment lists for each business model were adjusted appropriately to cater for the changes in the final product streams.

Start up time is a function of the type and complexity of the process, the degree of interdependency of the available process plants or sections and manpower (Perry *et al.*, 1997). Plants with an initial capital investment of up to US$7.8 million can be completed within 0.83 to 1.5 years and those with initial capital investments in excess of US$7.8 million are estimated to take between 1.5 to 2 years to complete (Perry *et al.*, 1997).

Each business model had different initial capital outlays. Start up times and costs for each of the business models were determined on this basis.

Start up costs range between 8 to 10% of the fixed capital investment (Peters & Timmerhaus, 1991). In this study it was assumed that start up cost was included in the total capital cost of the business models. It was also assumed that the start up cost was 9% of the fixed cost investment for all the models which is reflected in the capital cost estimation of each business model.

In this study all three plant categories were included. They are summarized in Table 7.2 as follows:

Table 7.2: Categories of the production plants for the proposed process

Solids processing sections	Solids/fluids processing sections	Fluids processing
Size reduction plant	Solder leaching plant Nitric acid wash circuit Copper extraction plant Gold extraction plant Soft water production unit Effluent treatment plant	Compressed air unit

The costs making up the direct, indirect and working capital costs can be evaluated as percentages of the delivered equipment cost. The percentages of the purchased equipment costs used were also a function of the type of materials being processed. Peters & Timmerhaus (1991) provide a detailed account of the percentage ranges for the different process plants. Table 7.3 provides the percentage factors used for the estimation of the direct, indirect and working capital cost estimates for the sections in this study.

Table 7.3: Percentages used for determining the costs associated with the project capital expenditure (Peters & Timmerhaus, 1991; Perry *et al.*, 1997)

Direct Costs	Solids processing	Solids/fluids processing	Fluids processing
Direct costs			
Purchased Equipment Delivered	Vendor prices and estimates based on known quotations	Vendor prices and estimates based on known quotations	Vendor prices and estimates based on known quotations
Equipment Installation	45%	39%	47%
Piping (Installed)	16%	31%	66%
Instrumentation & Controls (installed)	9%	13%	18%
Electrical (Installed)	10%	10%	11%
Service facilities (installed)	40%	55%	70%
Land (if purchase is required)	6%	6%	6%
Buildings (including services)	25%	29%	18%
Yard improvements	13%	10%	10%
Indirect costs			
Design, Engineering and Supervision	33%	32%	33%
Construction Expenses	39%	34%	41%
Contractors Fee	17%	18%	21%
Contingency	34%	36%	42%
Working Capital	68%	74%	86%

7.1.2. Capitalization costs

In accounting terms capitalization costs refer to the funds required for the repurchase of equipment that is regarded as an asset in the balance sheet. These assets and their purchase and replacement at the end of the useful service lives are not expenses since they continue to work for the business generating value. In this study capitalization costs were calculated for all the major equipment.

An estimated life of 11 to 12 years is reported for chemical process equipment (Peters & Timmerhaus, 1991). The useful service lives of the equipment ranged between 4 and 12 years. For the operations under study, an average equipment life of 10 years was assumed for the equipment. Some of the listed equipment had useful service lives of at least 2 years. Capitalized costs were evaluated using the expression 7.1, described in literature (Perry *et al.*, 1997).

$$C_K = [C_{FC} - (S/(1 + i)^n)] \times [(1 + i)^n/((1 + i)^n - 1)] \tag{7.1}$$

Where:

'C_K' is the capitalized cost

'S' is the salvage value at the end of the estimated useful life. This is evaluated by subtracting the equipment value after 'n' years of use from original equipment purchase price.

'n' is the estimated useful life of the equipment

'I' is the interest rate

C_{FC} is the fixed capital cost

Capitalization costs were also calculated for process non consumables detailed in Table 7.4.

Table 7.4: Capitalization costs for process non consumables

Section	Material	Estimated service life (years)	Replacement frequency over project life
Copper extraction plant	Kerosene	3	5
Copper extraction plant	LIX 984N	3	5
Copper extraction plant	Polypropylene macrospheres	5	3
Gold extraction plant	Activated carbon	2	9
Water softening unit	Resin (DOWEX Marathon C)	10	1

For the evaluation the capitalization costs were also used as one of the metrics for selecting the most attractive business model. The choice with the lowest capitalization cost was the most desirable.

Two methods can be used for factoring in the capitalization costs. The first method entails treating this cost as part of the initial capital expenditure during initial project evaluations (Peters & Timmerhaus, 1991). The second method discussed in literature describes a method for providing for capitalization costs through amortization throughout the project life (Perry *et al.,* 1997). In this study the first method was used to avoid the inflation of profitability associated with making provisions that reduce taxable income.

7.1.3.   Analysis of the capital cost estimates

Perry *et al.,* (1997) defines the total capital cost of a project as being the sum of the fixed capital cost, the working capital and the cost of land and other non-depreciable costs:

$$C_{TC} \quad = \quad C_F + C_{WC} + C_{\text{Land and other non-depreciable costs}} \tag{7.2}$$

In this study the total capital cost also embraced capitalized costs of equipment and process non consumables replaced at intervals of at least one year or more. The project life was specified at 20 years. The capital cost estimation of the remaining eight business models was undertaken. The equipment sizes were corrected based on the desired products of the respective business models. The summary of the capital cost estimates for the remaining eight business models are detailed in appendix F, Tables F.13 and F.14 and are illustrated graphically in Figure 7.1.
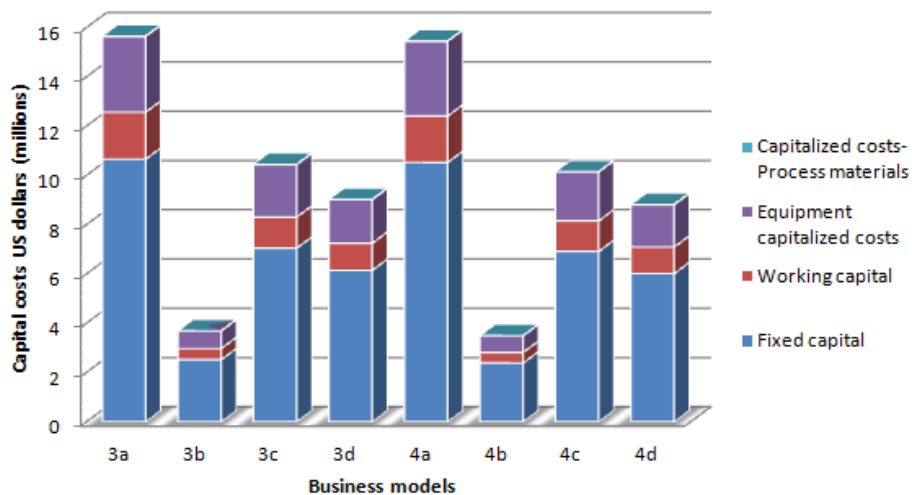
Figure 7.1: Overall capital cost distribution for each business model

An analysis of the capital cost distributions for the group 3 and 4 business models reveals that the business models associated with the production of the solid metal products were associated with the highest fixed capital investment and capitalization costs. A similar trend was observed for the start-up costs for each of the respective business models shown in Figure 7.2.
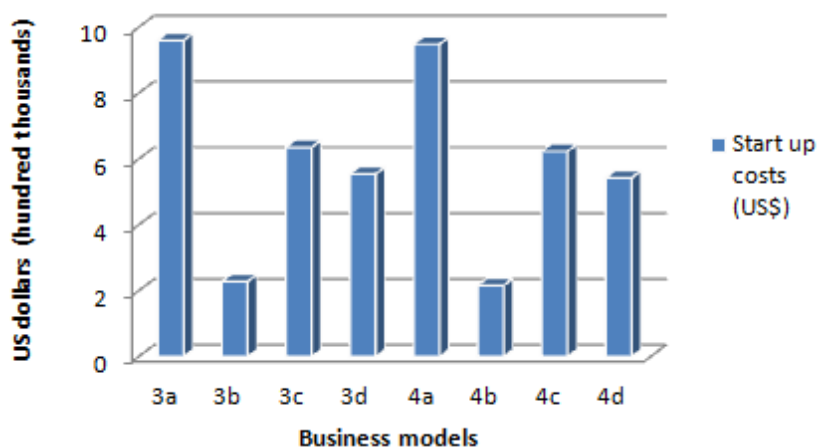
Figure 7.2: Start-up costs for each business model

The elimination of the size reduction plant associated with the group 4 business models resulted in a decrease in the total capital invested by margins ranging between 1% and 5.5%. These changes are summarized in Table 7.5.

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Table 7.5: Decrease in total capital investment in group 4 business models

Business model	4a	4b	4c	4d
Drop in total capital investment (%)	1.25	5.29	2.87	2.16

The largest change was witnessed in business model 4b that was associated with the production of gold and copper liquors. The decision to pursue business models 3b and 4b as opposed to the production of metals associated with 3a and 4a, would lead to a reduction of the initial capital outlay of 76.64% and 77.39% respectively. The greater reduction in the initial capital outlay was associated with business model 4b which was the least capitally intensive. The resultant effect of the elimination of the size reduction plant was more significant compared the other business models with superior initial capital outlays.

An analysis of the distribution of the capital costs on a plant basis was undertaken and is depicted in Figure 7.3. The significant contributors to the initial capital outlay were evidenced by the plants producing one or both of the metal products chiefly, the gold and copper plants associated with business models 3a, 3c, 3d and 4a, 4c and 4d.
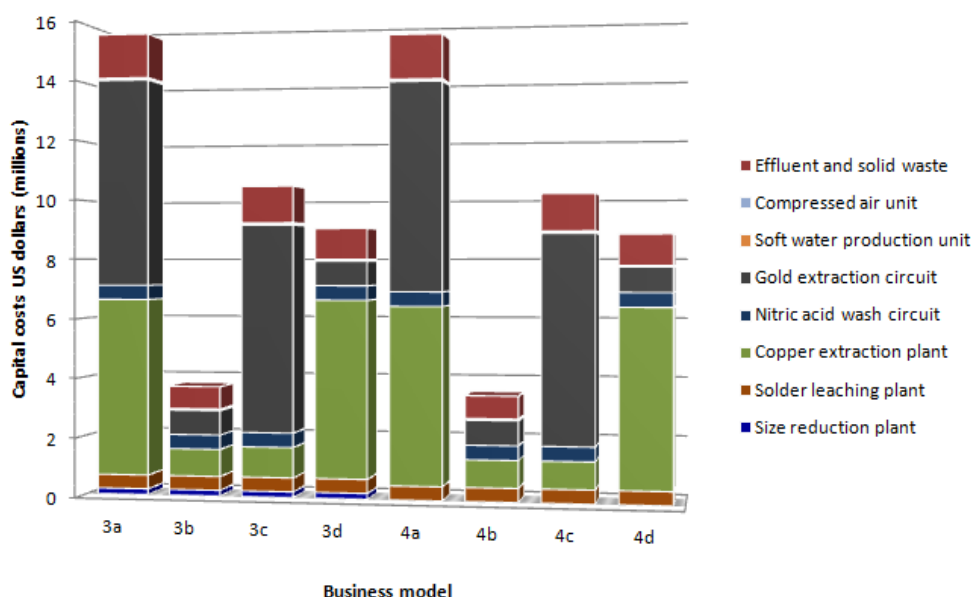
Figure 7.3: Capital cost distribution by plant in each business model

The gold metal sludge production commanded the highest capital investment in both group 3 and 4 business models, exceeding the capital investment in copper cathode production facilities by a margin of 15%. The elimination of equipment associated with the electrowinning of copper and gold liquors resulted in an almost equal distribution of capital outlay in the copper, gold and effluent treatment plants.

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7.2. Operating cost estimation

Operating expenses were calculated by summing the direct, indirect and the general operating costs. Contingency costs were added to the sum of these costs to give the total operating cost for each respective business model.

7.2.1. Raw material costs

The raw material requirements were calculated from the mass balances of the respective plants and service units. Prices of the materials for each section were gathered from online vendor prices, direct enquiries from agents and published data from raw material manufacturers. The purchase price of WPCBs used was based on literature as discussed in chapter 6 (Lydall *et al*., 2017).

An average gold content in the WPCBs feedstock of 250ppm was assumed to represent the closest typical average available when all three grades of WPCBs were blended. Using published data the purchase price for the WPCBs was estimated to be US2.49/kg (Lydall *et al.,* 2017). The WPCB prices used in each of the respective sections of the plant was based on the interplant transfer prices calculated from the knowledge of the total operating costs associated with the section from which the WPCBs were derived.

No firm market prices were however available to use for comparing and assigning a working cost for the crushed WPCBs in the study of the group 4 business models. The interplant transfer price for crushed WPCBs from the size reduction plant, derived from the costing of the group 3 business models was therefore used as the entry cost for all economic evaluations associated with the group 4 business models.

7.2.2. Solid and waste management costs

The projected material flows of waste and effluent discharges were determined from the detailed mass balances of all the sections. The material flows were characterised and grouped according to the degree of toxicity of each stream. Using these classifications, solid wastes and effluent costs can be estimated from heuristics in literature (Turton *et al.,* 2009; van Wyk, 2014). The costs previously estimated in another study (van Wyk, 2014) were found suitable for use in this study. Non toxic material incurred a charge of US$0.04/kg. This factored in costs associated with material transfer and handling. Moderately and very toxic material incurred charges of US$0.04/kg and US$0.1/kg respectively.

7.2.3. Labour costs

Operating labour requirements were determined specifically for each section and the labour costs were estimated from the respective industrial average of the specified job descriptions. Online data from major South African recruiters was utilized (Indeed, 2018; 2018b; Payscale, 2018; 2018b).

Refer to appendix F, Table F.12 for the specific wage rates and salaries used for the study. For this process supervisory labour was found to be 14.79% of the operating labour which approximates closely to the reported estimate in literature of 15% (Peters & Timmerhaus, 1991; Perry *et al.*, 1997). This labour component was found to constitute about 62% of the total operating cost which was above the reported maximum of 50% in literature (Peters & Timmerhaus, 1991). The cost of labour associated with the effluent treatment plant was assumed not to have been captured in the overall waste and effluent treatment of each section. It was included in the overall wage bill of the entire operation.

Labour in the plant service sections

The two plant service sections associated with the designed process included the compressed air unit and the water softening section. Two man-hours each day were allocated to the softening water unit and the central compressed air supply.

A cost of operating labour was allocated to the compressed air and water softening sections. The provision of operating personnel specifically dedicated to the compressed air and water softening sections was not included in the organograms of each business model.

The water softening section was manned by the gold extraction plant operators. The compressed air unit was manned by the engineering servicemen. The cost of labour from these sections was not included to avoid double accounting. It was only calculated to determine the specific cost of softened water.

7.2.4. Utility costs

The costs that were categorized as utility costs include the compressed air costs, power, steam, water to the softening unit and fuel (coal). The costs were calculated using the process usage ratios calculated for the respective plants (refer to appendix F, Table F.9). The compressed air and softened water costs were calculated based on the operating costs associated with running the unit. Peas coal prices were determined from online quotations of typical vendor prices. The prices of water and power were based on industrial tariffs in the Johannesburg metropolitan area. A cost of US$2.63/$m^3$ of water was used in the calculations (johannesburgwater, 2017). The cost of power used was pegged at US$1.17/kWh (Eskom, 2018). A coal price of US$0.11/kg was used in this study. Steam costs were not estimated since it was not used in this process (Alibaba, 2018).

7.2.5. Indirect operating costs

Indirect operating costs were estimated using heuristics from literature (Peters & Timmerhaus, 1991; Perry *et al., 1997*). These costs included the plant and payroll overheads, local taxes and insurance and general operating expenses.

An allowance for contingency was provided for each of the operating sections. Table 7.6 provides a summary of the percentages and method used for evaluating the operating costs.

Table 7.6: Estimation of operating costs

Direct operating costs	Method for estimation (Peters & Timmerhaus, 1991; Perry *et al., 1997*)
Raw materials	Calculated from material balances, process usage ratios and vendor prices.
Process raw material losses	Calculated from material balances, process usage ratios
Waste and effluent treatment	Calculated from material balances, process usage ratios and heuristics in literature.
Utilities	Calculated from material balances, process usage ratios and relevant vendor prices.
Operating labour	Calculated from specific requires of each section and online labour costs from recruiting agencies.
Direct supervisory	14.79% of operating labour
Maintenance and repairs	6% of fixed capital cost
Operating supplies	15% of maintenance and repairs
Laboratory charges	15% of operating labour
Patents and royalties	4% of total operating costs
Indirect operating costs (fixed)	
Local taxes and insurance	3% of fixed capital cost
Payroll overheads	15% of operating labour
Plant overhead costs	125% of operating labour
General operating expenses	
Administration costs (other labour)	61.69% of operating labour
Distribution and selling costs	8% of total manufacturing costs
Research and development	4% of total manufacturing costs
Gross operating cost	
Contingency	3.80% of total operating costs

7.2.6.   Analysis of operating costs

The annual operating costs for each business model were evaluated and are summarized in appendix F, Table F.15 and Figures 7.4. An analysis of the data revealed that the direct costs of the group 4 business models were greater than the direct costs of the group 3 business models. This was attributed to the cost of crushed WPCBs used in the costing of the group 4 business models. The internal transfer price of crushed WPCBs was used for the calculation since no market data was available for comparison. The results did show that the reduction in the direct costs of the group 4 business models would be realised when the purchase cost of WPCBs remained at least 30% lower than the internal transfer price of crushed WPCBs.

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This would improve the financial performance of the group 4 business models when compared to the group 3 business models.
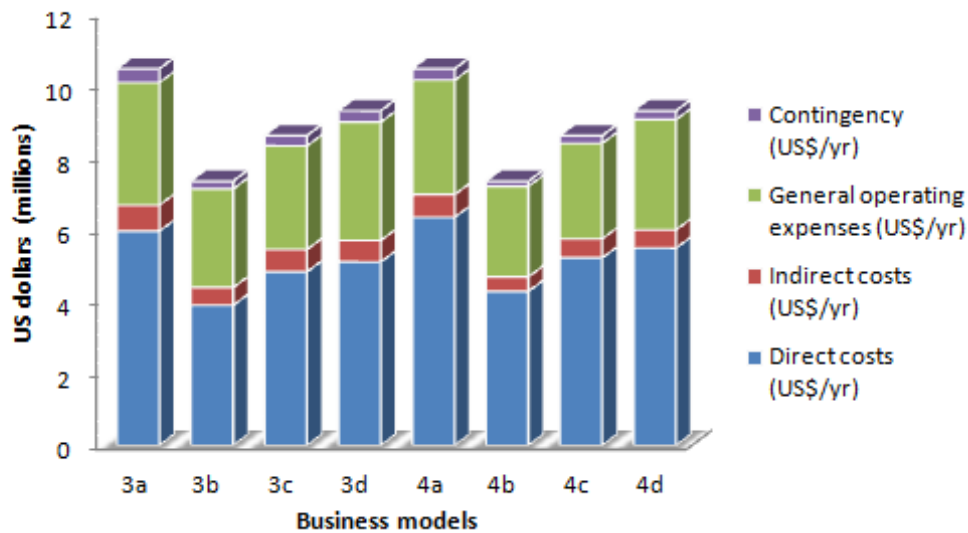
Figure 7.4: Graphical summary of the operating costs of each business model

An analysis of the indirect, general operating expenses and the contingency costs associated with the group 4 business models showed an expected decrease in comparison to the performance of the group 3 business models. This was due to the effect of eliminating the size reduction plant in the group 4 business models.

The overall picture showed that direct costs in all the business models contributed significantly to the annual total operating costs. The cost contribution of the annual direct costs associated with each of the business models ranged between 53% and 61% of the total annual operating costs. As a result the effect of the crushed WPCB price used in the costing of the group 4 business models had the effect of contributing towards the higher total operating costs associated with the group 4 business models.

The operating cost structures for each of the business models were interrogated further. It was observed that the top four operating cost drivers were associated with raw materials, effluent and waste treatment, sales and distribution and research and development. The detailed distribution of the costs is summarized in appendix F, Table F.16. The graphical summary of the distribution of the major operating cost drivers is depicted in Figure 7.5.
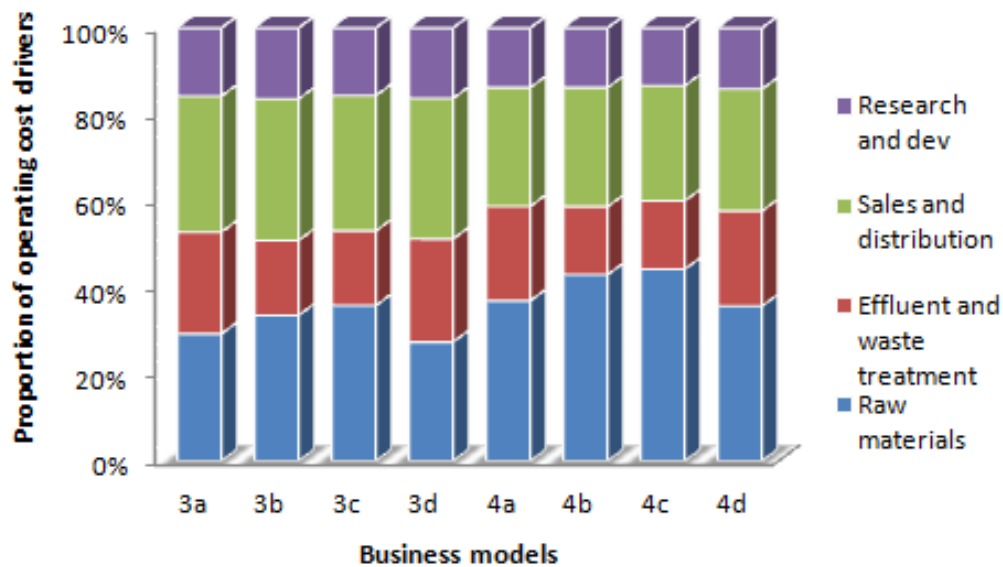
Figure 7.5: Percentage contributions of the major operating cost drivers

Operational leverage exists to manage the contribution of sales and distribution costs by establishing supply contracts wherein cyclic deliveries can be done over extended contractual agreements of up to a month as opposed to shorter time frames such as weekly deliveries. This fits well with the capacity of the operations under study which allows for the accumulation of product over a period of one month.

A well designed and engineered production process will also contribute towards trimming the contribution associated with the research and development costs. Little control however exists to manage the raw material costs of which, the costs of WPCBs were the significant raw material cost drivers. The prices of WPCBs within the operating framework of the South African e-waste recycling economy are governed strongly by supply and demand constraints. As the scarcity of the high value WPCBs increase so does the leverage of the supplier of the raw material. The prevalent raw material supply risk associated with WPCBs provides sufficient stimulus for supply driven inflation of prices.

As a new entrant into the recycling chain operating on a relatively low capacity, the bargaining power is very limited and potentially non-existent. The percentage contribution of the cost of WPCBs is shown in Figure 7.6 for each of the business models. The internal costs of effluent treatment can only be managed as far as the efficiency of the process is designed and operated.

External factors such as the existing regulations on Hazardous waste management and applicable statutes affecting the effluent and waste treatment operations cannot be dictated on both the tactical operational and corporate strategic levels. They are mandatory.
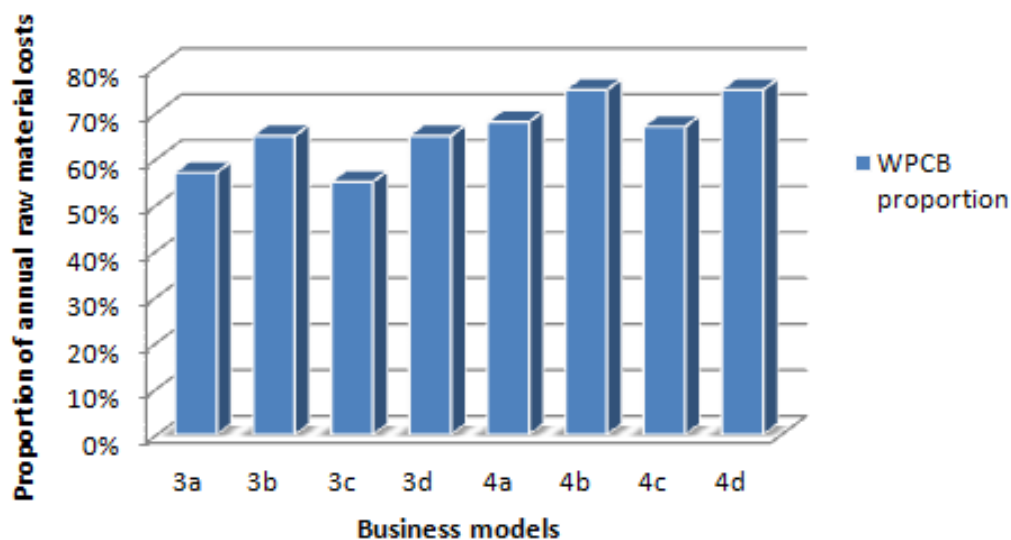
Figure 7.6: Contribution of WPCBs to the total raw material costs in each business model

The effluent and waste treatment costs contribution were highest in the business models, 3a and 4a which were associated with the recovery of both copper and gold in the metallic form. The lowest contribution from effluent and solid waste treatment was associated with the less operationally complex processes of business models 3b and 4b associated with gold and copper recovery as chemical intermediates in solution.

The comparative distribution of the operating costs is also depicted on a plant by plant basis as shown by Figures 7.7 through to 7.10. In both the group 3 and group 4 business models the significant operating costs were associated with copper extraction, nitric acid wash and gold extraction plants.

Moving across the graphs from the size reduction plant to the gold extraction plant, the observed trend was a progressive increase in the total operating costs associated with each plant. The increase was driven by the price of WPCBs. The value addition that occurred as the WPCBs were processed in each section was accompanied by a progressive increase in the interplant transfer prices of the WPCBs. To avoid double accounting, the internal transfer price of the WPCBs across these plants was therefore not used in the overall estimation of the annual operating costs of each business model. Only the WPCB purchase price at the point of entry into the system was used.
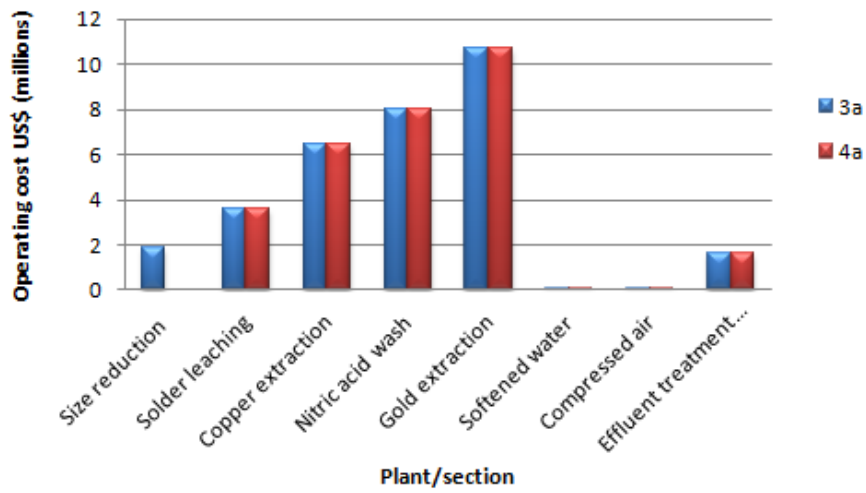
Figure 7.7: Comparison of plant operating costs for business models 3a and 4a
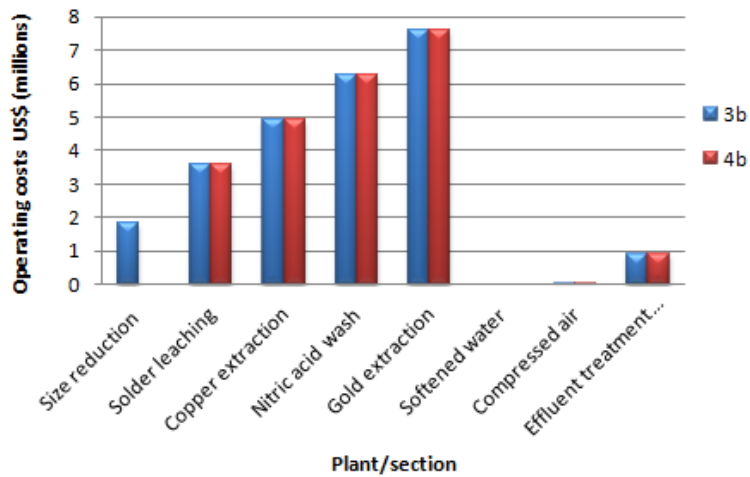
Figure 7.8: Comparison of plant operating costs for business models 3b and 4b
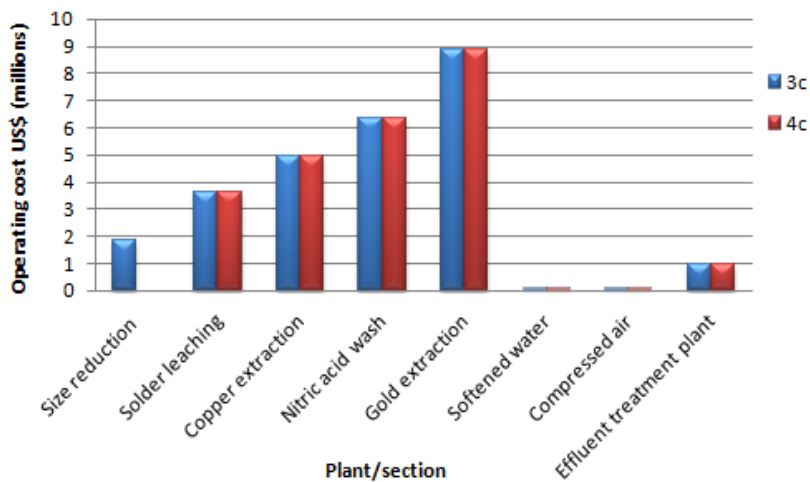
Figure 7.9: Comparison of plant operating costs for business models 3c and 4c
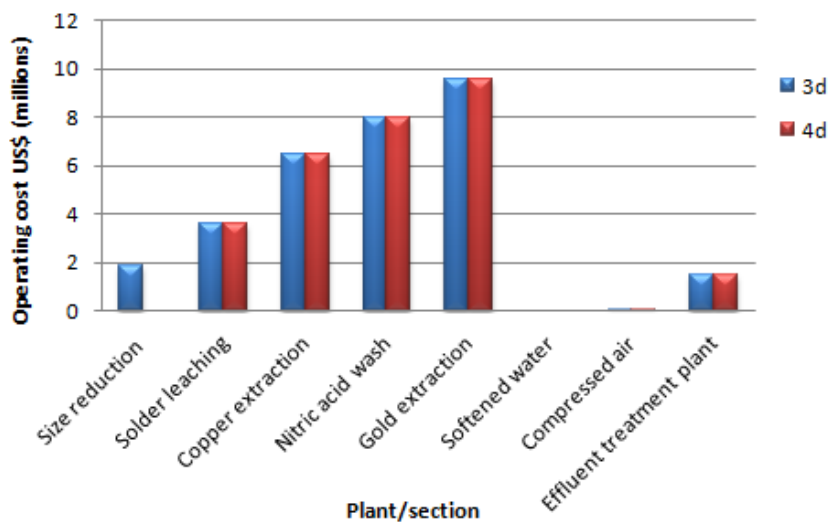
Figure 7.10: Comparison of plant operating costs for business models 3d and 4d

7.3.   Sales and revenue

In this study it was assumed that all the revenue was generated from sales of the gold and copper based products. It was also assumed that during the life of the project a fixed budget was used each year. This was based on the premise that sales volumes, market demand and raw material costs would remain fairly constant over the life of the project. The analysis of the specific effects of inflation, moving tax rates, sales projections reflecting market behaviour, were examined in depth in the sensitivity analysis of the selected business models.

7.3.1.   Product pricing

The methods available for allocating a price to a final product include full absorption pricing, marginal pricing, pricing on the basis of the cost of production and the rate of return pricing method (Perry *et al.,* 1997). The metals business is strongly market driven. The prevailing LME market prices of gold and copper metals were therefore used in the study. The market price of copper liquor was used as the comparative basis for evaluating the performance of the business models that produced copper liquor. The current market price for 30% copper sulphate solutions is US$460/tonne (Alibaba, 2018). The copper liquor produced in the process under study had a copper sulphate concentration of 14.26%. Based on the concentration of the solution in the liquor a selling price of US$218.65/tonne was used for the study.

No reference data was available to compare and benchmark the gold liquor selling price. A proportion of 80% of the gold content in the liquor was therefore used as the basis for allocating a price to the gold liquor. An evaluation of the competitiveness of the gold liquor price revealed that the cost of gold in the liquor exceeded the prevailing market price of gold by a margin of about 64%. This indicated that the gold liquor production cost was high.

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Reduction of the cost of gold in the liquor could only be achieved within the scope of the current process by the reduction of the proportion of the gold charged. Alternatively a reduction in the purchase costs of the crushed WPCBs would also lead to a reduction in the cost of producing the gold liquor.

A reduction in the proportion of the gold content charged in the product liquor price was associated with a decrease in gross revenues from the gold liquor business. The comparative performance of the gold liquor business against the prevailing market price of gold is illustrated in Figures 7.11 through to 7.14.
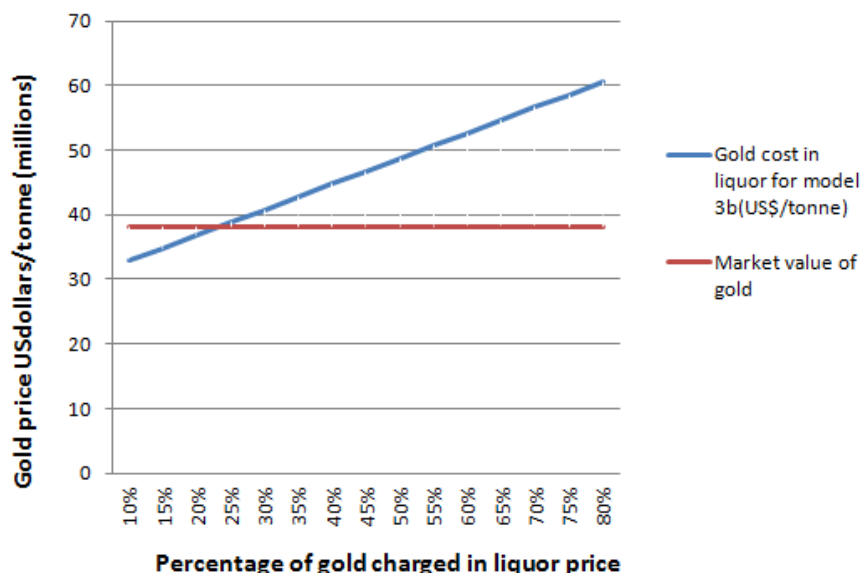
Figure 7.11: Comparing cost of gold in liquor versus market price of gold for business model 3b
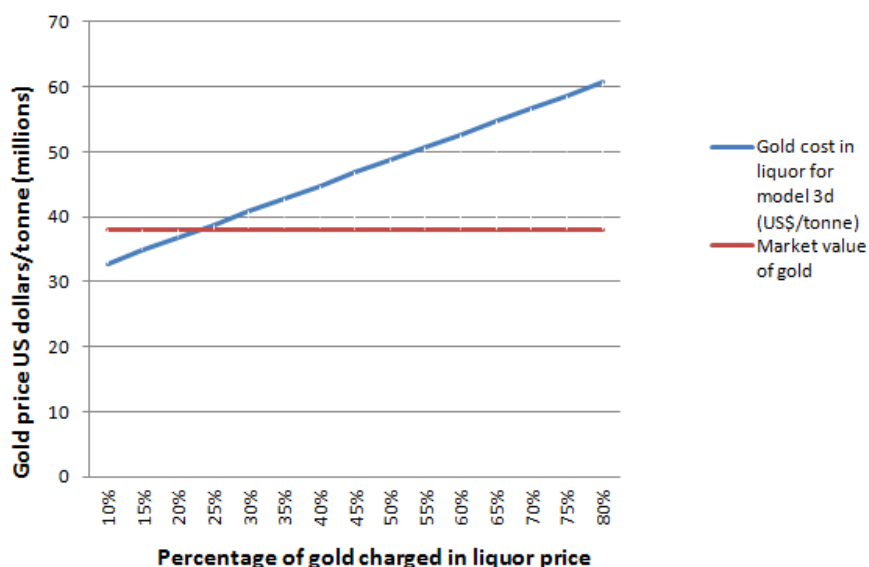
Figure 7.12: Comparing cost of gold in liquor versus market price of gold for business model 3d

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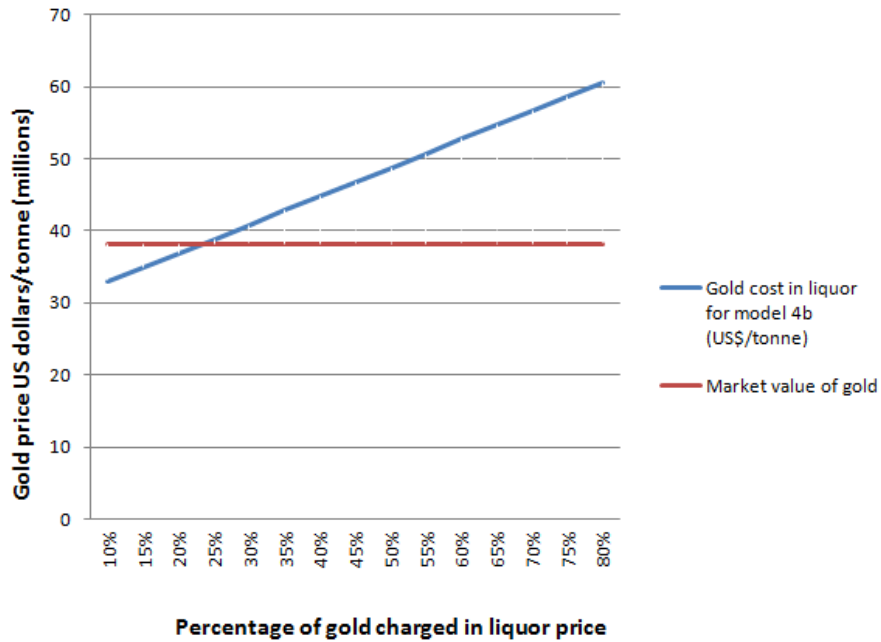
Figure 7.13: Comparing cost of gold in liquor versus market price of gold for business model 4b
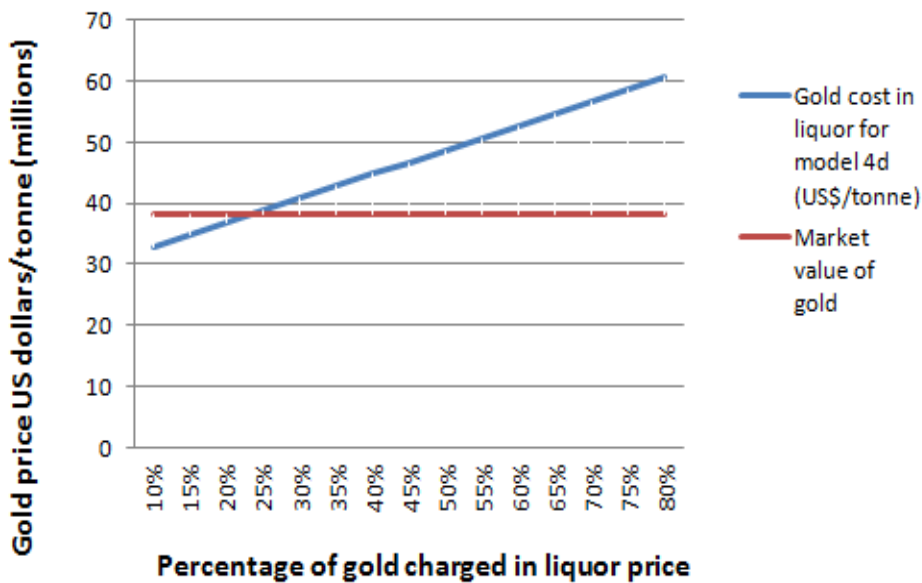
Figure 7.14: Comparing cost of gold in liquor versus market price of gold for business model 4d

An initial analysis of the graphical plots of the gold liquor market price versus the cost of gold in the liquor at the onset revealed that the gold liquor business is uncompetitive and using price as a market penetration strategy in the absence pre-existing buyer supplier contractual agreements, this business will fail. There will never be room for predatory pricing to protect this untapped niche in the event of emergent competition.

7.3.2. Handling product recovery streams in the process

7.3.2.1. Product recoveries

Revenue derived from the sale of gold and copper based products from the respective recovery streams was also factored into the economic evaluations. No consideration was however made of the potential revenue associated with other by-products that included: liquor streams from the solder leaching and nitric acid wash circuits and the dewatered solids exiting the gold extraction plant.

The recoveries of the target metals in each of the business models were evaluated from the material balances using the process considerations discussed in chapter 4. Product tracing was also done to account for the movement of the target metals for each of the business models. The detailed production volumes of the output streams is detailed in appendix F, Tables F.3 to F.6.

The recoveries for the copper metal as LME grade A cathodes and gold sludge of at least 98% purity in the main process were lower than the recoveries associated with the liquors respectively. The movement of the target metals in the process was analysed and the distribution of the metals in the process for each of the metals was found to be associated with tailings stream, scavenging and final product recovered for sale. The tracing of the target metals for each of the business models is summarized in appendix F, Tables F.17 and F.18. A plot of the distribution of the copper metal throughout the process of each of the business models is depicted in Figure 7.15.
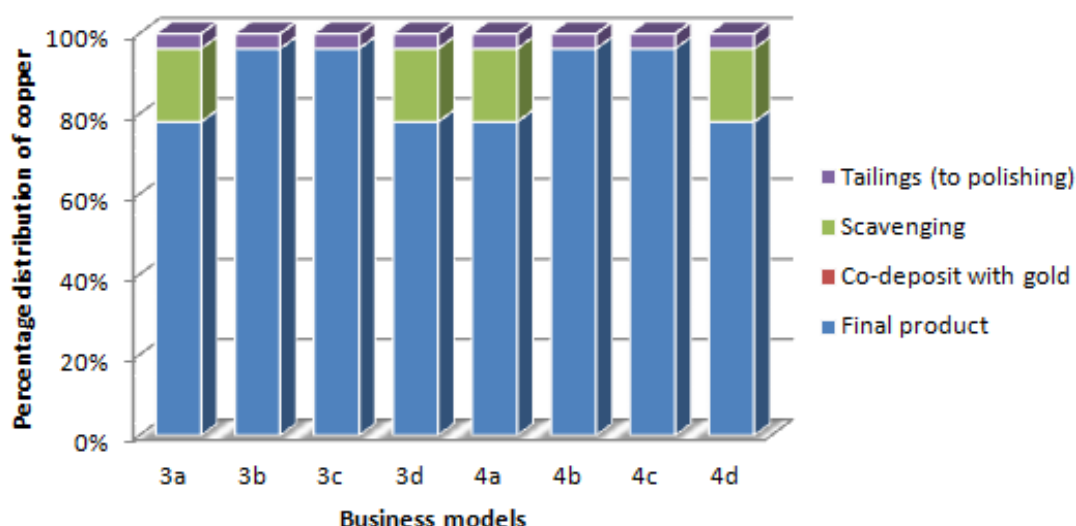
Figure 7.15: Accounting for the distribution of copper metal in each of the business models 3a to 4d

A plot of the distribution of the gold metal throughout the process of each of the business model is depicted in Figure 7.16. To improve the recoveries of both copper and gold in each of the business models the streams reporting to tailings and recycle were also directed to scavenging cells. The handling of the recovery streams is covered in detail in the sections to follow.
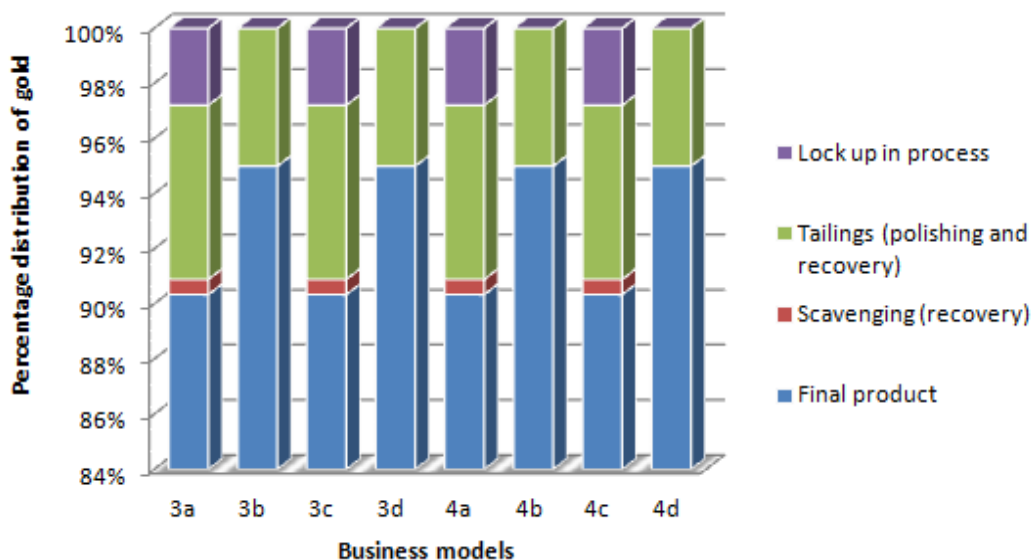
Figure 7.16: Accounting for the distribution of gold metal in each of the business models 3a to 4d

7.3.2.2.   Handling copper recovery streams

Copper liquor business

Copper in the form of copper sulphate reporting to tailings in the copper liquor business was not recovered as impound liquor. The tailings stream was found to contain high concentrations of lead, iron and nickel which would compromise the process efficiencies and final product qualities (see Table 6.4).

Copper cathode business

EMEW technology can be used to recover copper from solutions with copper concentrations low as 1g/L. Industrial experience with EMEW cell technology has shown that this technology is capable of producing LME grade A copper cathode or powder (Roux *et al.,* 2007). Based on the previous studies the copper concentrations in the liquor reporting to the scavenging circuit were expected to range between 20g/L and 35g/L (de Waal, 2018). Copper that reported to the scavenging circuit was recovered using EMEW cell technology depicted in the copper flow diagram (refer to appendix D, Figure D.4). In this study it was assumed that copper cathode plates were harvested from the electrowinning of this copper liquor stream in the copper solids business.

7.3.2.3.   Handling gold recovery streams

Gold liquor business

Gold that was in the entrained moisture (as sodium aurocyanide) contained in the dewatered filter cake exiting the leaching circuit was assumed to have been recovered as a dilute solution after sluicing with water.

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The weak sodium aurocyanide solution recovered was then directed back into the process as impound liquor thus enabling the recovery of the gold. In this study it was assumed that all the gold in the impound liquor was recovered. The recoveries of gold as sodium aurocyanide resulting from impound liquor generation and usage was applicable to the business models producing gold liquor as the final product for sale.

Gold solids business

RenoCell electrowinning technology is capable of achieving the recovery of precious metals from feed solutions containing metal concentrations as low as 1ppm (Renovare, 2011). The technology is useful for polishing tailings and effluent streams. In the gold solids business the gold was recovered from the impound liquor and the spent electrolyte. The spent electrolyte contained 2.61ppm of sodium aurocyanide (equivalent to 1.89ppm of pure gold) (Refer to mass the balances in appendix A, Table A.4). The two streams were blended and directed to the scavenging circuit that made use of RenoCell electrowinning technology as indicated in the gold flow diagram (Refer to appendix D, Figure D.9). It was assumed that gold sludge was recovered.

7.3.3. Revenue distributions

The comparison of the different revenue contributions on a product basis for each of the business models is depicted in Figure 7.17. The significant business units in each business model as depicted in the graphical plot are associated with gold related products. The performance of each of the models depends strongly on the success of the gold production, efficiency and recovery of gold and plant capacity utilization. The degree of sensitivity to the aforementioned factors was analysed in detail in the next chapter.
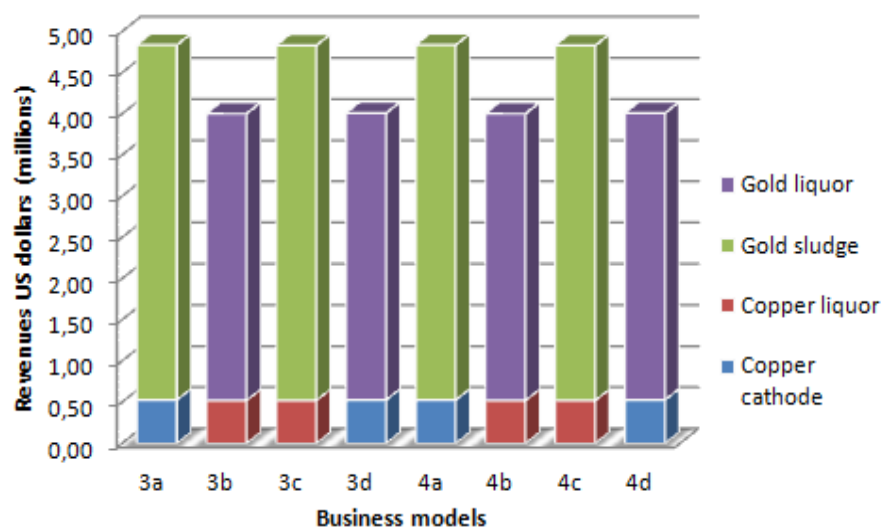
Figure 7.17: Gross revenue contributions from each of the business models

Table 7.7 Summarizes the revenue contributions associated with the gold business in each of the models.

Table 7.7: Contribution of gold revenues

Business model	4a	4b	4c	4d
Gold revenues contribution (%)	89.08	86.96	89.20	86.83

Business models associated with the production and sale of the gold and copper based products in the metallic state were associated with higher revenues. The observed trend in order of decreasing revenues for both the copper and gold businesses was shown as follows 4a > 4c > 4d > 4d. On the basis of revenue realisation, the production of gold in the metallic state, associated with business models 4a and 4c was the more lucrative option as opposed to the production of gold liquor. Similar remarks can be made regarding the copper business.

7.3.4. Evaluating economic efficiency of each business model

The analysis of WPCBs used in this study is summarized in Table 7.8 below.

Table 7.8: Analysis of WPCBs (adapted from Shuey & Taylor, 2004; Lydall *et al.*, 2017). The gold content was adjusted to 250ppm in this study

Metal price US$/tonne	Metal value	Composition	Intrinsic value US$
5 890.00	Cu	20.00%	1178.00
2 170.00	Al	2.00%	43.40
700.00	Fe	8.00%	56.00
20 950.00	Sn	4.00%	838.00
12 419.96	Ni	2.00%	248.40
2 638.05	Zn	1.00%	26.38
2 580.00	Pb	2.00%	51.60
	Mn	0.00%	0.00
	Sb	0.00%	0.00
38 104 430.00	Au	0.03%	9526.11
475 510.00	Ag	0.20%	951.02
30 237 780.00	Pd	0.01%	1511.89

The evaluation of the monetary value associated with the metals contained in the WPCBs was done and the results plotted in Figure 7.18.
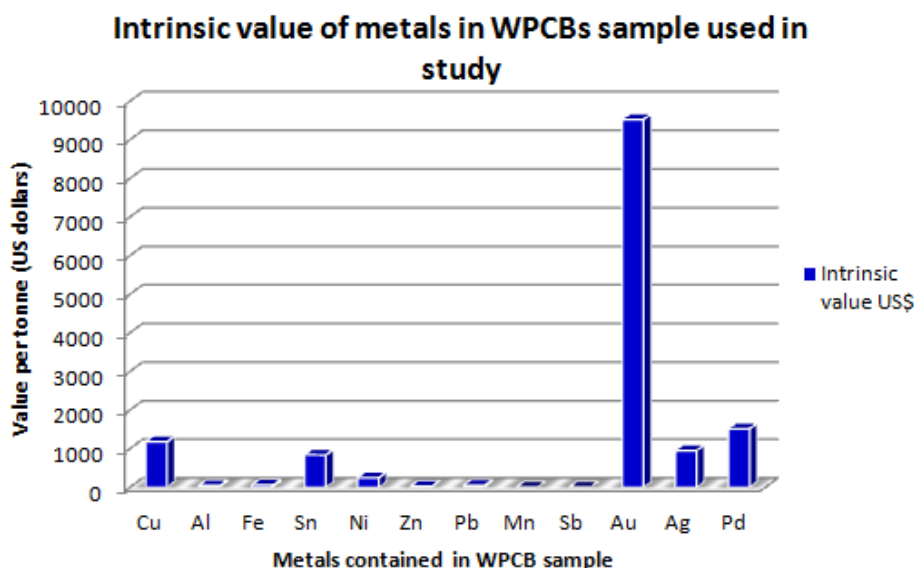
Figure 7.18: Value of metals in WPCBs (adapted from Shuey & Taylor, 2004; Lydall *et al.*, 2017). The gold content was adjusted to 250ppm for this study

Based on the plot in Figure 7.18 gold was the major revenue contributor accounting for 66% of the potential revenue realisable from metal sales in WPCBs. Potential revenue from copper sales accounted for an estimated 8%. A combination of sales from palladium, tin and silver revealed that gross turnover from these metals would account for approximately 23% of the potential revenue realisable from metals in WPCBs. Based on these observations the scope for the recovery of pure palladium, tin and silver was considered in the study. The impact on the viability of the business models of recovering palladium, silver and tin was discussed in section 7.4.3. An analysis of the economic efficiencies of the business models was undertaken. The overall economic efficiencies associated with each key revenue base were determined by working out the gross turnovers from copper and gold products as a percentage of the maximum realisable revenue of the combined metal sales in WPCBs. The economic copper and gold business efficiencies were calculated by expressing the realised revenues from the sales of the respective metal products as a percentage of the total realisable value of the respective target metal contained in the WPCBs. The results are summarized in Tables 7.9 to 7.11.

Table 7.9: Analysis of the economic efficiency of the copper and gold businesses

Overall metals efficiencies			
Business models	**Maximum revenue from all metals in WPCB**	**Actual revenues (US$/year)**	**Overall business efficiency (%)**
3a & 4a	6 601 251	4 836 231	73.26%
3b & 4b	6 601 251	4 001 948	60.62%
3c & 4c	6 601 251	4 829 960	73.17%
3d & 4d	6 601 251	4 008219	60.72%

Table 7.10: Analysis of the economic efficiency of the gold business

Economic gold business efficiency			
Business models	**Maximum gold revenue (US$/year)**	**Actual revenue (US$/year)**	**Gold business efficiency (%)**
3a & 4a	4 357 641	4 308 156	98.86%
3b & 4b	4 357 641	3 480 144	79.86%
3c & 4c	4 357 641	4 308 156	98.86%
3d & 4d	4 357 641	3 480 144	79.86%

Table 7.11: Analysis of the economic efficiency of the copper business

Economic copper business efficiency			
Business models	**Maximum copper revenue (US$/year)**	**Actual revenue (US$/year)**	**Copper business efficiency (%)**
3a & 4a	538 867	528 075	98.00%
3b & 4b	538 867	521 804	96.83%
3c & 4c	538 867	521 804	96.83%
3d & 4d	538 867	528 075	98.00%

An examination of the economic gold and copper business efficiencies revealed a strong correlation with the overall production process efficiencies and recoveries as saleable product associated with each product stream. An inspection of the data further revealed that superior economic efficiencies were associated with the recovery of copper and gold in the metallic state. This was attributed to the fact that these businesses were associated with higher revenues realised from the sale of the products as high grade metals. The incorporation of scavenging circuits contributed to the observed efficiencies. A detailed discussion of the product recoveries was covered in section 7.3.2.

7.4.   Profitability analysis

An initial analysis of the financial performance of all the business models was illustrated by a comparison of the annual gross revenues plotted against the annual operating costs. The performance data is summarized in appendix F, Table F.19 and depicted graphically in Figure 7.19. The analysis of the data revealed at this stage that the all the business models were incapable of financing operating activities without relying on external financing. This meant that the business models would run in a perpetual debt position.

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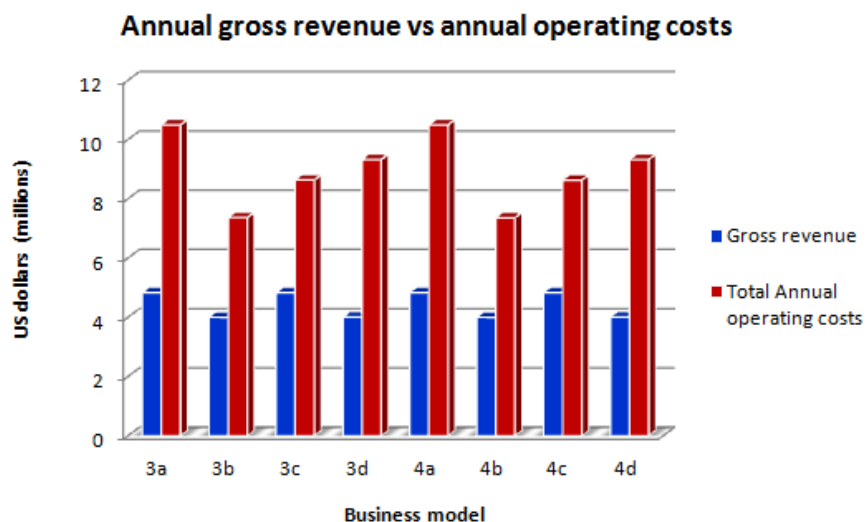
Figure 7.19: Evaluation of operating costs versus gross revenues of each business model

7.4.1.   Measures of profitability

Four categories of economic measures can be used for evaluating a project. The comparisons can be done based on discounted cash flows, time, cash and the basis of interest. Table 7.12 details the comparisons that are useful for decision making.

Table 7.12: Comparisons useful for economic decision making (Perry *et al.,* 1997)

Cumulative discounted cash flow or NPV curve for a specified discount rate	Comparisons based on time	Comparisons on the basis of cash	Comparisons on the basis of interest.
Maximum net outlay $(\sum A_{CF})_{max}$ Discounted breakeven point (DBEP). Plot of capital return ratio (CRR) for project life a discount rate at the cost of capital.	Duration of the project. Breakeven point (BEP) Discounted breakeven point (DBEP) Equivalent maximum investment period. (EMIP) Interest recovery period (IRP) Undiscounted payback period (PBP)	Maximum cumulative expenditure on the project $(\sum A_{CF})_{max}$ Maximum discounted cumulative expenditure on the project. Cumulative net annual cash flow $\sum A_{CF}$ NPV $(\sum A_{DCF})$ C_K capitalized cost	NPV DCFRR

In this study a combination of each of the measures was used to evaluate the project's performance to eliminate the possibility of bias associated with the use of at most two criteria. Table 7.13 summarises the performance measures used for evaluating the business models in this study.

Table 7.13: Summary of the definitions applied for the study evaluating projects (Perry *et al.,* 1997)

Measure	Application	Calculation	Key performance indicator used in the study
Discounted payback period	It is a measure of the time it takes for cash flows to recoup the original fixed capital expenditure. The earlier the capital is recovered the better the project is.	On a graphical plot it is the time period at which annual cash flows are equal to the depreciable fixed capital investment. $\sum A_{CF} = C_{FC}-S$ Where S is the scrap value.	< 4 years. The shortest time frame.
DCFRR	The investor's true rate of return	The interest rate yielding an NPV of 0 at the end of the project life.	The highest interest rate when compared with other projects being evaluated.
DCFRR$_{max}$	The maximum attainable rate of return for an infinite project life.	1/DPBP	The highest interest rate when compared with other projects being evaluated.
Net present value	Measures the actual profitability when the time value of money is taken into account. It is the sum of discounted cash flows for the life of the project.	NPV = $\sum A_{DCF}$ for life n years	Target an NPV> 0. The highest NPV.
Discounted breakeven point DBEP	Measure of the time it takes for the project to move from the capital risk zone to a point on the curve where y = 0.	Determined from a plot of $\sum A_{DCF}$ against time for the life of the project in years	< 4 years. The shortest time period.
CRR	Plot of capital rate of return ratio for the life of the project for a discount rate at the cost of capital.	CRR = $(NPV)/(\sum A_{CF})_{max}$ for A_{CF} < or = 0	Plots with the steepest gradient.
EMIP	Equivalent maximum investment period is the time frame during which the project debt would be outstanding if it were all incurred in one instant and all repaid in one instant. The shorter the EMIP the more attractive the project.	EMIP = area under the capital risk zone/$(\sum A_{CF})_{max}$ for A_{CF} < 0.	The smallest area under the capital risk zone.
IRP	Interest recovery period	On a plot of the graph of cumulative undiscounted cash flows against time yields a portion of the curve above $A_{CF} = 0$. The time period that results in the area above $A_{CF} = 0$ being equal to the area below $A_{CF} = 0$ is the IRP.	The shortest time period to cut the y=0 graph.
C$_K$	Capitalized cost. This is the amount of capital required to replace any piece of equipment in the project after it breaks down. This should be perpetual.	$C_K = [C_F - (S/(1+i)^n)] \times [(1+i)^n/((1+i)^n - 1)]$	The least capitalized cost

The results of the evaluation based on the metrics described in Table 7.13 are summarized in Tables 7.14 and 7.15 for the group 3 and 4 business models respectively.

Table 7.14: Performance evaluation of group 3 business models

Models	3a	3b	3c	3d
Key products	Copper cathode LME grade A Gold sludge >98%	Copper sulphate liquor 3.71% Cu Gold cyanide liquor, 142.52ppm Au	Copper sulphate liquor 3.71% Cu Gold sludge >98%	Copper cathode LME grade A Gold sludge >98% Gold cyanide liquor 142.52ppm Au
NPV	-$43 661 431	-$20 805 697	-$29 284 288	-$35 834 822
Capitalized cost	$3 078 960	$715 290	$2 129 930	$1 772 789
DCFRR no infl.	Unviable	Unviable	Unviable	Unviable
DPBP no infl.	Unviable	Unviable	Unviable	Unviable
EMIP	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.
DCFRR max	Unviable	Unviable	Unviable	Unviable
IRP	No recovery	No recovery	No recovery	No recovery
CRR	Decays in capital risk zone	Decays in capital risk zone	Decays in capital risk zone	Decays in capital risk zone
DBEP	Will not break even	Will not break even	Will not break even	Will not break even

Table 7.15: Performance evaluation of group 4 business models

Models	4a	4b	4c	4d
Key products	Copper cathode LME grade A Gold sludge >98%	Copper sulphate liquor 3.71% Cu Gold cyanide liquor , 142.52ppm Au	Copper sulphate liquor 3.71% Cu Gold sludge >98%	Copper cathode LME grade A Gold sludge >98% Gold cyanide liquor 142.52ppm Au
NPV	-$43 472 752	-$20 614 882	-$28 990 635	-$35 644 029
Capitalized cost	$3 037 995	$674 325	$1 984 806	$1 731 824
DCFRR no infl.	Unviable	Unviable	Unviable	Unviable
DPBP no infl.	Unviable	Unviable	Unviable	Unviable
EMIP	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.	Not done. Unviable. The project debt will remain outstanding.
DCFRR max	Unviable	Unviable	Unviable	Unviable
IRP	No recovery	No recovery	No recovery	No recovery
CRR	Decays in capital risk zone	Decays in capital risk zone	Decays in capital risk zone	Decays in capital risk zone
DBEP	Will not break even	Will not break even	Will not break even	Will not break even

Net present value

The analysis of the profitability assessments for the group 3 and 4 business models based on the net present value (NPV) indicates that all the business models associated with the project are unviable. Business models 3a and 4a with the highest capitalization costs showed the worst performance. Business models 3d and 4d showed an improvement of just below 19% when compared to business models 3a and 4a. This observed behaviour was due to the comparatively lower annual operating costs and lower capital investment associated with business models 3d and 4d.

The superior performance of business model 3c and 4c compared to business models 3a and 4a was attributed to the effect of high gross revenues associated with solid gold sales and the comparatively lower initial capital outlay and operating costs discussed in sections 7.1 and 7.2. The gross revenues of business model 3c and 4c surpassed those of models 3d and 4d allowing them to perform better as a result. Business models 3b and 4b had the lowest initial capital outlay and operating costs. As a result these models had the highest NPV values in comparison to the aforementioned models.

Capitalization costs

The business models were compared using the total capitalization costs. In this case it was desirable to have the lowest capitalization cost. The lowest capitalization costs were associated with the business models that produced the solutions of the intermediate metal salts as the final products, in this case business models 3b and 4b. The business models 3a and 4a had the highest capitalization costs because of the higher capital investment associated with these models.

Discounted cash flow rate of return

The performance of the business models based on the discounted cash flow rate of return revealed that none of the business models associated with the proposed process would realise a positive return on the capital invested. The plots of the cumulative discounted cash flows for each of the business models are illustrated in Figure 7.20.

Cumulative discounted cash flows for each business model 3a to 4d
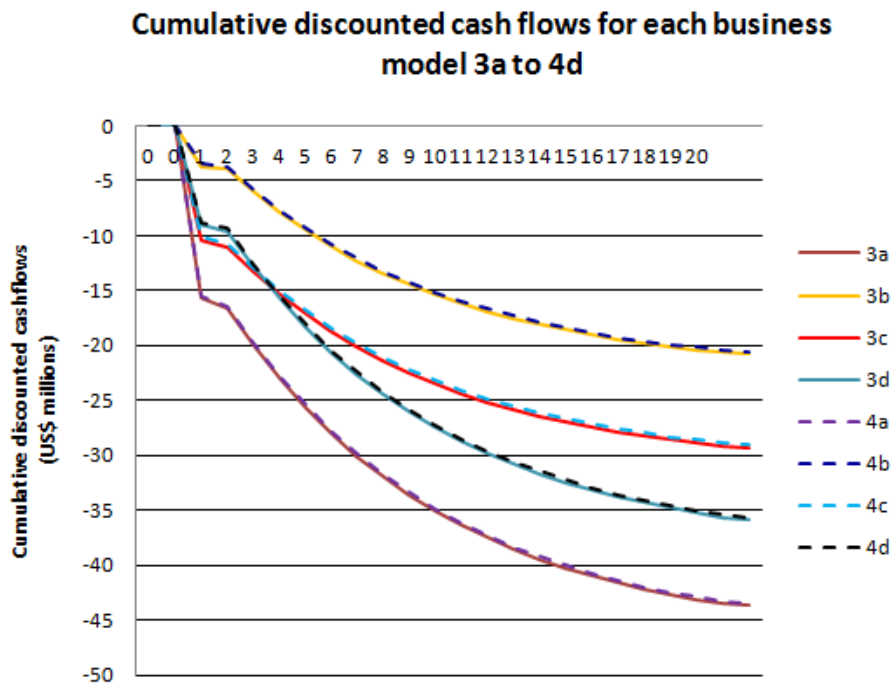
Figure 7.20: Comparison of discounted cumulative cash flows for each of the models (3a-4d)

None of the curves of the business models, 3a to 4d, moved out of the capital risk zone, that is, the region below y=0 or $\sum A_{DCF} = 0$. All the curves continued to decay throughout the life of the project indicating an increased debt position as the years progressed. Business models 3a and 4a displayed the most significant risk to capital invested as indicated by the greatest area in the capital risk zone.

Discounted payback period

Scanning the shape of all the cumulative discounted cash flow curves of each business model, it is clear that none of the business models will ever break even for the duration of the project life. It can be seen that when running the project using any of the business models, the project is incapable of paying back the capital invested.

Investor's true rate of return DCFRR_max

The calculation of the investors' rate of return could not be done for any of the business models since all the business models were found to be incapable of paying back the capital invested. Based on the results in Tables 7.14 and 7.15 no gain will be realised from this from project under the specified operating conditions. The same will hold true in the case when the business is run with an infinite project life. The curves displayed in Figure 7.20 will maintain the trend of decay for an infinite number of operating years.

Equivalent maximum investment period

The curves in Figure 7.20 remained in the capital risk zone. There is no point at which any of the curves in Figure 7.20 shifted into the capital risk free region.

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Based on the shape of the curves, should all the capital invested be due in a single instant, none of the business models under the conditions of operation were capable of repaying the capital at any stage of the project.

Interest recovery period

Plots of the cumulative undiscounted cash flows against time were done for each of the business models and are depicted in Figure 7.21. On a plot of the graph of cumulative undiscounted cash flows against time yields a portion of the curve above $\sum A_{CF} = 0$. The time period that results in the area above $\sum A_{CF} = 0$ being equal to the area below $\sum A_{CF} = 0$ is the interest recovery period (IRP).
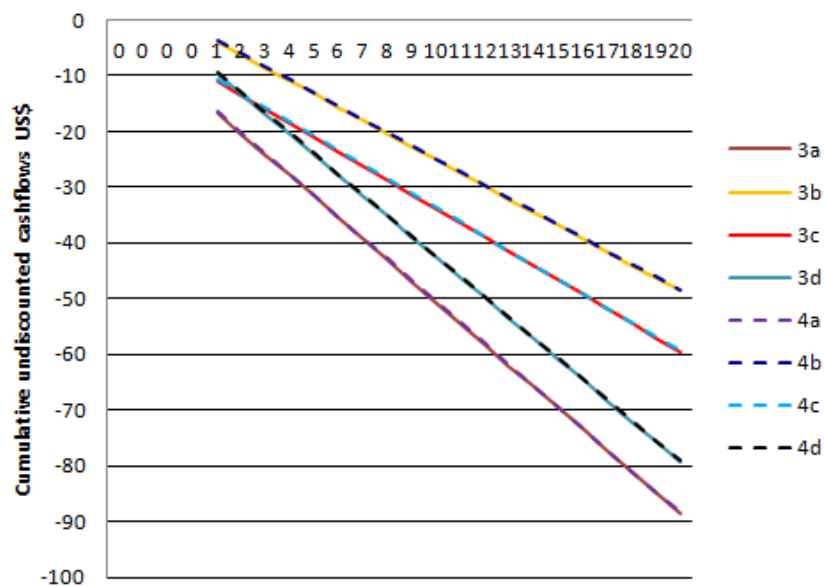
Figure 7.21: Plots of interest recovery period (IRP) against time for each of the models (3a-4d)

In the case of the business models under study, the plots of IRP against time for the entire duration of the project showed that the curves continued to decay. There was no point of inflection leading to the possibility of any of the curves crossing into the capital risk free region. This behaviour was strongly indicative of the fact that none of the business models at the current concentration of gold of 250ppm in the WPCBs feedstock, would at any stage of the project be in a position to recover the interest on the capital invested. Using the IRP curves as criteria for assessing each business model, it is advisable not to pursue the project operating with any of the business models.

Capital rate of return ratio

The net present value divided by the maximum initial capital outlay yields the capital rate of return ratio. When this ratio is used to develop a plot over the life of the project it is desirable to generate a curve that increases rapidly in the capital risk free region (area above y=0).

Plots of the capital rate of return ratios (CRR) against time were done for each business model as shown in Figure 7.22. The trend displayed by all the curves displayed a perpetual decay maintained in the capital risk zone. This observed trend for the business models based on CRR plots against time was indicative of the fact that no positive returns would be realised for the capital invested at any stage of the life of the project. This trend supports similar observations made when plots of IRP were made against time (Figure 7.21).
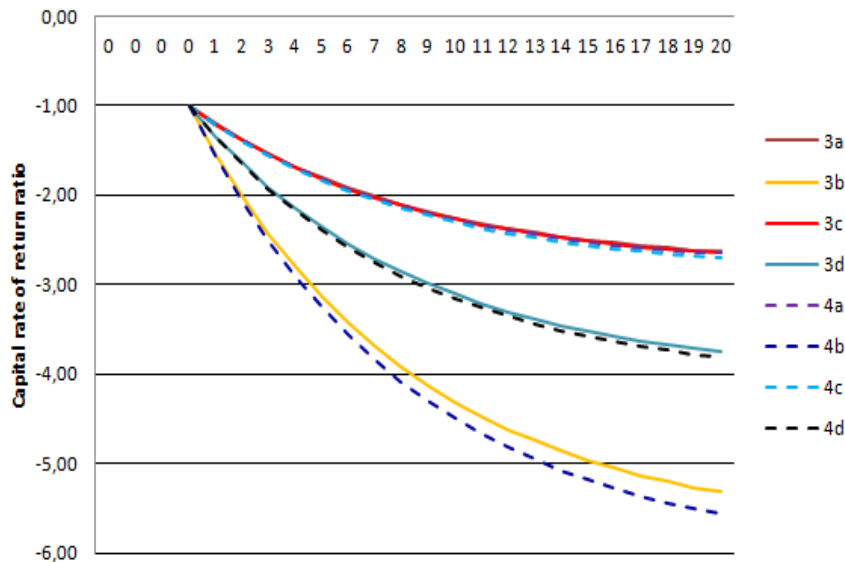
Figure 7.22: Plots of capital rate of return ratio (CRR) against time for each of the models (3a-4d)

On the basis of the plots of the DCFRR, IRP, CRR and the evaluation of DPBP, EMIP, $DCFRR_{max}$, NPV and capitalized costs, the project operating with any of the business models is unviable. All the measures used, indicate that investing in the proposed process under the specified operating conditions will result in a loss of capital that can never be recovered. Further interrogation of the business models was done to dissect the annual performance of the business models by use of profitability ratios outlined in section 7.4.2.

7.4.2.   Profitability ratios

The performance of the business models was also analysed by making use of additional profitability ratios. In this study six ratios were used and are summarized in Table 7.16 below:

Table 7.16: Profitability ratios used in the study (Perry *et al.,* 1997)

Ratio	Relevance	Performance indicator
Net profit margin	Measures the actual amount of money earned for the sales of product made.	>1 the project is performing well NPM = 0. The business is at breakeven
Return on investment	Measures the actual earning power for every dollar invested into the project.	>1 the project is performing well. A greater ratio is desirable.
Capital ratio	Measures the extent to which sales are generated from the fixed capital invested. Computationally worked as C_{FC}/A_s.	0< x < 1. A smaller ratio when comparing different projects or operating configurations is desirable. CR < industrial target.
Capital rate of return ratio (CRR_{term}) (end of project life)	Provides a measure of the monetary gain in the capital invested into the project.	CRR_{term}< 0 project is unviable, reject. CRR_{term} = 1, consider alternative and more profitable options. Project is capable of recovering invested capital and repaying interest. CRR_{term}> 0 Project is viable
Fixed asset turnover ratio	This ratio provides a measure of the operating performance and effectiveness in utilizing fixed assets to generate revenue.	>1. A greater ratio when comparing different projects or operating configurations is desirable. CR > industrial or set target.
Present value ratio	It is useful for measuring the economic relevance of a project.	PVR> 1 economically satisfactory PVR = 0 economic breakeven of project PVR< 0 economically unsound

The analysis of the business models based on these ratios was undertaken and the results are summarized in Tables 7.17 and 7.18 below.

Table 7.17: Analysis of the performance of group 3 business models using profitability ratios

Business models	3a	3b	3c	3d
Net profit margin (NPM)	-78.21%	-58.78%	-52.70%	-91.36%
Return on investment (ROI)	-0.23	-0.60	-0.23	-0.38
Capital ratio (CR)	2.21	0.63	1.46	1.54
Fixed asset turnover ratio (FATR)	0.45	1.58	0.68	0.65
Terminal capital rate of return ratio (CRR_{term})	-2.63	-5.31	-2.64	-3.74
Present value ratio (PVR)	-1	-1	-1	-1

Table 7.18: Analysis of the performance of group 4 business models using profitability ratios

Business models	4a	4b	4c	4d
Net profit margin (NPM)	-78.26%	-58.82%	-52.74%	-91.41%
Return on investment (ROI)	-0.23	-0.67	-0.26	-0.41
Capital ratio (CR)	2.18	0.60	1.43	1.50
Fixed asset turnover ratio (FATR)	0.46	1.67	0.70	0.67
Terminal capital rate of return ratio (CRR_{term})	-2.65	-5.55	-2.69	-3.80
Present value ratio (PVR)	-1	-1	-1	-1

Capital ratios (CR) and fixed asset turnover ratios (FATRs) of a concern must be compared against set targets within the industry. For this particular process however, no industrial targets were available for benchmarking the performance of the models. In this study the benchmarks were derived using the result of the sensitivity analysis performed around the concentration of gold in the WPCBs (see section 8.4.8). The results of the benchmarking the capital ratios and fixed asset turnover ratios are summarized in Table 7.19.

Table 7.19: Benchmarking capital ratios, FATRs and minimum performance of each business model

Business model	Minimum gold content	Minimum target sales per year	Target CR	Target FATR
3a	875ppm	15606621	0.68	1.46
3b	563ppm	8 359 089	0.30	3.31
3c	625ppm	11 292 194	0.62	1.60
3d	925ppm	13 404 608	0.46	2.18
4a	875ppm	15 606 621	0.67	1.48
4b	563ppm	8 359 089	0.29	3.49
4c	625ppm	11 292 194	0.61	1.63
4d	925ppm	13 404 608	0.45	2.23

The target minimum gold content in each business model used to evaluate the minimum sales required in each business model. The minimum sales value was then used in calculating industrial target ratios for both the CR and FATR against which each of the business models were compared.

Table 7.20: Summary of the results of the analysis of annual business performance of the group 3 business models

Business model and ratio	Net profit margin (NPM)	Return on investment (ROI)	Capital ratio (CR)	Fixed asset turnover ratio (FATR)	Terminal capital return ratio (CRR$_{term}$)	Present value ratio (PVR)
3a	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.23 for every dollar invested.	CR was greater than the target of 0.68. It was 3.25 times greater than the target. Annual sales in this business model are not sufficient to ensure viability of this business model.	FATR was less than 1 and below the target of .,46 by a margin of 324%. This indicates that the fixed assets were unable to generate revenue sufficient to shift and maintain the business in a profitable position each year under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
3b	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.60 for every dollar invested.	CR 2.1 times above the set target. Annual sales will not ensure profitability of the business model.	While FATR was greater than 1 however, a comparison against the minimum target revealed that the FATR was 209% below the target. The model is unviable.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
3c	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.23 for every dollar invested.	CR 2.35 times greater than the target. Business model unviable at this level of sales.	FATR is below the target by a margin of 235%. This model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
3d	The NPM is less than 1. This business model will run at an annual loss. This business model is the least profitable of the group 3 models.	The return on investment is negative. This business model will run in an annual loss position of US$0.38 for every dollar invested.	CR is greater than 1 and 3.35 times greater than the target. Business model unviable at this level of sales	The business model is 335% below the target. This model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.

Table 7.21: Summary of the results of the analysis of annual business performance of the group 4 business models

Business model and ratio	Net profit margin (NPM)	Return on investment (ROI)	Capital ratio (CR)	Fixed asset turnover ratio (FATR)	Terminal capital return ratio (CRR$_{term}$)	Present value ratio (PVR)
4a	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.23 for every dollar invested.	CR is 3.25 times above the target and greater than 1. Annual sales in this model will not guarantee positive returns each year.	This business model is 3.22 times less than the target. The model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
4b	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.67 for every dollar invested.	The capital ratio is 2.07 times above the target of 0.29. The business is unprofitable and uncompetitive at this level of sales.	This business model is 2.09 times less than the target. The model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
4c	The NPM is less than 1. This business model will run at an annual loss.	The return on investment is negative. This business model will run in an annual loss position of US$0.26 for every dollar invested.	The capital ratio is above the target ratio of 0.61 by a margin of 234%. At the current level of annual sales of the business model is unprofitable.	This business model is 2.33 times less than the target. The model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.
4d	The NPM is less than 1. This business model will run at an annual loss. It is the least profitable of all the models.	The return on investment is negative. This business model will run in an annual loss position of US$0.41 for every dollar invested.	The capital ratio is 3.33 times greater than the set target of 0.45. At the current level of annual sales this business model is unviable.	This business model is 3.33 times less than the target. The model is unviable under the specified operating conditions.	The CRR$_{term}$ for this model was less than 1 indicating that the net annual returns during the life the project up to maturity were negative.	The PVR was less than 0. This model is economically unviable and is an unsuitable investment option under the current operating conditions.

Overview of findings of the analysis of profitability ratios

The net profit margins for all the business models were all less than 1. This showed that all the business models on the basis of the net profit margin were unviable. A comparison of the NPM ratios revealed that business models 3d and 4d and 3a and 4a that had the lowest profit margins. This paralleled the observations made when the performance of these business models was analysed on the basis of the fixed asset turnover ratios (FATR).

An analysis of the CR and FATR values indicated that business models 3b and 4b had the lowest deviations from the bench mark performance targets detailed in Table 7.19. This indicated that the aforementioned business models displayed superior performance in comparison to the other business models in terms of operating performance and utilization of fixed capital in generating sales. On the basis of the observed CR and FATR values, using high grade WPCBs feedstock this business models 3b and 4b would be the most desirable investment option to consider.

The return on investment calculated for each of the business models showed a consistent annual loss position for each of the business models. The worst performance was displayed by business models 3b and 4b associated with the production of gold and copper liquors. These business models had the lowest annual sales revenues. Business models associated with the production of gold in the metallic state performed better.

The capital ratios of all the business models were below the minimum targets indicating that all the business models were incapable of generating positive returns each year for the life of the project. The CRR_{term} and PVRs associated with each business model were all negative. This behaviour was congruent with the results of the analysis of the performance curves of DCFRR, IRP and CR in section 7.4.1.

Comparisons of the operating costs of group 3 and 4 business models reveal that the reduction in the purchase price of crushed WPCBs will result in superior performance of the group 4 models. The elimination of the size reduction plant in the group 4 business models also resulted in a reduction in the CAPEX and associated overheads. It is on this basis that the group 3 business models were eliminated from further study. The group 4 business models were selected for a detailed sensitivity analysis covered in chapter 8.

7.4.3. Exploring options for improving performance

An evaluation of the additional revenue streams that could be realised from sales of other metals was undertaken. Based on the analysis of value in the WPCBs used in this study, it was found that silver, tin and palladium were also relevant contributors to revenue, contributing an estimated 14% of potential monetary revenue from the sale of these metals.

A mean business efficiency of 70% was used to approximate turnover from the tin, silver and palladium businesses. No operating costs were allocated and as a result the data obtained modelled an ideal business scenario that minimized overall annual operating costs for each of the business models analysed. The objective was to establish the extent to which the loss position would be contracted when revenues were increased by sales of tin, silver and palladium metals. In the first scenario the palladium had a composition of 50ppm based on the assays from previous work (Shuey & Taylor, 2004). The potential gross annual revenues from the sales of tin, silver and palladium metals are summarized in appendix F Table F.24 and depicted in Figure 7.23.
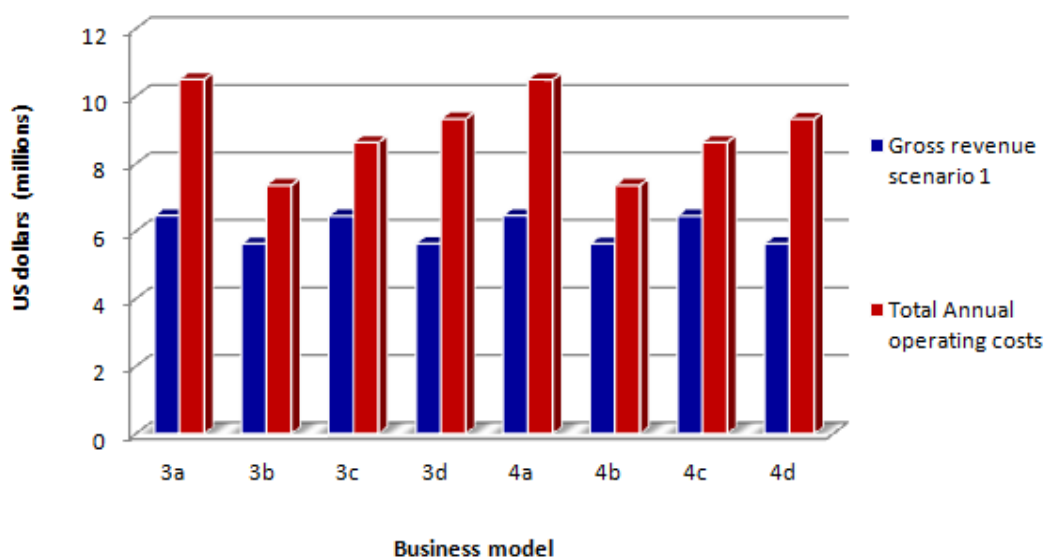
Figure 7.23: Assessing the impact of increasing revenue from palladium, silver and tin recovery (scenario 1 Pd 50 ppm)

Based on results of previous investigations in literature studies, a maximum palladium concentration of 294ppm has been analysed in WPCBs (Ogunniyi & Vermaak, 2009). Using this concentration as the basis of the second assessment, a similar analysis was undertaken whereby the palladium concentration in the WPCBs was increased to 294ppm. A mean business efficiency of 70% was again used to approximate gross turnovers from the tin, silver and palladium businesses. The costs of recovering the metals were not factored. Figures 7.23 and 7.24 therefore illustrate the ideal scenarios which are still not lucrative for all the business models understudy. It is reasonable to expect the magnitude of variance between the gross revenues and overall operating costs to increase further when the objective of tin, silver and palladium recovery is the metallic state. An analysis of the CAPEX and OPEX structures of the business models 3a and 4a are strong indicators that the inclusion of the recovery sections for these additional metals in the metallic state will be associated with high CAPEX and OPEX budgets. The potential gross annual revenues from sales of tin, silver and palladium metals are summarized in appendix F, Table F.25.

The results are depicted graphically in Figure 7.24. The additional operating costs related to the recovery of these metals were not included in the analysis. As a result the data obtained reflected a best case scenario that again minimized overall annual operating costs for each business model.
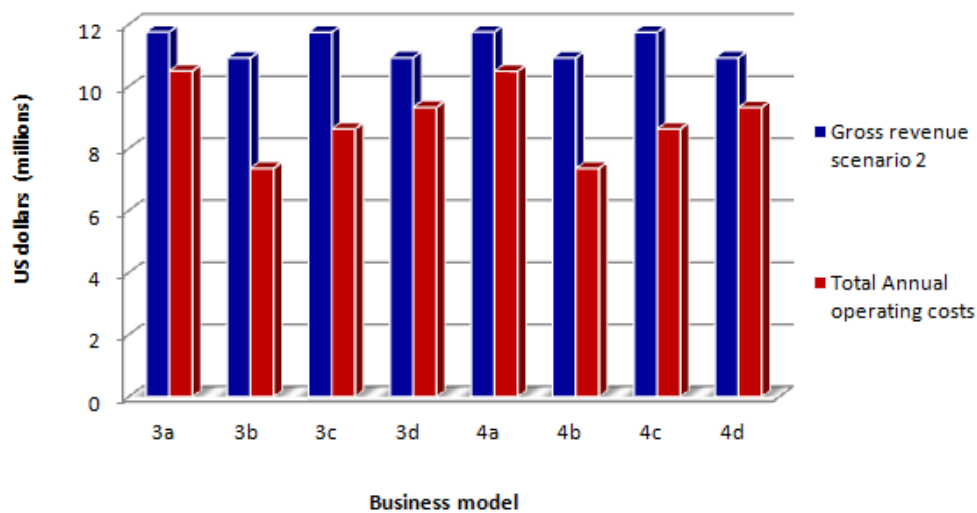
Figure 7.24: Assessing the impact of increasing revenue from silver, tin and palladium recovery (scenario 2 with maximum Pd (294ppm))

In the first scenario, an inspection of the comparative gross turnovers plotted in Figure 7.23 against the current operating costs of each business model revealed only a marginal improvement in the performance of each business model. The contribution of the additional revenue from tin, silver and palladium sales was however unable to shift any of the business models into the profitable position. The pursuit of further refining the silver, tin and palladium by-product streams is therefore not an advisable undertaking. The expansion of the operating portfolio of the proposed venture by installation of separate facilities for the recovery of pure silver, tin and palladium will be unprofitable.

In the second scenario (Figure 7.24) the increased palladium content in the hypothetical WPCBs feedstock showed an improvement in the performance of each business model. Business models 3a, 3c, 4a and 4c showed the most benefit from the increased palladium content. The gross revenues exceeded the annual operating expenses by an average margin of approximately 21%. The low average margin between the gross revenues and annual operating costs indicates that all the business models have a strong likelihood of remaining unviable when the additional operating costs for the recovery of palladium, silver and tin are factored. The additional revenue from the recovery of these metals at a gold concentration of 250ppm in the WPCB feedstock with a high palladium content does not improve the performance of the project. The expansion of the operating portfolio of the proposed venture by installation of separate facilities for the recovery of these metals will increase the capital risk of each business model.

7.5.   Chapter summary

The economic performance of the project was examined in detail in this chapter. It was split into sections that covered a brief discussion relating to the techniques available for capital cost estimates, the estimation of the capital expenditure, estimation of operating cost expenditures, revenue and sales estimation and the analysis of project profitability. All the costs were determined at full operating capacity, on a fixed budget with the added assumption that all the material produced was sold.

The study estimate for preliminary project evaluations was used. Capital cost estimates were calculated for each of the remaining business models 3a to 4d. The highest capital expenditures were associated with the models 3a and 4a that entailed the recovery of both copper and gold in the solid state.

The estimation of the operating costs for each section was undertaken. In this section the distribution of the costs was examined. Major cost drivers in each of the business models were identified to examine the potential areas for trimming and potentially modifying the process. Similar to the observations made in the capital cost estimates, the business models associated with the recovery of the copper and gold in the metallic state had the highest operating expenditures. The movement of the target metals in the process was also analysed.

The prevailing market prices of gold and copper metals were used in the study. The market price of copper liquor was used for evaluating the performance of the business models that produced copper liquor. No reference data was available to compare and benchmark the gold liquor selling price. A proportion of 80% of the gold content in the liquor was therefore used as the basis for pricing the gold liquor. An evaluation of the competitiveness of the gold liquor price revealed that the cost of gold in the liquor exceeded the prevailing market price of gold by a margin of 64%. This indicated that the gold liquor production cost was high.

Reduction of the cost of gold in the liquor could only be achieved within the scope of the current process by the reduction of the proportion of the gold charged in the final selling price of the liquor produced and utilizing crushed WPCBs with a purchase price lower than the transfer price used in the group 3 business models. Reduction of the gold proportion charged in the liquor however increased the loss position of the gold liquor business. Indications at the onset showed that the gold liquor business was uncompetitive. Penetration pricing of the liquor was a weak market entry strategy. There will never be room for predatory pricing to protect this untapped niche in the event of emergent competition.

An initial assessment of the performance of each of the business models was done by comparing the gross revenue to the total operating costs. The graph of these comparisons revealed that the business models were all unviable at the current gold concentration of 250ppm in the WPCBs.

This initial assessment was reinforced by the results of the profitability analysis. It was established that all the business models to varying degrees, as reflected by the curves in the recapitalization risk analysis, CRR, IRP and DCRFF plots, were incapable of repaying the interest on the capital, repaying the capital invested and continuing operations for the full term of the project.

It was also established that the operating costs were in excess of the gross revenues by margins ranging between 79% and 133%. These findings were further reinforced by a detailed analysis of the annual performance ratios associated with each business model. On the basis of these findings, investing in the proposed process without improvement in the revenues is not advisable. Against the background of the prevalent raw material supply risk within the South African e-waste recycling sector, the poor performance was found to be compounded by the strong likelihood of running below the name plate plant capacity of each business model.

Following the results of the evaluation of the business models it was seen that the performance of each the models could be improved by reducing the operating costs and increasing the gold content in the WPCBs. Comparisons of the operating costs of group 3 and 4 business models showed that the reduction in the purchase price of crushed WPCBs would result in superior performance of the group 4 models. The elimination of the size reduction plant in the group 4 business models also resulted in a reduction in the CAPEX and associated overheads. It was on this basis that the group 3 business models were eliminated from further study.

The group 4 business models were selected for the detailed sensitivity analysis to establish the conditions necessary to shift the performance of the models from the capital risk region into the risk free region associated with the performance curves. The sensitivity analysis of the group 4 business models was covered in depth in chapter 8.

Chapter 8:   SENSITIVITY ANALYSIS

8.1.   Sensitivity analysis

Internal and external forces in the concern's environment affect profitability. The cumulative effect may enhance or depress the overall profitability and life of the business. Graphical plots were made to depict the effect of these factors on the profitability of a project. More influential variables can be combined to examine different scenarios that attempt to simulate a random combination of events in the economic environment.

8.2.   Objectives of the analysis

8.2.1.   Selection of the models for the sensitivity analysis

In the chapter 7 all the business models were shown to be unviable. A consideration of the initial capital invested and plots of CRR, DCFRR and IRP showed that the elimination of the size reduction plant, associated with the group 4 business models did not result in large variations in the performance of the models. The performance of the group 4 business models and the group 3 business models was similar. It was observed that the performance of the group 4 models would surpass the performance of the group 3 business models when the purchase costs of crushed WPCBs was less than the internal transfer price of crushed WPCBs. It was also shown that one of the top four cost drivers to operating costs were associated with raw materials, of which WPCBs were the most significant. The elimination of the size reduction business unit in the entire portfolio of operations also led to a reduction in the operating and recapitalization costs associated with this plant. In light of the attendant advantages, the group 4 business models were selected for further study.

8.2.2.   Analysis of group 4 business models

The analysis of the group 4 models was undertaken with the aim of examining the influence of changes in WPCB gold content, initial capital investment, operating costs, capacity utilization, changing revenues (due to changing sales, metal prices and recoveries) and WPCB purchase prices on the performance of the business models.

The study was also undertaken to examine which factors had the most significant influence on the business models. The findings were necessary to provide further information in order to justify modifying the operations where necessary, thereby improving the economic performance of the proposed undertaking.

Thirdly a study of the effect of the gold content in WPCBs enabled an appreciation of the extent to which the proposed venture operating with any four of the business models was likely to run as a going concern in light of the features characterizing the South African e-waste recycling landscape. The data sheets used in plotting the graphs discussed in section 8.4 are included in appendix H.

8.3.   Limitations in this study

The detailed economic feasibility and profitability assessments revealed that all the business models were unviable. The business models 3a to 4d had negative NPV values. It was also shown in the study that all the business models using WPCBs with an average gold content of 250ppm were incapable of repaying the invested capital, replacing equipment that reached its useful service life before the maturity of the project and repaying the interest on the capital. As a result the calculation of DCFFR and the development of plots of DCFRR against time in the sensitivity analysis were not possible.

Secondly the effects of changes in tax, inflation and WACC could not be studied. Plots of changes in these three parameters were distorted by the negative NPV values giving rise to inverted curves that did not reflect the true performance of the business models under study. The plots of inflation rapidly tended towards a large negative value.

The effects of tax are not studied further since tax cannot be deducted where there is no income. As a result only plots showing the degree to which each variable factor under study affected the NPV over the life of the project were done in this study.

8.4.   Findings of the analysis

8.4.1.   Changes in total capital invested

The plot in Figure 8.1 examined the influence of the changes in the total capital invested for each of the group 4 business models. The slope of the curve associated with the business model with inherently high capital costs and capitalization costs (business model 4a), had steeper gradients compared to business models 4b, 4c and 4d.

Business model 4b which had the lowest capital outlay, showed the lowest sensitivity to changes in the initial capital invested. In this regard potentially doubling the name plate operating capacity of business model 4b would have a lower risk compared to extreme cases like models 4a and 4d. An evaluation of the potential for each of the models to move out of the risk zone revealed that a decrease in the initial capital outlay by a margin of 50% would not improve the performance of the models. All the models remained unviable.
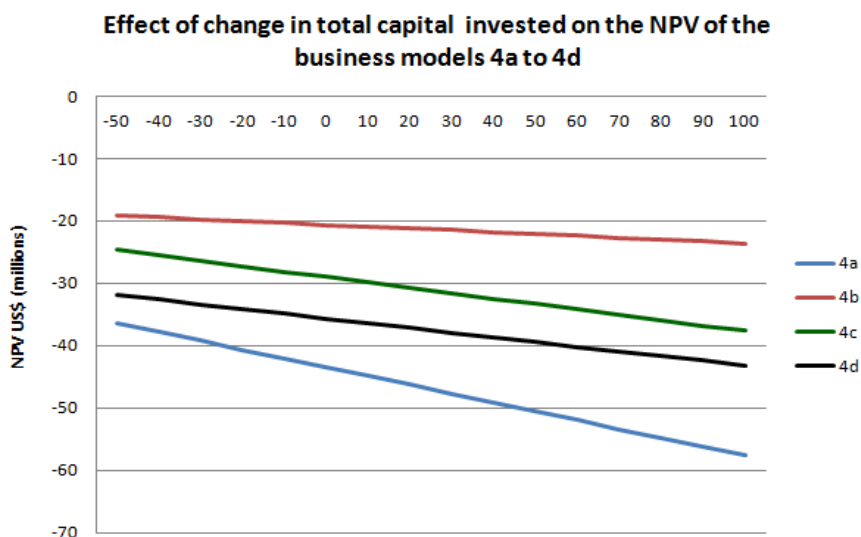
Figure 8.1: Effect of change in total capital invested on the NPV of the business models 4a to 4d

8.4.2.   Changes in operating costs

A plot of the changes in the overall operating costs was done for changes ranging between -50% and +100% as depicted in Figure 8.2. The plots for all the business models showed a rapidly declining performance with increasing operating costs. Only business model 4b displayed potential to move into the capital risk free region when the operating costs were reduced by a margin of 60%. The profit margins were however predicted to be marginal. Reducing operating costs by the elimination of the cost of WPCBs would not result in an improvement in the performance of the remaining business models 4a, 4c and 4d. WPCBs alone command a minimum of 56% of the total raw material costs. With the current demand for a scarce supply of WPCBs the likelihood of securing a constant supply of the material without paying for it is unlikely.
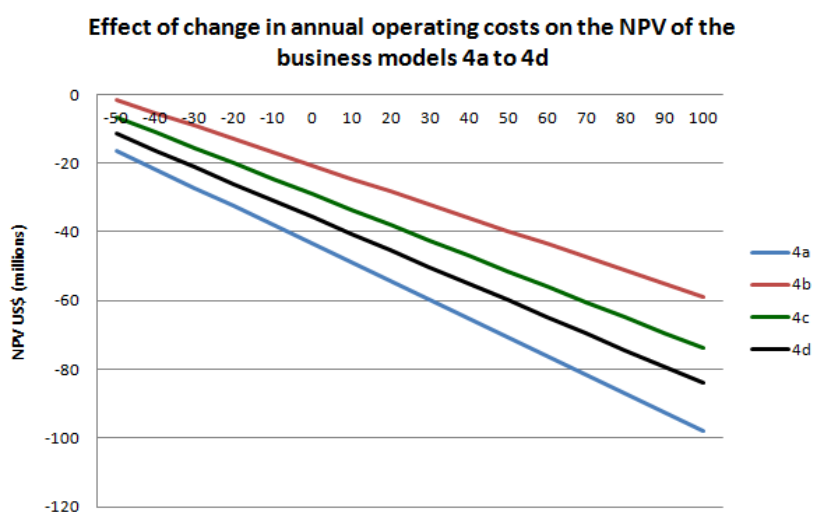
Figure 8.2: Effect of changes in operating costs on the NPV of the business models 4a to 4d

In the previous chapter a study of the operating costs revealed that raw material costs and effluent treatment costs were two of the top four operating cost drivers. The significant raw material cost driver was associated with WPCB purchase prices. A more intimate analysis within the framework of changing operating costs was done on the basis of changing effluent and waste treatment costs and changing WPCB purchase prices.

The effect of changes in WPCB purchase prices was studied for the margin of change ranging between -100% to +100%. The results of the analysis are depicted by the graphical plot in Figure 8.3. Despite a reduction in WPCB purchase prices by a margin of 100% (acquiring the WPCBS free of charge), none of the business models were able to move out of the capital risk region. The slopes of the plots for each business model did however indicate that the performance of the business models was affected significantly by the changes in WPCB purchase prices.
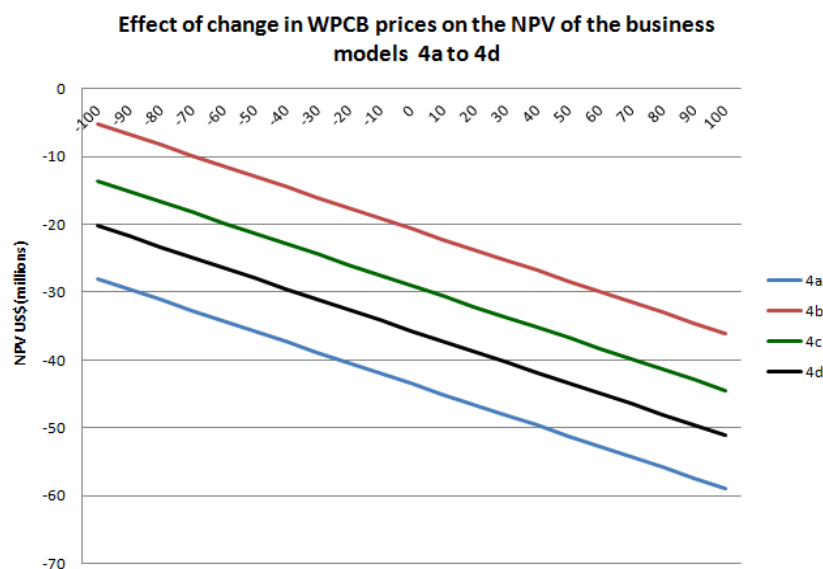
Figure 8.3: Effect of changes in WPCB purchase prices on the NPV of the business models 4a to 4d

Figure 8.4 displays the effect of the changes in effluent and waste treatment costs ranging between -50% and +100%. The slopes of each of the business models revealed a similar degree of sensitivity to changes in effluent and waste treatment costs. Although the effluent and waste treatment costs were reduced by a 50% margin there was no improvement in the performance of the business models. All the business models remained in the capital risk region.

Effluent and waste treatment costs constitute an average of approximately 14% of the annual operating costs for all the business models. The elimination of these costs entirely would not realise a shift of any of the business models into the capital risk free region, since an equivalent reduction in overall operating costs due to the elimination of the effluent and waste treatment by a margin of only 10% would not improve the project performance.

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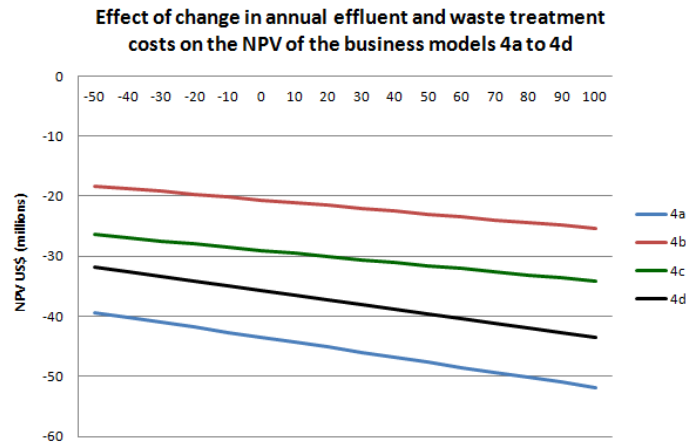
Figure 8.4: Effect of changes in annual effluent treatment costs on the NPV of the business models 4a to 4d

8.4.3. Changes in both CAPEX and OPEX

The simultaneous changes of CAPEX and OPEX were studied and a plot of the effects is depicted in Figure 8.5. The capitally intensive business models showed the greatest rate of decay in project performance with increasing operating costs. A reduction in both the initial capital outlay and annual operating expenses by margins of at least 48% for model 4b, 54% for model 4c and 65% for both models 4a and 4d was necessary to improve the performance of the project. The observed findings qualify the need to modify the proposed process in a manner that reduces the operating steps and hence the initial capital outlay and annual operating expenditures. Alternatively, the entire process can be rejected and another process with CAPEX and OPEX budgets within the neighboured of the margins depicted in Figure 8.5 can be designed.
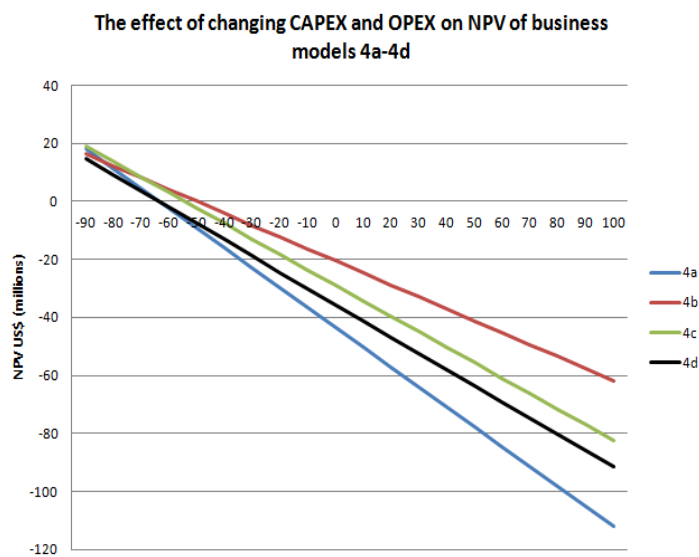
Figure 8.5: The effect of changes in both CAPEX and OPEX on NPV of the business models

8.4.4. Changes in copper revenues

Changes in copper product based revenues were attributed to one or more of the following factors: the movement in the copper metal prices, changes in copper sales volumes or the changes in copper recoveries in the process.
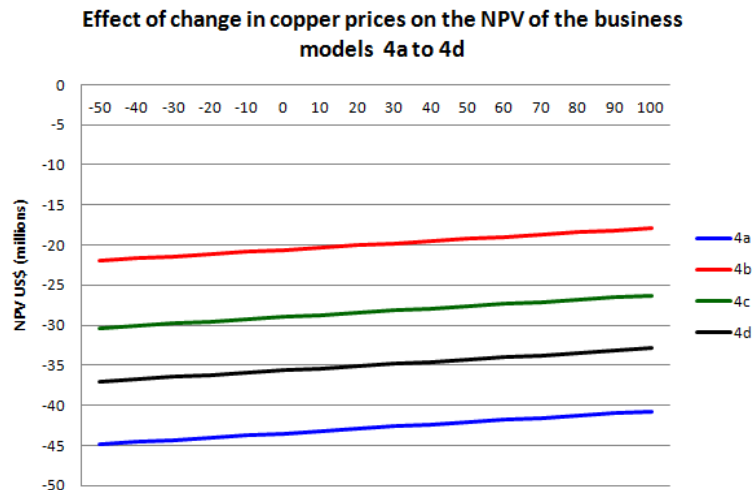
Figure 8.6: Effect of changes in copper prices on the NPV of the business models 4a to 4d
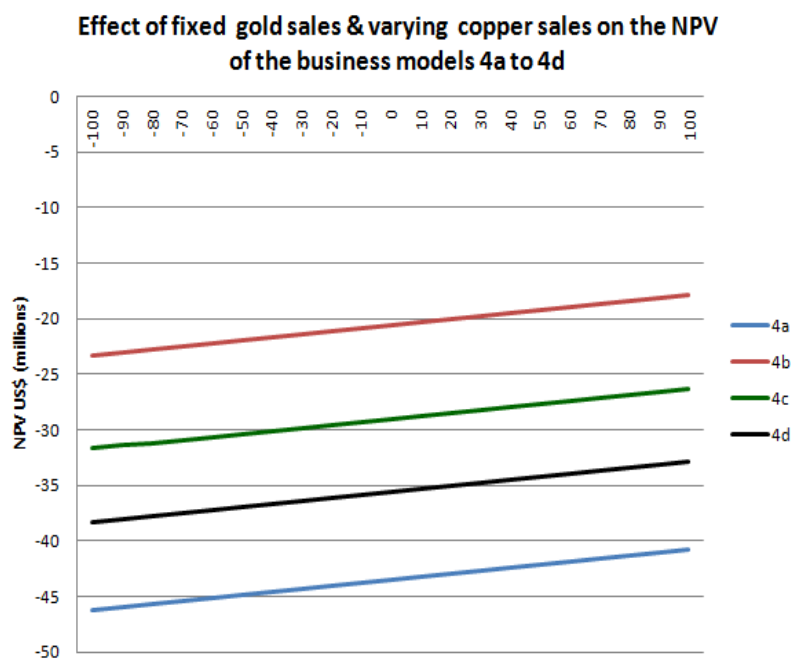
Figure 8.7: Effect of changes in copper product sales on the NPV of the business models 4a to 4d

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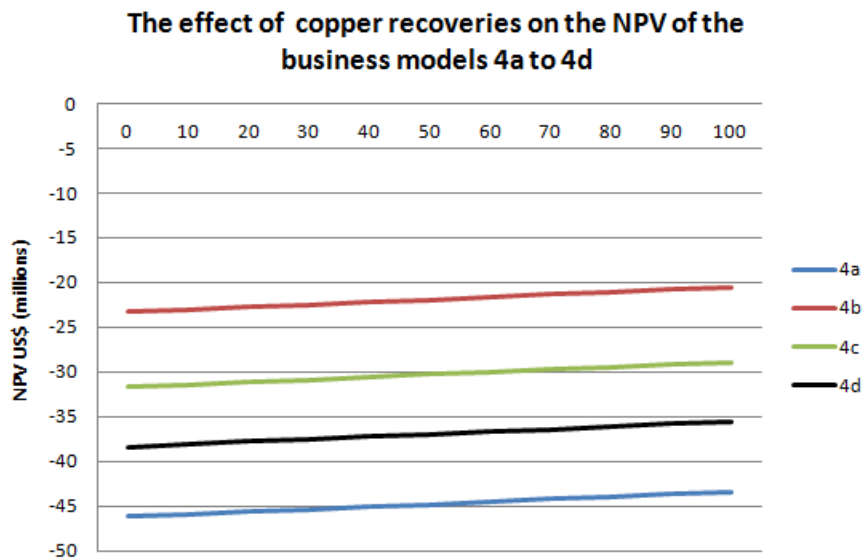
Figure 8.8: Effect of changes in copper recoveries on the NPV of the business models 4a to 4d

The plots for each of the business models, in Figures 8.6 and 8.7 revealed a very low sensitivity toward changes in copper metal prices and copper sales volumes. All the business models displayed a relaxed sensitivity towards changing copper metal prices and sales volumes. Despite doubling the copper metal price all the business models remained unviable.

The observed similarity between the slopes of curves in Figures 8.6 and 8.7 (changing copper prices and changing copper sales) was due to the fact that the copper based products were priced according to the contained metal value based on current market rates.

Increasing the copper sales beyond the base case (at y=0) was only possible through an increased copper content in the WPCB feed material. Despite doubling the copper sales as a result of increased copper content in the WPCBs, all the business models remained unviable.

The effect of copper product recoveries as LME grade A copper or copper sulphate solution (3.71%), was also analysed. The graphical illustration also showed a relaxed trend for the plots of each business model. The recoveries of copper also displayed little influence on the profitability all the business models. This observation almost mirrors the analysis of revenue contribution that was done in chapter 7, section 7.2.

A comparison of the gradients of the slopes of the curves in Figures 8.6 through to 8.8 ($|\Delta NPV/\Delta\%|$), revealed that the copper recoveries in the process had a stronger influence on the returns associated with the copper business, than the changes in copper prices or copper sales.

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8.4.5. Changes in gold revenues

Changes in gold product based revenues were attributed to one or more of the following factors: the movement in the gold metal prices, changes in gold sales volumes or changes in gold recoveries. The effect of the aforementioned factors is depicted in Figures 8.9, 8.10 and 8.11. The movement of gold revenues due to changing gold metal prices, gold sales volumes or gold recoveries showed a stronger influence on all the models in contrast the changes in the factors influencing copper revenues, as revealed by high values of ($|ΔNPV/Δ\%|$).
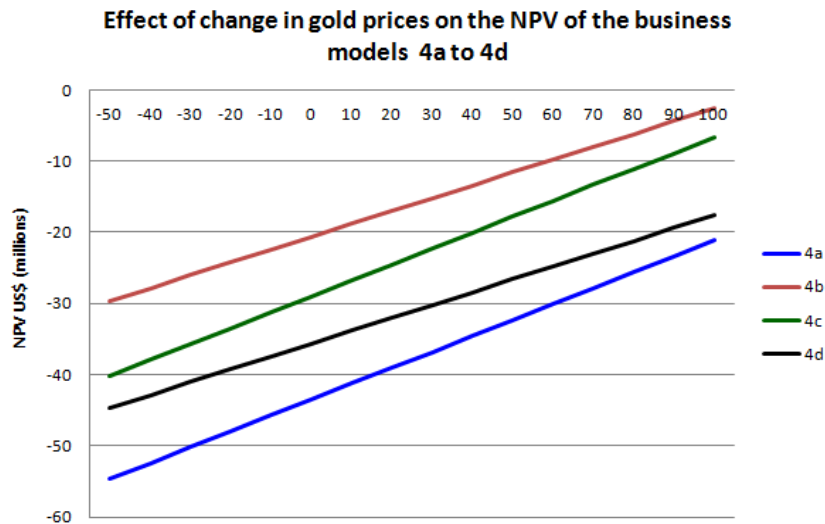
Figure 8.9: **Effect of changes in gold prices on the NPV of the business models 4a to 4d**
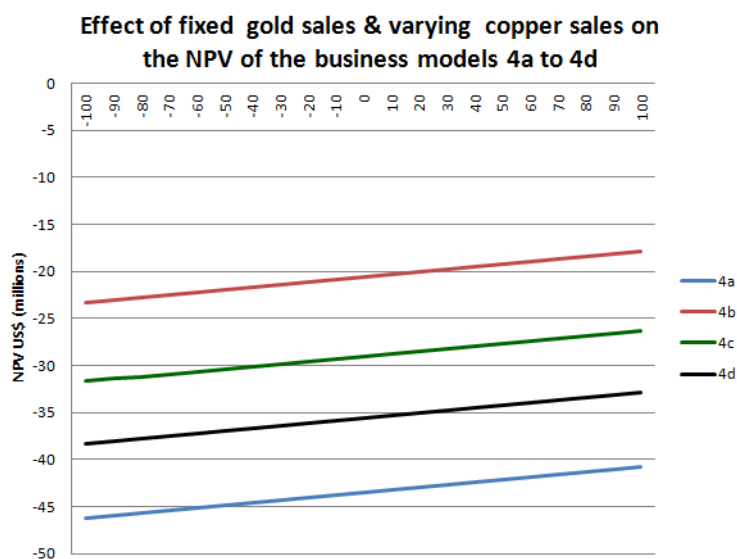
Figure 8.10: Effect of changes in gold product sales on the NPV of the business models 4a to 4d

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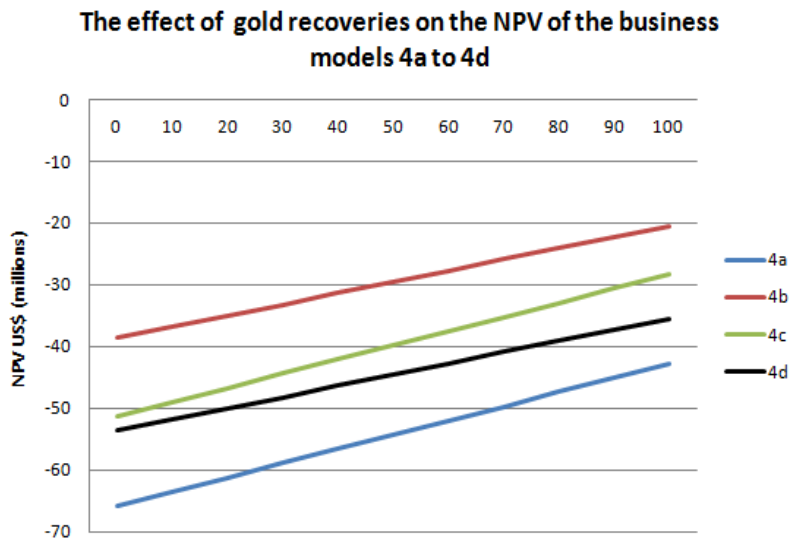
Figure 8.11: Effect of changes in gold recoveries on the NPV of the business models 4a to 4d

Business models 4a and 4c displayed the greatest sensitivity to changing gold prices, gold sales volumes and gold recoveries. An all time high on the gold prices of US$67.65/g (Infomine, 2017) equivalent to a change of 60% depicted by the graph in Figure 8.9, did not register an improvement in the performance of any of the business models.

Business model 4a with the highest initial capital outlay and annual operating costs remained the most unviable business model. The performance of all the business models was in the negative at the base case scenario (0% change). Increasing sales beyond the base case meant that there would have to be an increase in the gold content in the WPCBs feedstock.

Increasing revenue based on gold based product sales volumes beyond the base case implied that the initial WPCB stock would need to have a gold content greater than 250ppm.

A doubling of the revenue from sales translated to an equivalent gold concentration in the WPCBs feedstock of 500ppm. At this concentration however, all the business models remained unviable. A further interrogation of the effect of gold content in the WPCBs was therefore covered in section 8.4.8.

8.4.6.   Changes in gold and copper product revenues

The changes in revenues from both copper and gold based products were attributed one or more of the following: the movement in the metal prices, the changes in the total product sales volumes and changes in the copper and gold recoveries. The changes in revenues from changing metal prices, changes in copper and gold product sales and changing recoveries are shown in Figures 8.12, 8.13 and 8.14. The slopes of each of the business models in Figures 8.12 and 8.13 were congruent.

An analysis however, of the slopes of the curves depicted in Figure 8.14 revealed that the effect of changing metal recoveries was more significant than either changing metal product sales volumes or metal prices. This observation was similar to the analogy of the curves discussed in sections 8.4.5 and 8.4.6. In all cases the impact of recoveries of the metals was more significant than either sales or metal prices. Efficient process design with minimal values reporting to tailings or scavenging should therefore be targeted. The process designed should avoid high metal value lock up and focus on maximising the output of saleable product. Any scavenging and recovery circuits incorporated in the process should aim to maximise the final quality of the recovered material in the tailings, effluent and recycle streams.
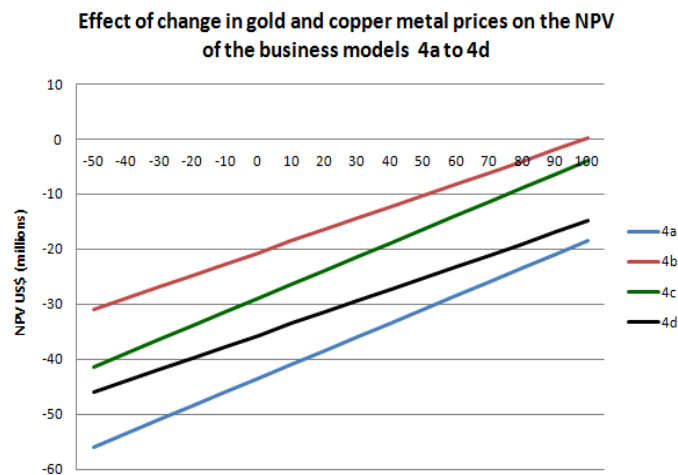
Figure 8.12: Effect of changes in gold and copper metal prices on the NPV of the business models 4a to 4d
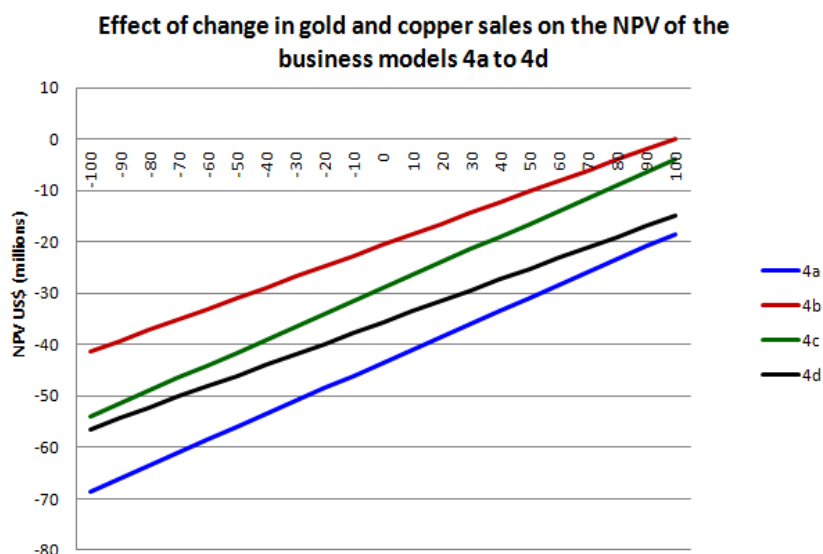
Figure 8.13: Effect of changes in copper and gold product sales on the NPV of the business models 4a to 4d

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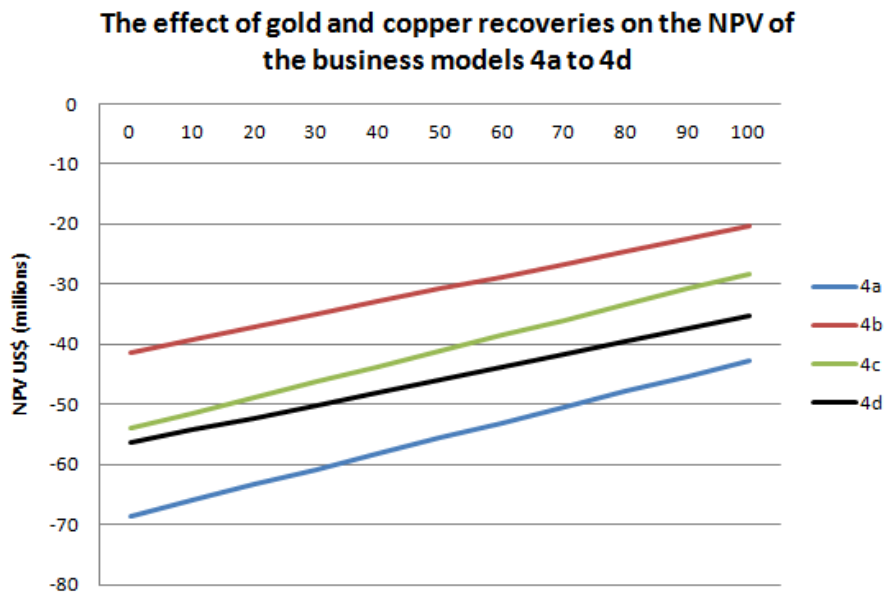
Figure 8.14: The effect of changes in copper and gold recoveries on the NPV of the business models

Comparing the plots of the changes in revenue from gold based product to the plots of changes in the revenue from both copper and gold based products showed that the impact of the revenues from the copper business was not as significant as that of the gold business. The plots in Figures 8.12, 8.13 and 8.14 are shifted up marginally.

The similarity of the plots discussed in section 8.4.6 and Figures 8.12, 8.13 and 8.14 showed that the performance of the business models is strongly dependent on the success of the gold business.

The focal point of process improvement and upgrading should therefore be directed to the significant revenue driver, the gold business. This observation agrees with findings of the analysis of revenue contribution from gold products that was done in section 7.2. The average contribution of gold revenues to the combined metal product gross revenues in all the business models averaged over 88%.

8.4.7. Changes in capacity utilization

The effect of changing capacity utilization on the performance of the business models was analysed and the results are depicted in Figure 8.15.
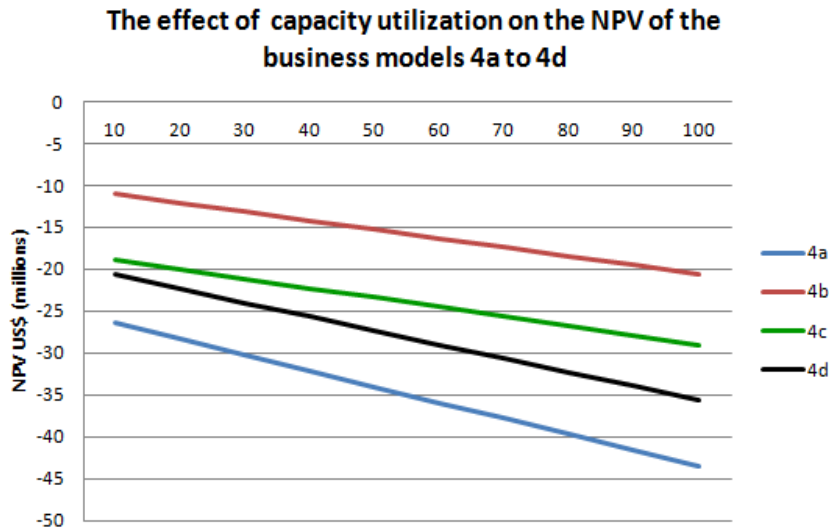
Figure 8.15: Effect of changes in capacity utilization on the NPV of the business models 4a to 4d

The activity based costing method was applied in the project costing done in chapter 7. The effect of this accounting method meant that the operating costs showed a direct proportional change to changes in capacity utilization. As the capacity utilization increased the overall project profitability decreased. This decrease was due to the overshadowing effects of the steep operating costs associated with each business model. Revenues realised could not outweigh the effects of annual operating costs as discussed in section 7.2.

8.4.8. Changes in gold content in WPCBs

An analysis of the influence of changing gold content in the WPCBs was undertaken. The base case in this study was a gold composition of 250ppm in the WPCBs, which is associated with medium grade WPCBs. The upper concentration studied was 1000ppm which corresponds to the highest WPCB grade. The purchase prices of the WPCBs were adjusted in this sensitivity study based on the gold content as reported in literature (Hagelüken, 2006; Lydall *et al.* 2017).

The results of the analysis are summarized in Table 8.1 which shows the effect on the NPV of each business model as the gold content increased. These results were illustrated graphically in Figure 8.16.

Table 8.1: NPV of the business models as gold content changes (US$)

Sensitivity of models to changes in gold content				
	NPV (US$)			
Gold content (ppm)	**4a**	**4b**	**4c**	**4d**
250	-43 472 752	-20 614 882	-28 990 635	-35 644 029
500	-22 665 062	-3 625 881	-8 182 946	-18 606 483
750	-7 373 040	7 847 452	7 109 077	-7 084 603
1000	7 880 944	19 282 747	22 363 061	4 399 237

The results showed that all the business models displayed a strong sensitivity to the gold content in the WPCBs feedstock. The lowest minimum gold content required for a business model to become profitable was associated with the business model 4b. This was attributed to the lower initial capital investment and annual operating costs associated with this business model. Business model 4b became profitable when a minimum gold content in the WPCB feedstock of 563ppm was used.

Business model 4c with a lower initial capital outlay and annual operating costs than model 4a performed better, becoming profitable at a WPCB feedstock gold concentration of 625ppm. Business model 4c performed better than 4d since it has greater annual gross revenues and the potential for a greater return on investment and higher profit margins as demonstrated in section 8.5.2, Table 8.20.

The performance of business model 4a matched that of business model 4d at a gold content of 775ppm and became viable when a minimum gold content of 875ppm in the WPCB feedstock was specified for the operations of model 4a.

Business model 4d displayed the poorest performance. This was attributed to the combined effects of running with high operating costs, typical of operations associated with the production of metal products.

A comparison of the capital and the fixed asset turnover ratios (Table 7.18) of business models 4a and 4d revealed that business model 4a would rapidly outperform model 4d. This was because business model 4a was able to generate revenue at a faster rate than model 4d. Business model 4d only became viable when the gold content in the WPCB feedstock was 925ppm.
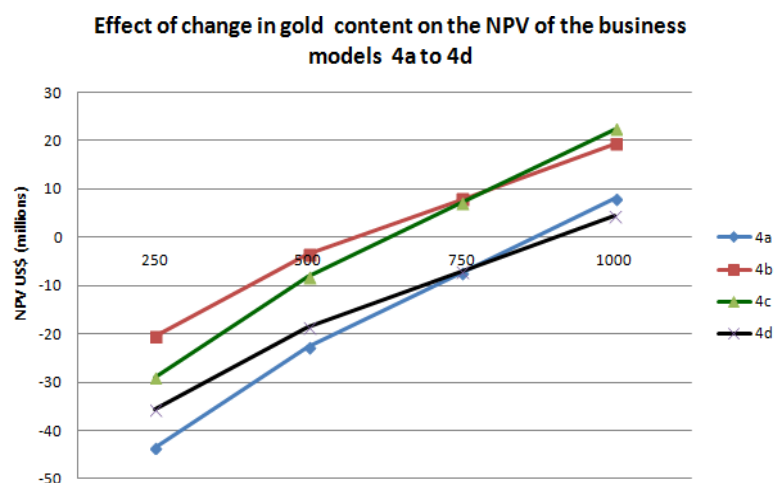
Figure 8.16: Effect of changes in the gold content in the WPCBs on the NPV of the business models 4a to 4d

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The effects of the gold concentration in the WPCB feedstock on the DCFRR and DPBP were also considered. DCFRR values increased for each of the business models and the results are summarized by the data in Table 8.2.

Table 8.2: DCFRR of the business models as gold content changes (%)

Gold content (ppm)	DCFRR (%)			
	4a	4b	4c	4d
250				
500				
750	4.76%	43.09%	22.57%	-2.15%
1000	19.94%	85.93%	42.56%	19.78%

The results in Table 8.2 were consolidated into a graphical plot depicted by Figure 8.17.
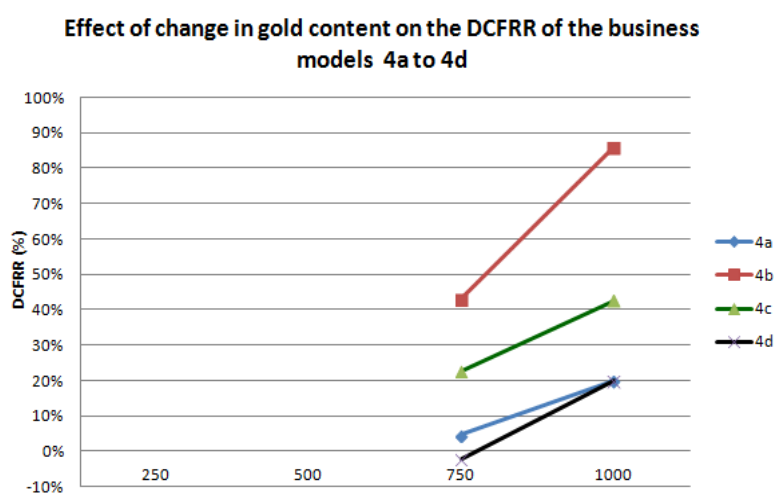
Figure 8.17: Effect of changes in the gold content in the WPCBs on the DCFRR of the business models 4a to 4d

The increasing gold content in the WPCBs feedstock witnessed declining discounted payback periods for each of the business models depicted in Table 8.3.

Table 8.3: DPBP of the business models as gold content changes

Gold content (ppm)	DPBP (Years)			
	4a	4b	4c	4d
250				
500				
750		2.92	6.74	
1000	8.09	1.35	2.96	8.20

8.5. Implications of the sensitivity analysis

The output of the sensitivity analysis is summarized in Table 8.4 which showed a comparison of the rate of change of NPV ($|\Delta NPV/\Delta\%|$) over the project life for every percentage change of the variable factor under study. The results apply to WPCBs with a gold concentration of 250ppm.

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A mean rate of change of NPV for all the four models was calculated. The mean values of each variable factor were compared against each other.

The mean with the greatest magnitude of change was associated with the variable factor having the most significant influence on the performance of the business models. The mean values were ranked according to the level of significance of the impact.

Table 8.4: Evaluation of the influence of the different factors

Sensitivity of business models to changes in the different parameters Rate of change of NPV as factors change (US$/%) ($\lvert\Delta NPV/\Delta\%\rvert$)						
Business model	**4a**	**4b**	**4c**	**4d**	**I Mean I**	**Rank**
Operating costs	-543 931	-381 566	-447 381	-483 356	-464 059	1
CAPEX and OPEX	-541 056	-324 768	-421 548	-441 098	432 118	2
Gold and copper recoveries	254 098	209 624	254 123	209 599	231 861	3
Copper and gold prices or Copper and gold sales	250 611	207 379	250 286	207 703	228 995	4
Gold recoveries	229 653	182 161	229 653	182 161	205 907	5
Gold prices or Gold sales	223 246	180 339	223 246	180 339	201 793	6
WPCB purchase prices	-154 439	-154 439	-154 439	-154 439	-154 439	7
Capacity utilization	-188 946	-106 625	-112 274	-165 943	-143 447	8
Total capital investment	-141 407	-29 807	-86 580	-75 368	-83 291	9
Effluent treatment costs	-83 424	-46 974	-51 538	-78 655	-65 148	10
Copper recoveries	28 413	28 076	28 076	28 413	28 244	11
Copper prices or Copper sales	27 365	27 040	27 040	27 365	27 202	12
Gold content in WPCBs	128 384	99 744	128 384	100 108	104 958	Critical

Figure 8.18 provides a graphical representation that compares the magnitudes of the average mean changes of the NPV against percentage change in the variable factor under study ($\lvert\Delta NPV/\Delta\%\rvert$). The data was extracted from Table 8.4.

Based on the analysis of the means in Table 8.4 and the graphical plot in Figure 8.18, it was established that the performance and overall profitability of the business models was most sensitive to changes in the overall operating costs. A combination therefore, of reduced effluent and WPCB purchase prices would have a more positive impact on the performance of each business model than they would when reduced individually.
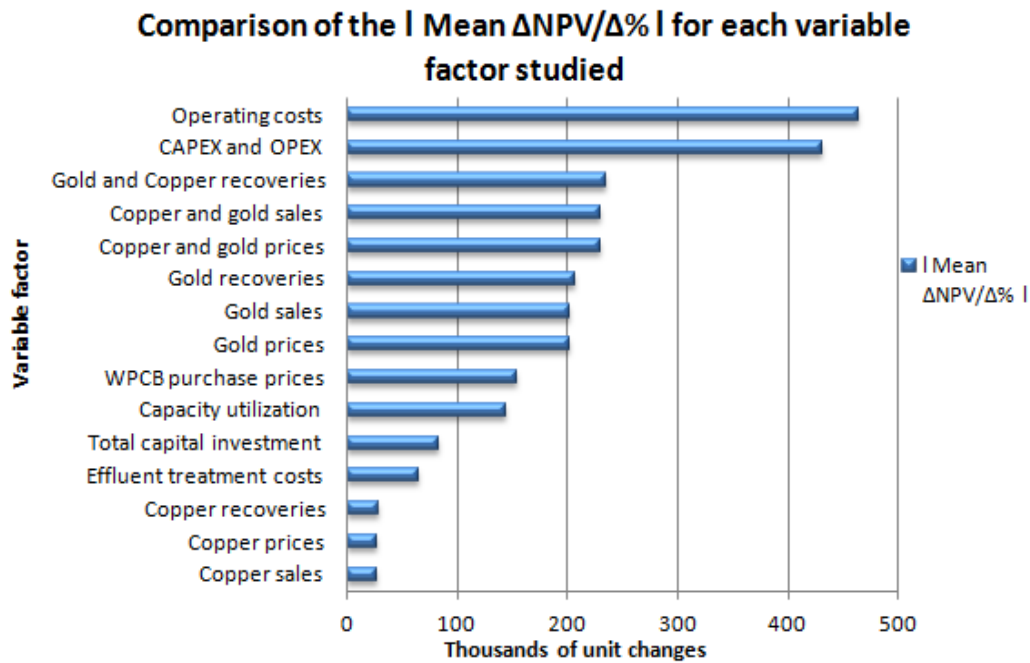
Figure 8.18: Comparison of the level of significance of each variable factor studied

The combined effect of reduced CAPEX and OPEX had the second most significant influence on the overall profitability of the business models. Based on the plot of the changing CAPEX and OPEX (see Figure 8.5) it was seen that a reduction of these costs by margins of at least 48% for model 4b, 54% for model 4c and 65% for models 4a and 4d allowed all the business models to move into the capital risk free region.

The movement of metal prices also displayed a significant influence on the performance of the business models. Running operations using any of the business models 4a, 4c or 4d using medium grade WPCBs with an average gold content of 250ppm was unviable. Despite doubling the gold content which meant using high grade WPCBs, there was no change in the viability of these business models. Only business model 4b, which was the least capitally intensive became profitable when metal prices realised a 100% increase.

The effect of combined product sales without moving metal prices did not realise an overall improvement in the profitability of the business models. Increasing sales further at static metal prices and maximum capacity utilization could only be achieved by increasing the gold and copper content in the WPCB feedstock. Even without increasing the operating costs associated with this increase in target metal content, it was seen that the business models still remained unviable.

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Table 8.5: Sensitivity ranking of each business model

Business model	4a	4b	4c	4d	Most sensitive model
Total capital investment	1	4	2	3	4a
Operating costs	1	4	3	2	4a
CAPEX and OPEX	1	4	3	2	4a
Effluent treatment costs	1	4	3	2	4a
Copper prices or Copper sales	2	1	1	2	4b, 4c
Gold prices or Gold sales	1	2	1	2	4a,4c
Copper and gold prices or Copper and gold sales	2	3	1	4	4c
WPCB purchase prices	1	1	1	1	4a - 4d
Capacity utilization	1	4	3	2	4a
Copper recoveries	2	1	1	2	4b, 4c
Gold recoveries	1	2	1	2	4a, 4c
Gold and copper recoveries	2	3	1	4	4c
Gold content in WPCBs	1	2	1	2	4a, 4c
Overall sensitivity	**1.38**	**2.56**	**1.56**	**2.38**	**4a**

The sensitivity ranking of a business model in comparison to the others for each variable factor was also done. Based on the results in Table 8.5 overall sensitivity was calculated for each business model. This was done by calculating the arithmetic mean of the sensitivity rank scores for each business model to each factor. The results are summarized in Table 8.5. The results in this Table showed a strong congruency with output from the project costing and profitability studies covered in chapter 7.

It can be seen that the capitally intensive business models, associated with recovery of metals in the solid state, were most sensitive to the effect of changing operating costs and initial capital outlay. These models also displayed great sensitivity towards changes in metal prices and product sales.

The general conclusion drawn from the sensitivity analysis is that the project profitability was compromised by steep operating costs which could not be matched by the revenues derived from gold and copper sales. The performance of the business models is driven by gold revenues. All the business models displayed a zero tolerance towards low recoveries. Operating costs must be managed by running operations with high single stage recoveries.

High in-process recycling frequencies push up the operating costs. If the costs of recycling processes outweigh revenues associated with the sale of the recovered metals process redesign or rejection are the two options left to consider.

While operating at full capacity it was seen that it is necessary to increase revenue from gold sales by upgrading the choice of WPCB grade used for the proposed process. The proposed process can only run profitably with a guaranteed supply of high grade WPCBs with a minimum gold content ranging between 563ppm and 925ppm as depicted by Figure 8.15.

Successful operation using medium grade WPCBs can only be achieved by drastic process modification that is characterised by a simultaneous reduction in the initial capital outlay and annual operating costs. As the grade of WPCBs deteriorates, marked by decreasing gold content, the proposed process is more of a high risk venture for investors to consider as an investment option. Further there is no means of buffering the effects of other risk factors that will be discussed in section 8.6.

8.6. Risk analysis

8.6.1. Recapitalization risk assessment

DCFRR and NPVs for projects can never be predicted absolutely because cash flow data for such projects are subject to uncertainty. A measure of confidence is required for these projects. The uncertainty of estimates increases the farther into the future that estimate is projected.

The venture premium method used to assess the riskiness of a project is based on a subjective estimate of the fraction of capital investment likely to be lost if the project is unexpectedly terminated (Perry *et al.,* 1997). The risk is used to determine insurance Figures that creep into the project's profitability. As risk values increase due to increasing likelihood of fire, theft, accidents due to technical factors of the project, delayed start up, law suits, the sensitivity of the project increases due to these risk factors. These factors are threats to the life of the project which often times require determination of capitalization costs. The comparison of ratios of capitalization cost and NPV gives an indicator of the potential risk of a project terminating as a result of failure to finance equipment replacement. The capitalization ratio is useful for comparing technologies or comparing changes in capital investment within the context of a sensitivity analysis.

Probabilistic methods, some depending on historical performance are available for estimating risk. At the preliminary stage of the project evaluations, detailed determinations using numerical methods like the Delphic method were not used in the analysis since all the business models analysed were found to be unviable (Perry *et al.,* 1997).

In this study the magnitude of the ratios of capitalized cost and NPV were compared to determine the level of risk associated with each business model. Plots of the ratios over the life of the project were done.

Figure 8.19 is a plot of the recapitalization risk assessment for each of the business models 3a to 4d.
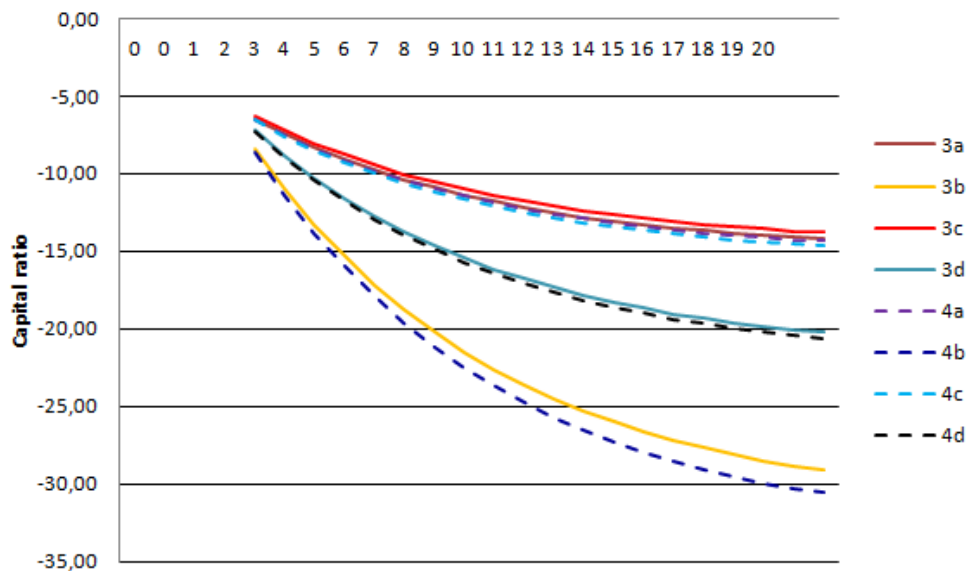
Figure 8.19: Recapitalization risk assessment for each of the business models 3a to 4d

Based on the plots for each business model in Figure 8.19, the probability of risk associated with closure and failure of the business models is 100%. Each of the plots shows a perpetual decay in the recapitalization ratios over the life of the project. The implication of the observed trend points towards the failure of all the models to finance the recapitalization of process equipment at any stage during and over the entire life of the project. The certainty of prematurely terminating all operating activities before the life of the project is 100%.

All the business models operating with WPCBs feedstock with an average gold content of 250 ppm were found to be incapable of sustaining continued operations even beyond a minimum period of one year. The risk position can only be improved by ensuring a consistent supply of high grade WPCBs with a minimum values of 875ppm, 563ppm, 625ppm and 925ppm for business models 4a, 4b 4c and 4d respectively.

Secondly, reducing the operating costs or reviewing the entire scope of operations in favour of less fiscally intensive processes. In conditions of strained economic operating conditions the tactical decision at an operational level and strategic intervention at a corporate level would be the disinvestment in the copper business.

Focus must be shifted to outsourcing the fiscally intensive operations of the copper business to well established copper processors or intermediate chemical product processors with larger economies of scale. Thirdly it is prudent to recover other precious metals as more concentrated solutions or salts for sale if the net returns are positive. Investing in purifying the other precious metals streams is not advisable.

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8.6.2. Evaluating susceptibility to risk factors

Based on the analysis of the sensitivity of each business model to the factors discussed in section 8.4, four additional categories of major risk areas were identified that were found to affect the operations and overall performance of the proposed process. The clustering of the variable factors studied under the most appropriate risk area was done and is illustrated in Figure 8.20.
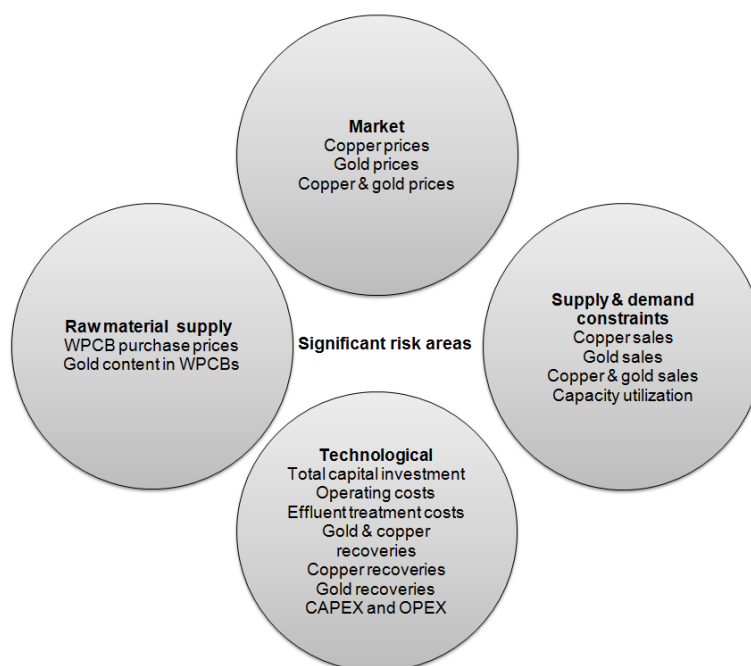
Figure 8.20: Clustering of variable factors in the sensitivity study under the areas of risk

The relative assessment of the susceptibility of each business model to each defined areas of risk is detailed in Table 8.6. Each of the variable factors in Table 8.5 was allocated to the type of risk area shown in Figure 8.20. A mean of the ranking scores for each business model in the risk areas was computed. The business model with the lowest score was found to display the greatest susceptibility to that area of risk in comparison to the other business models.

Table 8.6: Evaluation of models' susceptibility to risk factors

Risk susceptibility					
	4a	**4b**	**4c**	**4d**	**Most susceptible model**
Market	1.67	2.00	1.00	2.67	4c
Supply and demand constraints	1.50	2.50	1.50	2.50	4a, 4c
Raw material supply	1.00	1.50	1.00	1.50	4a, 4c
Technological	1.29	3.14	2.00	2.43	4a
Overall analysis (ẋ)	**1.37**	**2.29**	**1.38**	**2.28**	**4a**

Based on the results in Table 8.5 overall sensitivity was calculated for each business model. This was done by calculating the arithmetic mean of the sensitivity rank scores for each business model to each factor. The results are summarized in Table 8.6.

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The sensitivity analysis revealed that the most capitally intensive business model 4a had the highest overall degree of susceptibility to risk as indicated by the lowest score of 1.37 (see Table 8.6). The least capitally intensive business model, 4b displayed the lowest overall susceptibility to risk scoring an average of 2.29 for the overall sensitivity. The results mirrored the findings in Table 8.5. The most capitally intensive business model displayed the greatest sensitivity to intrinsic and extrinsic influences in the business environment.

The least capitally and operationally intensive models displayed the lowest sensitivity to the different factors studied. The likelihood of failure of business model 4a was also exposed by the position of the plots of model 4a relative to other business models in each of the graphs in section 8.4. The combination of high operating costs and high recapitalization rates makes it most susceptible. The lowest overall susceptibility to risk displayed by 4a is congruent with the position of all the plots of business model 4a in relation to the other business models for all the variables studied in section 8.4.

To circumvent the observed areas of risk it is advisable to ensure a broad and secure customer base with minimum specialised product requirements Reliable customer supplier relationships must also be secured and maintained especially for specialised new entry products like the aurocyanide liquor with limited market data. The aurocyanide liquor business is most susceptible to this market risk. While business model 4d scores the highest in this area, it is not immune to the market risk associated with gold liquor sales volumes. Reliable supplier contracts and symbiotic partnerships across the value and supply chains are crucial to nullify the effect of the prevalent raw material supply risk and supply and demand related constraints. From the conceptual stage of project the design of a process with high single stage recoveries and few complex steps must be targeted as noted earlier.

8.7.   Chapter summary

The sensitivity analysis was done on the group 4 business models. It had been identified in chapter 7, that the economic performance curves of the group 3 and group 4 business models were congruent. Significant departures in the overall economic performance were expected when the crushed WPCB purchase costs were less than the internal transfer price used in the group 3 business models. The elimination of the size reduction business unit also meant a reduction in the overall operating costs when each group 4 business model was compared to its respective group 3 business model. It was against these arguments that the group 3 business models were eliminated leaving the group 4 business models for further analysis.

A detailed sensitivity analysis of the group 4 business models was undertaken. The effects of the movement of annual corporate taxation rates were not studied since tax cannot be levied against a perpetual loss position. The influence of changes in the annual inflation rate and changes in WACC were also not studied in depth since the associated plots were distorted by the negative discounted cash flows throughout the life of the project for all the business models studied.

An examination of the different plots showed a recurrent trend for all the business models, which reinforced the findings of the project evaluations made in the previous chapter. It was also revealed that at an average gold content of 250ppm in the WPCBs, all the business models remained in the capital risk zone. In spite of improving recoveries and running at full rates, coupled with high product off takes, the business models remained unviable.

The variation of the gold content was studied since the chief contributor to revenue in WPCBs is gold. As the gold content increased a rapid performance increase was observed for the business models. Model 4b showed the greatest sensitivity and moved out of the capital risk zone at a minimum gold content of 563ppm. The more capitally intensive models associated with recovery of copper and/or gold in the solid state displayed positive returns at higher gold contents in excess of 620ppm. This high gold content is associated with high grade WPCBs which at present do not have a consistent supply as indicated in chapter 3.

An assessment of all the factors affecting the operations of the proposed project was rated and it was observed that the business models displayed the greatest sensitivity to changes in the operating costs and changing revenues driven by the gold business. Coupled with the findings of the project evaluations it was established that the pursuit of the proposed hydrometallurgical process for the recovery of copper and gold is an unviable undertaking. The performance of all the business models is affected by the restrictive operating costs. A review of the process is requisite. It is advisable to consider less capitally and operationally intensive processes. The findings of the sensitivity analysis enabled an evaluation of the susceptibility to areas of risk that affected the project's performance to be undertaken.

Combining the output of the profitability studies in chapter 7 it has been found that the success of the proposed venture using medium grade WPCBs requires business and process re-engineering. From the conceptual stage of project the design of a process with high single stage recoveries and few complex steps must be targeted. This intervention must be complemented by reliable producer/client contracts. Symbiotic partnerships across the value and supply chains are crucial to nullify the effect of the prevalent raw material supply risk emanating from low e-waste out flows in the country.

Chapter 9: CONCLUSIONS AND RECOMMENDATIONS

9.1. Overview

The overarching goal of this study sought to determine the economic feasibility of establishing a hydrometallurgical facility to recycle waste printed circuit boards. The broad objective of the project was to achieve the recovery of gold and copper. The undertaking was confined to the South African business sphere. In the course of the study 16 potential business models were developed. An initial screening and selection of the business models was undertaken following which the first eight models hosting the collection and dismantling function were eliminated. Costing and economic performance evaluation of the remaining eight business models, 3a to 4d, was undertaken. It was found that none of the business models were viable. The findings of this analysis enabled further screening of the business models. A sensitivity analysis was performed on the remaining business models to examine the existence of specific intrinsic and exogenous operational conditions that would potentially improve the performance of the project.

9.2. Addressing the aims of the study

9.2.1. Objective 1

Develop an understanding of the current position of e-waste management and legislation affecting e-waste recycling in South Africa.

Aspects of the South African business environment were examined in detail using six facets. The PESTEL analysis approach was used to describe the status of e-waste recycling and management in South Africa.

The findings of the study revealed that the recycling of e-waste in the country is progressively evolving into a more structured system. Scope however exists for improving the execution of legislative frameworks in a more homogeneous fashion. A clear policy and budget were identified as being relevant to implementing municipal focused intervention in managing the diversion of waste to landfill. While there has been growing activity in the last three decades the scene remains largely dominated by inconsistent legislative frameworks. The efforts to crystallize the legal framework have had retrogressive impacts on several formal e-waste recycling enterprises.

A large divide between formal and informal activities, with an unclear accounting framework to monitor activities and output from e-waste recycling is still one the attendant features of e-waste recycling activities in the country. There still lacks a means of regulating the flow of and handling of materials along the supply chain. No pricing strategy exists for appropriately valuing e-waste. Coupled with the low margins characterised by the e-waste recycling business as one moves further up the value chain, the level of participation is drastically reduced.

There is however an active concerted effort to redress the low recycling rate of e-waste in the country. This has been reflected by the drive towards the promotion of small and medium scale initiatives, attempts to homogenise the legislation influencing e-waste recycling and enhancing the involvement of the participation of stakeholders and electronic goods producers. This has been evidenced by efforts to move towards a more circular recycling economy through the adoption of EPR.

The findings of the study revealed that, while there exists vibrant research activity in the area of e-waste recycling and management, the technical capacity to handle large outflows of toxic and metallurgically diverse by products and waste is still inadequate. Failure to adequately absorb the outflows contributes to the continued land filling and quarantining of the waste as the most effective solution to handle the waste.

Findings of the study also revealed that low barriers to entry however exist downstream of the e-waste recycling chain that can be exploited with alternative technologies, to add value to the output of e-waste recycling. Room exists for the development of large scale hydrometallurgical activities to exploit WPCBs in the country. The barriers to entry with respect to the level of participation remain low. Prevalent environmental influences were found to exert a negative influence on achieving meaningful activity downstream of the e-waste recycling and value chain. These factors include the legislative structures in place, material flow across the value chain and technical capacity for handling waste and by-product streams.

As yet the current status of e-waste inventory in the country has remained the predominant motivating factor for conservative activity in the area marked by low levels of expansion. The bulk of the key raw material is reported to be within private and public sectors in storage. The outflows of e-waste and as a result, WPCBs, present a significant threat to the viability and life of the proposed hydrometallurgical facility. The scarcity of the key raw material presents supply and demand related constraints that limit the full utilization of installed facilities, delayed capital repayment, thereby countermanding efforts to ensure a new and relevant source of employment.

The present e-waste recycling climate in South Africa is in the infantile stages when compared to the strides made in the developed economies. Findings of the study revealed that the country is yet to strengthen the technical and fiscal muscle to launch a large scale hydrometallurgical processing facility described in the study.

As outflows of e-waste and hopefully consistent supply of high grade WPCBs for the tenure of the project improve in the country, potential exists to expand the operating capacity and viably produce copper and gold. The viability of the concern will potentially be faced with a new threat, namely the reduction in the precious metals content in printed circuit board manufacture. Concerned stakeholders in the e-waste recycling sector in South Africa must be operationally and technically prepared to embrace these developments.

9.2.2.   Objective 2

Define and compare different business models for WPCB metals recycling.

Sixteen business models were developed based on the two major criteria namely: the potential product portfolio of the business models and the scope of focus of the internal plant operations of the entire hydrometallurgical process flow. A due diligence was conducted to ascertain the extent to which the different facets of the South African business environment could support the undertaking at the initial proposed capacity of 30 000 tonnes per annum. It was established that the existent deficiencies and inconsistencies in the legal system, the high raw material supply risk and low capacity to absorb a high production capacity and the associated outflow of toxic by-products was a major stumbling block to such an undertaking. Such an undertaking was noted to be in need of an organised and well funded reverse logistics and warehousing structure. This would also have to be supported by a consistent off take of final and by-product streams to remain viable in the long term.

It was established that the collection and dismantling function was retrogressive towards achieving high profit margins within each of the group 1 and 2 business models. The costs and infrastructural demands associated with the reverse logistics on both a large and small scale were noted to be restrictive. Through entering into collaborative partnerships and restructuring of business models some of the businesses have managed to circumvent the retrogressive effects associated with high reverse logistics costs. Failure to restructure business models has resulted in closure of business operations of some of the recycling firms like Africa Sky. In the current business environment the competition for WPCBs is steep and significant barriers to entry were foreseen due to potentially protective reactions from existent players in this niche of WEEE recycling.

A consideration of the operating capacity of a well established recycling facility revealed that a safe entry capacity for the proposed process should be pegged at a capacity of less than 1000 tonnes per annum. The capacity used in this study was pegged at a recycle rate of 400 tonnes per annum of WPCBs. At this capacity it was envisaged that the existent raw material supply risk will be managed to the extent that it is less of a threat to the continued operations of the proposed hydrometallurgical facility.

It is expected that the burden of managing the costs associated with the reverse logistics and handling toxic by products of dismantling activities will be shouldered by proprietors specialising in this role as revealed by the SWOT analysis.

An analysis of the e-waste recycling supply and value chains in South Africa has shown that sufficient capacity exists to supply crushed WPCBs to the proposed process. This allows for the elimination of another plant, the size reduction plant and the plant's associated plant and labour overhead costs.

It was on this basis that the most favourable business models selected were those that did not incorporate the collecting and dismantling function and the size reduction of WPCBs as part of the internal operating portfolio. An analysis of the scope of products produced and the available market to absorb the products reveals that detailed market data is required to support the gold liquor business. Knowledge of the minimum acceptable quality specifications of the liquor is required prior to committing all resources into full scale production of the liquor. With scarce market intelligence on sodium aurocyanide liquor the product off take rates are difficult to predict. Established long term buyer and supplier contracts are required to protect the integrity surrounding the gold liquor business.

The production of copper sulphate liquor, LME grade A copper cathodes and gold sludge is a viable undertaking supported by an established metals processing industry and market in South Africa. The risks associated with these products are driven chiefly by global metal prices which will affect the returns to the business. It is therefore necessary to ensure the consistent supply of high grade WPCBs to improve the viability of the operations. This analysis was covered in depth in the evaluation of the project economics. At this stage it is advisable to reject the proposed process in view of the prevalent raw material supply risk and inconsistent supply of high grade WPCBs. The uncertain supplies of key raw materials in the absence of concrete statistics on e-waste volumes complicate the operations planning and budgeting in the short and long term for the proposed venture.

9.2.3.   Objective 3

Evaluate and assess the economic performance of each business model based on the proposed hydrometallurgical process

The detailed costing and economic evaluations of the project under the scope of the screened business models 3a to 4d was undertaken. The findings revealed that for the specified operating conditions the project is unviable. Performance curves based on the discounted and undiscounted cash flows were used in tandem with profitability ratios to assess the long and short term performance of the selected business models. The results of the metrics were congruent.

The concentrations of silver, tin and the PGMs were in quantities too low to shift the returns of the business models into a profitable position. Additional hydrometallurgical processing to recover these metals would prove to be uneconomical.

A detailed sensitivity analysis interrogating the most significant factors affecting the performance of the business models was also done. Using the rate of change of NPV against percentage change in the variable factor under study, the most significant influences on the performance of the business models were identified. The results of analysis revealed that all the business models displayed a marked sensitivity to the combined effect of the high operating costs and gold revenues. An analysis of the sensitivity to changing gold revenues revealed that changes in revenues due to gold recoveries had the most significant impact.

The level of sensitivity of each business model relative to the other models in the group was also studied. It was established that business model 4a, with the highest initial capital outlay, gross turnover and operating costs was the most sensitive to changes in the intrinsic and external influences around the business. It was seen that for the performance of the project to improve, intervention at a technical engineering level is necessary. This would be characterised by an improvement in the process by reducing the number of processing stages or choosing an alternative process route. It was seen however that the compositional diversity of the WPCBs complicates the process as the degree of product purity targeted is increased. The use of medium grade WPCBs with an average gold content of 250ppm would require the modification of the current process or the development of an alternative process route. This was supported by findings of the sensitivity analysis done, that examined the combined effect of changing CAPEX and OPEX. A reduction of CAPEX and annual operating costs by margins of 65% for models 4a and 4d, 48% for business model 4b and 54% for business model 4c, would ensure project viability using medium grade WPCBs with a minimum average gold content of 250ppm. It is therefore evident that the proposed process is unsuitable for treating low grade WPCBs. In conditions of strained economic operating conditions the tactical decision at an operational and corporate strategic level would be disinvestment in the copper business, and divert attention to expanding the portfolio around the gold business, capture the market around the gold business and pursue long term buyer/supplier contracts with fiscally and operationally robust business entities, capable of cushioning exogenous risk factors.

The analysis of findings from the sensitivity analysis also showed that all the models have a low tolerance for operating below the name plate capacity. The success of the business models is hinged upon the gold concentration in the WPCBs. It is therefore imperative to ensure optimum and undisrupted supplies of high grade WPCBs feedstock with a minimum gold content ranging between 563ppm and 925ppm for the proposed process to be viable. The viability however is least favourable as the more capitally intensive business models are adopted.

Secondly focus must be shifted to outsourcing the fiscally intensive operations of the copper business to well established copper processors or intermediate chemical product processors with larger economies of scale.

Thirdly it is prudent to recover other precious metals as more concentrated solutions or salts for sale only if the net potential returns exceed the operating costs. Investing in purifying the other precious metals streams is not advisable.

9.3. Recommendations for further study

Improve collection and distribution of e-waste

The creation and operation of a centralized e-waste distribution system should be adopted. The central distribution system will act as the point of contact between the informal and formal recyclers on a broader scale. It should be housed with the appropriate facilities to handle e-waste with sensitive data. Operating frameworks and guidelines can be specified which are governed by existing national legislation on the handling and recycling of e-waste. The existence of the system will improve the planning and operations management of the recyclers. Secondly it will offer a safer and more competitive business environment for informal recyclers and improve the accessibility to e-waste to more recyclers or potential entrants into this niche of the recycling economy. Within the scope of operations of the central distribution system, the collective interests of the informal recycling sector can be addressed. By drawing informal recyclers scope exists for enhancing the accountability and regulation of informal recycling activities.

Improving the accountability and material flow analysis

To circumvent the negative effect of low recovery rates associated with the proliferation of informal recycling activities in developing and transitional economies for example, a detailed analysis of material flows and material balances of existent recycling chains must be undertaken. In the case of South Africa an account of options available for materials recovery must be interrogated. Owing to the unknown volumes of material salvaged and pushed into the e-waste recycling chain by informal recyclers, extrapolation of e-waste volumes recycled can be done using production records of the recycling companies. With the creation of a structured mechanism for accounting for e-waste recycling activities, it is possible to capture this data accurately and integrate it into the national industrial statistics data pool.

Creating and maintaining a central material technical reference data bank

The output of the study was based on a very specific analysis of WPCBs which was adjusted to reflect the typical grade of WPCBs handled in the country. The results of additional economic evaluations and profitability studies are therefore subject to variation.

Findings from the work of numerous researchers have shown how the composition of WPCBs varies (see Tables 2.2 and 2.3). For future work a mean analysis for each category of WPCBs should be defined relative to the period of study. The mean values should be updated annually to parallel the changing precious metals composition of WPCBs. In earlier discussions covered in chapters 2 and 3 it was noted that there has been a shifting trend towards the reduction of the precious metals content in printed circuit boards. This finding is supported by comparative assays of WPCBs from older e-waste sources against the more recent e-waste sources (Cuchiella *et al.,* 2016).

Deepening the understanding WPCB chemistry

The findings in the study can also be improved by a more detailed understanding of the chemistry of WPCBs. A detailed investigation of the solvent extraction of copper can be done while varying the mix of metallic species and operating conditions in acidic media using LIX 984N. The study of a multi component medium is especially useful in handling complex leach solutions from complex polymetallic sources like WPCBs.

In this study the adsorption of gold onto carbon was based on published performance data. The findings of the study and future studies in this area can be improved by empirical work to determine the closest fit to adsorption models studied by previous researchers. Empirical investigations will enable a better understanding of gold adsorption from WPCB leach liquor to be developed.

Exploring alternative processing options

Alternative technical studies exploring the leaching of all metals with the objective of electrowinning with more current technologies can be considered. The impure metal product harvested from the cathodes can be directed to the local pyrometallurgical facilities. The iron content however must be reduced prior to the bulk leaching of the residual material.

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APPENDICES

A: Mass balances used for the economic evaluations

The mass balances that are detailed in this appendix were updated from the original design specifications and empirical work presented by Rossouw (2015) and de Waal (2018).

Table A.1: Mass balances for the shredding and milling plant at a design basis of 400Tonnes processing capacity of WPCBs

(adapted from Rossouw, 2015; de Waal, 2018)

Pre-treatment mass balance		Stream No.								
	Units	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Temperature	(K)	298,15	298,15	298,15	298,15		298,15	298,15	298,15	298,15
Pressure	(kPa)	101,32	101,32	101,32	101,32		101,32	101,32	101,32	101,32
Material Flow (Total)	kg/hr	61,05	61,05	61,05	61,05	0,00	0,00	61,05	0,00	61,05
Metal Flow (Total)	kg/hr	24,42	24,42	24,42	24,42	0,00		24,42	0,00	24,42
Ferrous Metal	kg/hr	4,97	4,97	4,97	4,97	0,00		4,97	0,00	4,97
Fe	kg/hr	4,97	4,97	4,97	4,97			4,97	0,00	4,97
Non-Ferrous Metal	kg/hr	19,45	19,45	19,45	19,45	0,00		19,45	0,00	19,45
Cu	kg/hr	12,43	12,43	12,43	12,43			12,43	0,00	12,43
Al	kg/hr	1,24	1,24	1,24	1,24			1,24	0,00	1,24
Sn	kg/hr	2,49	2,49	2,49	2,49			2,49	0,00	2,49
Ni	kg/hr	1,24	1,24	1,24	1,24			1,24	0,00	1,24
Zn	kg/hr	0,62	0,62	0,62	0,62			0,62	0,00	0,62
Pb	kg/hr	1,24	1,24	1,24	1,24			1,24	0,00	1,24
Mn	kg/hr	0,00	0,00	0,00	0,00			0,00	0,00	0,00
Sb	kg/hr	0,00	0,00	0,00	0,00			0,00	0,00	0,00
Au	kg/hr	0,06	0,06	0,06	0,06			0,06	0,00	0,06
Ag	kg/hr	0,12	0,12	0,12	0,12			0,12	0,00	0,12
Ceramic	kg/hr	18,32	18,32	18,32	18,32			18,32	0,00	18,32
SiO2	kg/hr	9,16	9,16	9,16	9,16			9,16	0,00	9,16
Al2O3	kg/hr	3,66	3,66	3,66	3,66			3,66	0,00	3,66
Alkali and alkaline earth oxides	kg/hr	3,66	3,66	3,66	3,66			3,66	0,00	3,66
Titanates and mica, etc.	kg/hr	1,83	1,83	1,83	1,83			1,83	0,00	1,83
Plastic	kg/hr	18,32	18,32	18,32	18,32			18,32	0,00	18,32
Polyethylene	kg/hr	6,04	6,04	6,04	6,04			6,04	0,00	6,04
Polypropylene	kg/hr	2,93	2,93	2,93	2,93			2,93	0,00	2,93
Polyesters	kg/hr	2,93	2,93	2,93	2,93			2,93	0,00	2,93
Epoxies	kg/hr	2,93	2,93	2,93	2,93			2,93	0,00	2,93
Polyvinyle chloride	kg/hr	1,47	1,47	1,47	1,47			1,47	0,00	1,47
Polytetraflouroethane	kg/hr	1,47	1,47	1,47	1,47			1,47	0,00	1,47
Nylon	kg/hr	0,55	0,55	0,55	0,55			0,55	0,00	0,55
H2O	kg/hr	0,00	0,00	0,00	0,00		0,00	0,00	0,00	0,00

Table A.2: Mass balances for the solder leaching plant at a design basis of 400Tonnes processing capacity of WPCBs (adapted from Rossouw, 2015; de Waal, 2018)

Solder leaching mass balance	Units	Stream No. 9	10	11	12	13	14
Temperature	(K)	298,15	298,15	306,00	306,00	306,00	306,00
Pressure	(kPa)	101,32	405,00	1600,00	101,33	101,33	101,33
Material Flow (Total)	kg/hr	61,05	957,30	1015,45	960,22	55,23	2,95
Metal Flow (Total)	kg/hr	24,42		18,60		18,60	0,00
E-Waste Elements							
Ferrous Metal (Elemental)	kg/hr	4,97		1,18		1,18	
Fe	kg/hr	4,97		1,18		1,18	
Non-Ferrous Metal (Elemental)	kg/hr	19,45		17,41		17,41	
Cu	kg/hr	12,43		12,40		12,40	
Al	kg/hr	1,24		1,05		1,05	
Sn	kg/hr	2,49		2,31		2,31	
Ni	kg/hr	1,24		0,83		0,83	
Zn	kg/hr	0,62		0,61		0,61	
Pb	kg/hr	1,24		0,03		0,03	
Mn	kg/hr	0,00		0,00		0,00	
Sb	kg/hr	0,00		0,00		0,00	
Au	kg/hr	0,06		0,06		0,06	
Ag	kg/hr	0,12		0,12		0,12	
Ceramic	kg/hr	18,32		18,32		18,32	
SiO2	kg/hr	9,16		9,16		9,16	
Al2O3	kg/hr	3,66		3,66		3,66	
Alkali and alkaline earth oxides	kg/hr	3,66		3,66		3,66	
Titanates and mica, etc.	kg/hr	1,83		1,83		1,83	
Plastic	kg/hr	18,32		18,32		18,32	
Polyethylene	kg/hr	6,04		6,04		6,04	
Polypropylene	kg/hr	2,93		2,93		2,93	
Polyesters	kg/hr	2,93		2,93		2,93	
Epoxies	kg/hr	2,93		2,93		2,93	
Polyvinyle chloride	kg/hr	1,47		1,47		1,47	
Polytetraflouroethane	kg/hr	1,47		1,47		1,47	
Nylon	kg/hr	0,55		0,55		0,55	
Reagents							
H2O	kg/hr	0,00	900,55	900,55	903,55		
HNO3	kg/hr		56,75	35,06	35,06		
NaCl	kg/hr						
Products							
Pb(NO3)2	kg/hr			1,94	1,94		
Sn(NO3)2	kg/hr			0,36	0,36		
Ni(NO3)2	kg/hr			1,30	1,30		
Zn(NO3)2	kg/hr			0,03	0,03		
Fe(NO3)3	kg/hr			16,40	16,40		
Al(NO3)3	kg/hr			1,50	1,50		
AgNO3	kg/hr			0,01	0,01		
Cu(NO3)2	kg/hr			0,07	0,07		
NO2	kg/hr						0,65
NO	kg/hr						2,29
H2	kg/hr						0,01
AgCl	kg/hr						
NaNO3	kg/hr						
Cu(NO3)2	kg/hr						
H2O				2,99			
Phase		Liquid	Liquid	Slurry	Liquid	Solids	Gas

Table A.3: Mass balances for the copper extraction plant at a design basis of 400Tonnes processing capacity of WPCBs (adapted from Rossouw, 2015; de Waal, 2018)

Copper extraction mass balance			Leaching								SX						EW	
	Units	13	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29
Temperature	(K)	306,00	298,00	298,00	312,00	312,00	312,00	312,00	298,00	312,00			301,00	301,00	301,00	345,00	345,00
Pressure	(kPa)	101,33	400,00	400,00		1600,00	400,00	101,00	400,00	400,00			300,00	343,00	300,00	101,33	300,00
Material Flow (Total)	kg/hr	59,01	29,66	273,81	0,00	359,13	41,62	317,51	3,96	321,47			308,19	747,96	737,20	9,70	2,44
Metal Flow (Total)		22,38															
E-Waste Elements																	
Ferrous Metal (Elemental)	kg/hr	4,97				0,81	0,81										
Fe	kg/hr	4,97				0,81	0,81										
	kg/hr																
Non-Ferrous Metal (Elemental)	kg/hr	17,41				4,18	4,18										
Cu	kg/hr	12,403				0,43	0,4341										
Al	kg/hr	1,05				0,86	0,86										
Sn	kg/hr	2,31				2,29	2,29										
Ni	kg/hr	0,83				0,35	0,35										
Zn	kg/hr	0,61				0,04	0,04										
Pb	kg/hr	0,03				0,03	0,03										
Mn	kg/hr	0,00				0,00	0,00										
Sb	kg/hr	0,00				0,00	0,00										
Au	kg/hr	0,06				0,06	0,06										
Ag	kg/hr	0,12				0,11	0,11										
Ceramic	kg/hr	18,32				18,32	18,32										
SiO2	kg/hr	9,16				9,16	9,16										
Al2O3	kg/hr	3,66				3,66	3,66										
Alkali and alkaline earth oxides	kg/hr	3,66				3,66	3,66										
Titanates and mica, etc.	kg/hr	1,83				1,83	1,83										
Plastic	kg/hr	18,32				18,32	18,32										
Polyethylene	kg/hr	6,04				6,04	6,04										
Polypropylene	kg/hr	2,93				2,93	2,93										
Polyesters	kg/hr	2,93				2,93	2,93										
Epoxies	kg/hr	2,93				2,93	2,93										
Polyvinyle chloride	kg/hr	1,47				1,47	1,47										
Polytetraflouroethane	kg/hr	1,47				1,47	1,47										
Nylon	kg/hr	0,55				0,55	0,55										
Reagents																	
H2O	kg/hr		20,76	228,91		249,66		257,98		257,98			257,98	0,48			
HNO3	kg/hr																
NaCl	kg/hr																
H2SO4	kg/hr			44,90		22,28		22,28		22,28			22,28				
H2O2	kg/hr		8,90			1,05		1,05		1,05			1,05				
LIX984N	kg/hr																
Cobalt sulphate 12,5%	kg/hr													0,00478			
Kerosene	kg/hr																
Products																	
Pb(NO3)2	kg/hr																
H2SnO3	kg/hr																
Ni(NO3)2	kg/hr																
Zn(NO3)2	kg/hr																
Fe(NO3)3	kg/hr																
Al(NO3)3	kg/hr																
AgNO3	kg/hr																
NO2	kg/hr																
NO	kg/hr																
H2	kg/hr				0,00												
AgCl	kg/hr																
NaNO3	kg/hr																
Cu(NO3)2	kg/hr																
CuSO4	kg/hr					30,06		30,06		30,06			6,01	81,25	56,87		
Al2(SO4)3	kg/hr					1,23		1,23		1,23			1,23				
FeSO4	kg/hr					2,20		2,20		2,20			2,20				
NiSO4	kg/hr					1,26		1,26		1,26			1,26				
ZnSO4	kg/hr					1,41		1,41		1,41			1,41				
PbSO4	kg/hr					0,00	0,00									2,75	
SnSO4	kg/hr					0,03		0,03		0,03							
Ag2SO4	kg/hr					0,02		0,02		0,02							
H2O	kg/hr					8,31								564,76	563,40	1,36	
H2SO4(aq)	kg/hr												14,78	101,47	116,44		
O2	kg/hr																2,44
Cu	kg/hr															9,704	
CoSO4 12,5%	kg/hr														0,48318		
Copper Balance for SX/EW																	
V	L/hr							282,16	3,80	285,95	1010,80		285,95	646,91	646,91		
Cu	kg/hr					11,97		11,97		11,97	9,70		2,39	32,35	22,64	9,704	
Cu (g/l)	g/l							42,42		41,85	9,60		8,371	50,00	35,00		
H2SO4	g/l							78,95		77,90				156,85	180,00		
H2SO4(M)	M																
pH								1,37		1,80							
NaOH	M								2,00								
NaOH	kg/hr								0,30	0,30							
H2O	kg/hr								3,65	3,65							
Phase	kg/hr	Solids			LS + Cu-Raffinate		PLS		PLS		Loaded Org	Stripped Org	Raffinate	A-Electrolyte	S-Electrolyte	Cu- cathode	Gas

Table A.4: Mass balances for the gold extraction plant at a design basis of 400Tonnes processing capacity of WPCBs

(adapted from Rossouw, 2015; de Waal, 2018)

Gold extraction mass balances	Units	19	Leaching 30	31	32	33	34	35	36	Filtering 37	38	Adsorption 39	40	41	Acid Washing 42	43	44	45	Elution 46	47	48	49	50	Electrowinning 51	52	53
Temperature (K)		312,00	298,00	298,00	298,00	298,00	298,00		299,00	299,00	299,00		298,00	298,00	298,00	298,00	298,00	298,00	298,00	298,00	298,00	363,00	363,00	363,00	363,00	363,00
Pressure (kPa)		400,00	400,00	101,33	300,00	101,33	101,33		1600,00	400,00	350,00		450,00	450,00	450,00	450,00	450,00	450,00	450,00	450,00	450,00	400,00	400,00	400,00	400,00	400,00
Material Flow (Total)	kg/hr	43,27	116,13	309,66	0,00	467,51	0,01	0,01	462,48	61,92	405,60	36,37	404,10	36,45	150,67	602,67	36,45	753,33	292,38	9,03	301,33	602,75	36,37	0,01	602,69	0,06
Metal Flow (Total)		3,32																								
Ferrous Metal (Elemental)	kg/hr	0,00			0,00	0,00		0,00	0,00	0,00	0,00		0,00				0,00	0,00								
Fe	kg/hr	0,00029	0,00		0,00	0,00029			0,00029		0,00029		0,00029				0,00000	0,00029								
Non-Ferrous Metal (Elemental)	kg/hr	3,32				0,06	0,00		0,00	0,00	0,00		0,00				0,00	0,00				0,00				
Cu	kg/hr	0,00016				0,0002			0,00016		0,00016		0,00016				0,00016	0,00000				0,00016				
Al	kg/hr	0,00031				0,0003			0,00031		0,00031		0,00031				0,00000	0,00031								
H2SnO3	kg/hr	3,25891				0,0000			0,00000	0,00	0,00000		0,00000				0,00000	0,00000								
Ni	kg/hr	0,00013				0,0001			0,00013		0,00013		0,00013				0,00000	0,00013								
Zn	kg/hr	0,00001				0,0000			0,00001		0,00001		0,00001				0,00000	0,00001								
Pb	kg/hr	0,00000				0,0000			0,00000		0,00000		0,00000				0,00000	0,00000								
Mn	kg/hr	0,00000				0,0000			0,00000		0,00000		0,00000				0,00000	0,00000								
Sb	kg/hr	0,00000				0,0000			0,00000		0,00000		0,00000				0,00000	0,00000								
Au	kg/hr	0,06214				0,0621			0,00000		0,00000		0,00000				0,00000	0,00000								
Ag	kg/hr	0,00004				0,0000			0,00000		0,00000		0,00000				0,00000	0,00000								
Ceramic	kg/hr	18,32			0,00	18,32			18,32	18,32	0,00		0,00				0,00	0,00								
SiO2	kg/hr	9,16			0,00	9,16			9,16	9,16	0,00		0,00				0,00	0,00								
Al2O3	kg/hr	3,66			0,00	3,66			3,66	3,66	0,00		0,00				0,00	0,00								
Alkali and alkaline earth oxides	kg/hr	3,66			0,00	3,66			3,66	3,66	0,00		0,00				0,00	0,00								
Titanates and mica, etc.	kg/hr	1,83			0,00	1,83			1,83	1,83	0,00		0,00				0,00	0,00								
Plastic		18,32			0,00	18,32			18,32	18,32	0,00		0,00				0,00	0,00								
Polyethylene	kg/hr	6,04			0,00	6,04			6,04	6,04	0,00		0,00				0,00	0,00								
Polypropylene	kg/hr	2,93			0,00	2,93			2,93	2,93	0,00		0,00				0,00	0,00								
Polyesters	kg/hr	2,93			0,00	2,93			2,93	2,93	0,00		0,00				0,00	0,00								
Epoxies	kg/hr	2,93			0,00	2,93			2,93	2,93	0,00		0,00				0,00	0,00								
Polyvinyl chloride	kg/hr	1,47			0,00	1,47			1,47	1,47	0,00		0,00				0,00	0,00								
Polytetraflouroethane	kg/hr	1,4652			0,00	1,47			1,47	1,47	0,00		0,00				0,00	0,00								
Nylon	kg/hr	0,5495			0,00	0,55			0,55	0,55	0,00		0,00				0,00	0,00								
H2O	kg/hr	0,00	114,68	309,58		424,26			424,26	21,21	404,09		404,09		146,15	602,67		748,81	286,28		301,33	587,61			587,61	
Ca(OH)2	kg/hr		1,45			1,45			1,45	0,07	1,37															
Ca(OH)2	ppm		12616																							
Air (enriched Oxygen)	kg/hr						0,0068	0,00																		
N2	kg/hr						0,0048	0,00																		
NaCN	kg/hr			0,08		0,08			0,05	0,00	0,04									9,03		9,03			9,03	
NaOH	kg/hr																		6,03			6,03			6,00	
Carbon	kg/hr											36,37		36,37			36,37						36,37			
HCl	kg/hr														4,52			4,52								
V	L/hr														150,67	150,67			301,33				0,00			
pH desired						11,00																				
pH actual						12,96																				
NaOH	kg/hr								0,012	0,001	0,012		0,012			0,01									0,00	
NaCN	kg/hr																								0,040	
NaAu(CN)2	kg/hr								0,08	0,00	0,08		0,00	0,08	0,08	0,079			0,079		0,08	0,001			0,002	
NaAg(CN)2	kg/hr								0,00	0,00	0,00		0,00	0,00	0,00	0,00			0,0001		0,00	0,000			0,000	
Au	kg/hr																									0,05613
Ag	kg/hr																									0,00
O2	kg/hr																							0,01		
H2	kg/hr																							0,00		
H2O	kg/hr					1,04			1,04	1,04	1,04															
CaSnO3	kg/hr					3,99			3,99	3,99																

E-Waste Elements / Reagents / Products (row group labels)

# B:	Mass balances for base metal treatment option – The Nitric acid wash

Nitric acid wash circuit

Figure B.1: Nitric acid wash circuit process flow sheet

Table B.1: Detailed mass balances for the nitric acid wash process flow sheet

	Basis 400 Tonnes per annum	Units	19	19a	19b	19c	19d	19e	19f	19g	19h	19i	19j	19k	19l	19m	Mr	Density/SG g/cm3
	Temperature	K				>363	>338											
	Pressure	Kpa																
	Material flows	Kg/hr	41,62	68,70		29,62	455,62	199,22	44,33	236,88	281,22	237,18	0,00	44,04	44,04	29,62		
	Feed						20% solids						acid 10-18%					
Cake from	Fe	Kg/hr	0,81														55,85	7,87
Copper leach	Cu	Kg/hr	0,43														63,55	8,96
	Al	Kg/hr	0,86														26,98	2,70
	Sn	Kg/hr	2,29														118,70	7,31
	Ni	Kg/hr	0,35														58,69	8,90
	Zn	Kg/hr	0,04														65,41	7,13
	Pb	Kg/hr	0,03														207,20	11,35
	Mn	Kg/hr	0,00														54,94	7,43
	Sb	Kg/hr	0,00														121,80	6,70
	Au	Kg/hr	0,06														196,96	19,32
	Ag	Kg/hr	0,11														107,90	10,50
Ceramics	SiO2	Kg/hr	9,16				9,16		9,16		9,16			9,16	9,16			2,27
	Al2O3	Kg/hr	3,66				3,66		3,66		3,66			3,66	3,66			3,89
	Alkali & Alkali earth oxides	Kg/hr	3,66				3,66		3,66		3,66			3,66	3,66			3,00
	Titanates and Mica	Kg/hr	1,83				1,83		1,83		1,83			1,83	1,83			3,50
Plastics	Polyethylene	Kg/hr	6,04				6,04		6,04		6,04			6,04	6,04			1,40
	Polypropylene	Kg/hr	2,93				2,93		2,93		2,93			2,93	2,93			1,24
	Polyesters	Kg/hr	2,93				2,93		2,93		2,93			2,93	2,93			8,95
	Epoxies	Kg/hr	2,93				2,93		2,93		2,93			2,93	2,93			1,25
	Polyvinylchloride	Kg/hr	1,47				1,47		1,47		1,47			1,47	1,47			1,47
	Polytetraflouroethane	Kg/hr	1,47				1,47		1,47		1,47			1,47	1,47			2,20
	Nylon	Kg/hr	0,55				0,55		0,55		0,55			0,55	0,55			1,15
	Reagents																	
Reagents and	Water	Kg/hr		21,32			212,07			236,88							18,00	1,00
utilities	55% Nitric acid	Kg/hr		47,38													63,00	1,34
	Bleach air	Kg/hr																0,00
	Ca(OH)2	Kg/hr															74,10	
	N2	Kg/hr															28,00	
	NaCN	Kg/hr															49,10	
	NaOH	Kg/hr															40,00	
	Ca(OH)2	Kg/hr															74,10	
	HCl	Kg/hr															36,50	
	V	L/hr																
	pH desired											>6.5			>6.5				
	pH actual																	
	Products																	
Products	Fe(NO3)3	Kg/hr					3,51	3,44	0,07		0,07	0,07		0,00	0,00		241,86	1,68
	Cu(NO3)2	Kg/hr					1,28	1,26	0,03		0,03	0,03		0,00	0,00		187,56	3,05
	Al(NO3)3	Kg/hr					6,78	6,64	0,14		0,14	0,14		0,00	0,00		213,00	1,72
	Sn(NO3)2	Kg/hr					0,00	0,00	0,00		0,00	0,00		0,00	0,00		242,72	
	Ni(NO3)2	Kg/hr					1,08	1,06	0,02		0,02	0,02		0,00	0,00		182,70	2,05
	Zn(NO3)2	Kg/hr					0,12	0,11	0,00		0,00	0,00		0,00	0,00		189,36	2,06
	Pb(NO3)2	Kg/hr					0,04	0,04	0,00		0,00	0,00		0,00	0,00		331,20	4,53
	Mn(NO3)2	Kg/hr					0,00	0,00	0,00		0,00	0,00		0,00	0,00		178,95	1,54
	Sb2O3	Kg/hr					0,00	0,00	0,00		0,00	0,00		0,00	0,00		291,58	5,20
	AgNO3	Kg/hr					0,18	0,17	0,00		0,00	0,00		0,00	0,00		169,87	4,35
	Au	Kg/hr					0,06	0,00	0,06		0,06	0,00		0,06	0,06		196,96	19,32
	H2	Kg/hr				0,00	0,00	0,00	0,00		0,00	0,00			0,00		2,00	0,00
	NO2	Kg/hr				3,98	0,00	0,00	0,00		0,00	0,00			0,00		46,00	0,00
	NO	Kg/hr				0,78	0,00	0,00	0,00		0,00	0,00			0,00	0,78	30,00	0,00
	NH4NO3	Kg/hr					0,00	0,00	0,00		0,00	0,00		0,00	0,00		80,00	1,72
	H2SnO3	Kg/hr					3,26	0,00	3,26		3,26	0,00		3,26	3,26		168,70	
	H20	Kg/hr				24,86	190,61	186,50	4,11		240,99	236,91		4,08	4,08	24,08	18,00	1,00
	HNO2	Kg/hr															2,03	
	HNO3	Kg/hr															2,72	

Table B.2: Summary of the main reactions proceeding during the nitric acid wash

Reaction outputs (Literature)

Original Basis 400 Tes/yr feed rate per hour

Reactions proceeding with 55% HNO3 at ambient conditions	Metal/Element	Mass in feed	Moles in feed	Products							Acid required 55%	Mr element	Mr Oxide	Mr stannic	Mr Nitrate
				NO2	NO	H20	Nitrate	Oxide	Stannic acid	H2					
		Kg/hr	moles/hr	Kg/hr	Kg/hr	Kg/Hr	Kg/hr	Kg/hr	Kg/hr	Kg/hr	Kg/hr	g/mole	g/mol	g/mol	g/mol
Fe + 6HNO3 ⟶ Fe(NO3)3 + 3NO2 + 3H2O	Fe	0,81	14,52	2,00	0,00	0,78	3,51	0,00	0,00	0,00	9,98	55,85	N/A	N/A	241,86
Cu + 4HNO3 ⟶ Cu(NO3)2 + 2NO2 + 2H2O	Cu	0,43	6,84	0,63	0,00	0,25	1,28	0,00	0,00	0,00	3,13	63,55	N/A	N/A	187,56
2Al + 6HNO3 ⟶ Al2O3 + NO2 + 3H2O	Al	0,86	31,84	0,73	0,00	0,86	0,00	1,62	0,00	0,00	21,88	26,98	101,96	N/A	ʒ213,00
Al2O3 + 6HNO3 ⟶ 2Al(NO3)3 + 3H2O		0,00	0,00	0,00	0,00	0,86	6,78	0,00	0,00	0,00					N/A
3Sn + 4HNO₃ + nH₂O ⟶ 3H₂SnO₃. nH2O + 4NO	Sn	2,29	19,32	0,00	0,77	0,26	0,00	0,00	3,26	0,00	8,85	118,70	N/A	168,70	242,72
Ni + 4HNO3 ⟶ Ni(NO3)2 + 2NO2 + 2H2O	Ni	0,35	5,93	0,55	0,00	0,32	1,08	0,00	0,00	0,00	2,72	58,69	N/A	N/A	182,70
Zn + 4HNO3 ⟶ Zn(NO3)2 + 2NO2 + 2H2O	Zn	0,04	0,62	0,06	0,00	0,03	0,12	0,00	0,00	0,00	0,28	65,41	N/A	N/A	189,36
Pb + 4HNO3 ⟶ Pb(NO3)2 + 2NO2 + H2O	Pb	0,03	0,13	0,01	0,00	0,01	0,04	0,00	0,00	0,00	0,06	207,20	N/A	N/A	331,20
Mn + 2HNO3 ⟶ Mn(NO3)2 + H2	Mn	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	54,94	N/A	N/A	N/A
2Sb + 2HNO3 ⟶ Sb2O3 + 2NO + H2O	Sb	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	121,80	291,58	N/A	N/A
3Ag + 4HNO3 ⟶ 3AgNO3 + 2H2O + NO	Ag	0,11	1,04	0,00	0,01	0,01	0,18	0,00	0,00	0,00	0,48	107,90	N/A	N/A	169,87
Au + 3HNO3 ⟶ Au + 3HNO3	Au	0,06	0,32	0,00	0,00	0,02	0,06	0,00	0,00	0,00	0,00	196,96	N/A	N/A	N/A
Total		4,99	80,54	3,98	0,78	3,40	13,06	1,62	3,26	0,00	47,38				

Assumptions for the calculations: For the thermodynamic data available there is a predicted heat rise in the system. All fine aluminium in oxide and metallic form ultimately reacts with HNO_3, all reactions proceed to completion, metals are in solid form and salts produced exist in the aqueous media with respect to their solubilities. Heat produced will catalyse the decomposition of nitric acid. During pilot tests, factor in the excess acid to be added to ensure reactions proceed to completion. An ideal basis is assumed for the calculations wherein all NO_2 and NO is generated solely from the reactions listed in the Table above. The maximum tolerable moisture content in solids for the vacuum filter with minimum cake thickness of 5mm is 10% without upstream and down thickening operations.

Table B.3: Evaluation of main objective of the acid wash performance

Analysis of feed stream
Initial feed ex Copper circuit

Component	Kg/hr	Percentage composition
Fe	0,81	1,95
Cu	0,43	1,04
Al	0,86	2,06
Sn	2,29	0,27
Ni	0,35	0,04
Zn	0,04	0,00
Pb	0,03	0,00
Mn	0,00	0,00
Sb	0,00	0,00
Ag	0,11	0,01
Au	0,06	0,01
NH4NO3	0,00	0,00
H2SnO3	0,00	0,00
H20	0,00	0,00
Total leachables	**4,99**	
SiO2	9,16	1,09
Al2O3	3,66	0,44
Alkali & Alkali earth oxides	3,66	0,44
Titanates and Mica	1,83	0,22
Polyethylene	6,04	0,72
Polypropylene	2,93	0,35
Polyesters	2,93	0,35
Epoxies	2,93	0,35
Polyvinylchloride	1,47	0,17
Polytetraflouroethane	1,47	0,17
Nylon	0,55	0,07
Total mass	**41,62**	**4,97**

Analysis of washed stream
Feed ex Acid and water wash

Component	Kg/hr to T402 (19f)	Kg/hr of pure metals	Percentage composition as nitrates	Δ component (mass)	Δ component (%)	Proportion of dissolved salts in liquid phase
Fe(NO3)3	0,07	0,02	0,17	-0,79	-1,78	0,02
Cu(NO3)2	0,03	0,01	0,06	-0,43	-0,98	0,01
Al(NO3)3	0,14	0,02	0,32	-0,84	-1,74	0,03
Sn(NO3)2	0,00	0,00	0,00	-2,29	-0,27	0,00
Ni(NO3)2	0,02	0,01	0,05	-0,34	0,01	0,01
Zn(NO3)2	0,00	0,00	0,01	-0,04	0,00	0,00
Pb(NO3)2	0,00	0,00	0,00	-0,03	0,00	0,00
Mn(NO3)2	0,00	0,00	0,00	0,00	0,00	0,00
Sb2O3	0,00	0,00	0,00	0,00	0,00	0,00
AgNO3	0,00	0,00	0,01	-0,11	0,00	0,00
Au	0,06	0,06	0,00	0,00	-0,01	
NH4NO3	0,00	0,00	0,00	0,00	0,00	0,00
H2SnO3	3,26	2,29	0,00	0,00	0,00	0,00
H20	4,11	0,00	9,26	0,00	9,26	0,92
Total leachables	**7,70**	**2,41**				206,86
SiO2	9,16	68,68	1,05	59,52	-0,04	
Al2O3	3,66	27,47	0,42	23,81	-0,02	
Alkali & Alkali earth oxides	3,66	27,47	0,42	23,81	-0,02	
Titanates and Mica	1,83	13,74	0,21	11,91	-0,01	
				0,00		
Polyethylene	6,04	45,33	0,69	39,29	-0,03	
Polypropylene	2,93	21,98	0,34	19,05	-0,01	
Polyesters	2,93	21,98	0,34	19,05	-0,01	
Epoxies	2,93	21,98	0,34	19,05	-0,01	
Polyvinylchloride	1,47	10,99	0,17	9,52	-0,01	
Polytetraflouroethane	1,47	10,99	0,17	9,52	-0,01	
Nylon	0,55	4,12	0,06	3,57	0,00	
Total mass	**44,33**	**277,14**	**5,09**	235,52	0,12	

Table B.4: Stream characterisation after water rinsing

Rinsing of acid washed solids in T402
Worked with basis 400Tes/yr

Material flows of solids ex T402 stream 19h				Material flows of solids ex T402 stream 19k after filtration				
Materials	Amount exiting T402 (kg/hr) 19h	Proportion of soluble salts	Proportion of soluble salts %	Materials	Amount exiting F402 (kg/hr) 19k	Proportion of materials in stream 19k %	Proportion of materials after acid washing 19h%	Δ component (%)
Fe(NO3)3	0,07	0,00	0,03	Fe(NO3)3	0,00	0,00	0,03	-0,02
Cu(NO3)2	0,03	0,00	0,01	Cu(NO3)2	0,00	0,00	0,01	-0,01
Al(NO3)3	0,14	0,00	0,06	Al(NO3)3	0,00	0,01	0,05	-0,05
Sn(NO3)2	0,00	0,00	0,00	Sn(NO3)2	0,00	0,00	0,00	0,00
Ni(NO3)2	0,02	0,00	0,01	Ni(NO3)2	0,00	0,00	0,01	-0,01
Zn(NO3)2	0,00	0,00	0,00	Zn(NO3)2	0,00	0,00	0,00	0,00
Pb(NO3)2	0,00	0,00	0,00	Pb(NO3)2	0,00	0,00	0,00	0,00
Mn(NO3)2	0,00	0,00	0,00	Mn(NO3)2	0,00	0,00	0,00	0,00
Sb2O3	0,00	0,00	0,00	Sb2O3	0,00	0,00	0,00	0,00
AgNO3	0,00	0,00	0,00	AgNO3	0,00	0,00	0,00	0,00
Au	0,06	0,00	0,00	Au	0,06	0,14	0,02	0,12
NH4NO3	0,00	0,00	0,00	NH4NO3	0,00	0,00	0,00	0,00
H2SnO3	3,26	0,00	0,00	H2SnO3	3,26	7,40	1,16	
H20	240,99	0,99	98,56	H20	4,08	9,27	85,70	-76,42
Total solubles	**3,53**	0,00		**Total solubles**	0,12	0,01	1,26	-1,25
SiO2	9,16			SiO2	9,16	20,79	3,26	17,54
Al2O3	3,66			Al2O3	3,66	8,32	1,30	7,01
Alkali & Alkali earth oxides	3,66			Alkali & Alkali earth oxides	3,66	8,32	1,30	7,01
Titanates and Mica	1,83			Titanates and Mica	1,83	4,16	0,65	3,51
						0,00	0,00	0,00
Polyethylene	6,04			Polyethylene	6,04	13,72	2,15	11,57
Polypropylene	2,93			Polypropylene	2,93	6,65	1,04	5,61
Polyesters	2,93			Polyesters	2,93	6,65	1,04	5,61
Epoxies	2,93			Epoxies	2,93	6,65	1,04	5,61
Polyvinylchloride	1,47			Polyvinylchloride	1,47	3,33	0,52	2,81
Polytetraflouroethane	1,47			Polytetraflouroethane	1,47	3,33	0,52	2,81
Nylon	0,55			Nylon	0,55	1,25	0,20	1,05
Total material flow	**281,22**			**Total material flow**	**44,04**		100,00	

Notes:

For vacuum filtration the maximum tolerable moisture retained in solid should not exceed 10% (Literature). Using simultaneous equations the solute in stream 19k was calculated to be 0.116 at a retained moisture content of 10% in the filter cake. After acid washing and rinsing the percentage composition of the gold has been upgraded to 1.35% up from 0.50% in the solid material exiting the copper leach and recovery circuit.

Table B.5: Evaluating likelihood of premature crystallization in the circuit

Prediction of premature crystallization

Basis at 400Tes/ annum and T = 298K

Salt	Solubility g/100g H_2O	Moisture demand at saturation factor	Moisture demand at saturation (kg/hr)	Residual H_2O in T401 kg/hr
				190.61
$Fe(NO_3)_3$	150	0.67	2.34	188.26
$Cu(NO_3)_2$	381	0.26	0.34	187.93
$Al(NO_3)_3$	73	1.37	9.29	178.64
H_2SnO_3	NA	NA	NA	NA
$Ni(NO_3)_2$	243	0.41	0.45	178.19
$Zn(NO_3)_2$	184.3	0.54	0.12	178.08
$Pb(NO_3)_2$	597	0.17	0.01	178.07
$Mn(NO_3)_2$	118	0.85	0.00	178.07
Sb_2O_3	0.037	2702.70	0.15	177.92
$AgNO_3$	256	0.39	0.07	177.85
Au	NA	NA	0	177.85
H_2	NA	NA	0	177.85
NO_2	NA	NA	0	177.85
NO	NA	NA	0	177.85
NH_4NO_3	150	0.67	0	177.85
Net H_2O			**12.75**	**177.85**

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Remarks

A positive net demand of water has been calculated for maximum solubilities of the nitrates between 293K and 298K. The value predicts that no crystallization of the nitrates is expected in the nitric acid wash circuit. All salts listed are soluble

C: Calculations for process considerations

Table C.1: Performance comparison of two options for managing the increased gold concentration as a result of the nitric acid wash

Comparison of dilution versus increasing carbon concentration 400 tonnes per annum plant design basis Figures analysed at an hourly rate			
Option	**Units**	**Dilution**	**Increase carbon concentration**
Solution flow capacity	g/L	1730.24	402.64
Gold to tailings in barren solution	g/hr	0.58	0.58
Carbon concentration	g/L	15.00	58.76
Gold on carbon locked in adsorption circuit	g/hr	0.56	2.20
Carbon losses	g/t hr	4.67	18.69
Gold losses from carbon lost	g/hr	0.02	0.07
Total gold losses	**g/hr**	**0.60**	**0.65**

Remarks

Introducing the nitric acid wash circuit to remove base metals resulted in the head grade of gold reporting to the adsorption being increased by a factor of 3.92. To maintain the steady state performance and pseudo equilibrium conditions defined in the previous work by de Waal (2018) it was proposed to increase the carbon concentration to match the increased gold head grade to adsorption or to dilute the feed to the concentration used in the original studies. Gold losses associated with increasing the carbon concentration were predicted to be higher than the dilution scenario by a margin of 8.55%. An excess of the spent electrolyte recycled to the leach solution can also be used in the dilution.

Basis of calculations

Gold extraction calculations with acid wash 400Tonnes/hour design basis (de Waal, 2018; Rogans, 2012). Losses of carbon (Davidson & Schoeman, 1991; Rogans, 2012)

Determination of gold losses with increasing carbon loading (Stange, 1999; Snyders *et al.,* 2017)

Table C.2: Costs associated with dilution as an option round the adsorption circuit

Costs associated with dilution as an option around adsorption circuit

400 tonnes per annum plant design basis

Figures analysed at an hourly rate

Costs at start up

	Units	Dilution	Unit cost	Total cost US $
Activated carbon inventory	g/L	155.72	$2.00/kg	0.31
Water	kg/hr	1 730.24	$1.41/Kl	2.44
Hydrated lime Ca(OH)$_2$	kg/hr	0.05	US$93/tonne	0.00
Carbon losses	g/t hr	4.67	$2.00/kg	0.01
Gold losses from carbon lost	g/hr	0.02	US$39.35	0.73
Gold to tailings in barren solution	g/hr	0.58	US$39.35	22.80
Total costs at start up				**26.30**

Running costs at steady state

	Units	Dilution	Equivalent cost US $	Total cost US$/hr
Water	kg/hr	1 327.60	$1.41/Kl	1.87
Hydrated lime Ca(OH)$_2$	kg/hr	0.05	US$93.00/tonne	0.00
Carbon losses	g/t hr	4.67	$2.00/kg	0.01
Gold losses from carbon lost	g/hr	0.02	US$39.35	0.73
Gold to tailings in barren solution	g/hr	0.58	US$39.35	22.80
Total operational costs at steady state				**25.42**

Source of prices

(Alibaba, 2018; Infomine (2017); www.johannesburgwater, 2017)

Table C.3: Costs associated with increased carbon concentration as an option around the adsorption circuit

Costs associated with increased carbon concentration as an option around adsorption circuit

400 tonnes per annum plant design basis

Figures analysed at an hourly rate

Costs at start up

	Units	Dilution	Unit cost	Total cost US $
Activated carbon inventory	g/L	610.03	$2.00/kg	1.22
Water	kg/hr	402.64	$1.41/Kl	0.57
Hydrated lime Ca(OH)$_2$	kg/hr	0.00	US$93.00/tonne	0.00
Carbon losses	g/t hr	18.69	$2.00/kg	0.04
Gold losses from carbon lost	g/hr	0.07	US$39.35	2.93
Gold to tailings in barren solution	g/hr	0.58	US$39.35	22.80
Total costs at start up				**27.56**

Running costs at steady state

	Units	Dilution	Equivalent cost US $	Total cost US$/hr
Water	kg/hr	402.64	$1.41/Kl	0.57
Hydrated lime Ca(OH)$_2$	kg/hr	0.00	US$93.00/tonne	0.00
Carbon losses	g/t hr	18.69	$2.00/kg	0.04
Gold losses from carbon lost	g/hr	0.07	US$39.35	2.93
Gold to tailings in barren solution	g/hr	0.58	US$39.35	22.80
Total operational costs at steady state				**26.34**

Source of prices

(Alibaba, 2018; Infomine (2017); www.johannesburgwater, 2017)

Table C.4: Nitric acid wash overall performance analysis for removal of base metals in WPCB recycling campaign

Nitric acid wash performance analysis for removal of base metals **Basis 400 Tonnes per annum**			
Feed	Quantities in feed kg/hr	Pure metals after acid wash kg/hr	Composition to adsorption circuit kg/hr
Fe	0.81	0.00	0.00
Cu	0.43	0.00	0.00
Al	0.86	0.00	0.00
Sn	2.29	0.00	0.00
Ni	0.35	0.00	0.00
Zn	0.04	0.00	0.00
Pb	0.03	0.00	0.00
Mn	0.00	0.00	0.00
Sb	0.00	0.00	0.00
Au	0.06	0.06	0.06
Ag	0.11	0.00	0.00
SiO_2	9.16	9.16	0.00
Al_2O_3	3.66	3.66	0.00
Alkali & Alkali earth oxides	3.66	3.66	0.00
Titanates and Mica	1.83	1.83	0.00
Polyethylene	6.04	6.04	0.00
Polypropylene	2.93	2.93	0.00
Polyesters	2.93	2.93	0.00
Epoxies	2.93	2.93	0.00
Polyvinylchloride	1.47	1.47	0.00
Polytetraflouroethane	1.47	1.47	0.00
Nylon	0.55	0.55	0.00
Material to adsorption kg/hr			**0.062**
Proportion of gold to adsorption%			**99.85**

Table C.5: SART overall performance analysis for removal of base metals in WPCB recycling campaign

SART performance analysis for removal of base metals Basis 400 Tonnes per annum				
Feed	Quantities in feed kg/hr	Metals after cyanidation kg/hr	Precipitated metals after SART kg/hr	Metals to adsorption circuit kg/hr
Fe	0.81	0.81	0.53	0.28
Cu	0.43	0.00	0.00	0.00
Al	0.86	0.86	0.00	0.77
Sn	2.29	2.29	1.49	0.80
Ni	0.35	0.23	0.08	0.04
Zn	0.04	0.04	0.00	0.04
Pb	0.03	0.03	0.02	0.01
Mn	0.00	0.00	0.00	0.00
Sb	0.00	0.00	0.00	0.00
Au	0.06	0.06	0.06	0.06
Ag	0.11	0.11	0.07	0.04
SiO_2	9.16	9.16	0.00	0.00
Al_2O_3	3.66	3.66	0.00	0.00
Alkali & Alkali earth oxides	3.66	3.66	0.00	0.00
Titanates and Mica	1.83	1.83	0.00	0.00
Polyethylene	6.04	6.04	0.00	0.00
Polypropylene	2.93	2.93	0.00	0.00
Polyesters	2.93	2.93	0.00	0.00
Epoxies	2.93	2.93	0.00	0.00
Polyvinylchloride	1.47	1.47	0.00	0.00
Polytetraflouroethane	1.47	1.47	0.00	0.00
Nylon	0.55	0.55	0.00	0.00
Material to adsorption kg/hr				**2.05**
Proportion of gold to adsorption%				**3.03**

D: Equipment flow sheets

Size reduction equipment process flow diagram

[Adapted from Rossouw (2015), De Waal (2018)]

WPCBs holding bin

WPCB crusher feed hopper

Hammer crusher

Screen

Oversize to recycle

Crushed WPCBs buffer hopper

Ball mill

To milled WPCBs buffer hopper in solder leachng plant
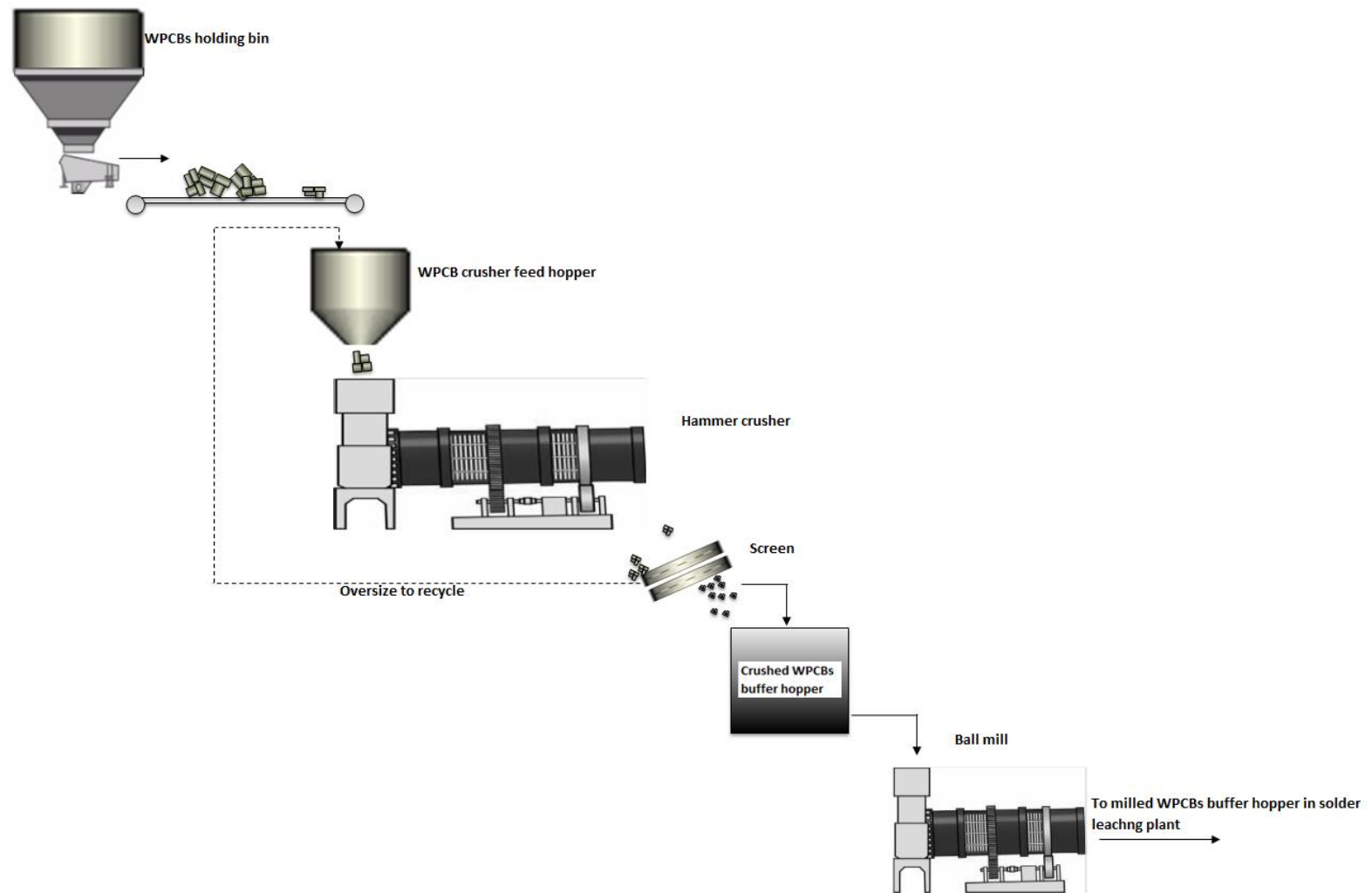
Figure D.1: Size reduction equipment process flow diagram
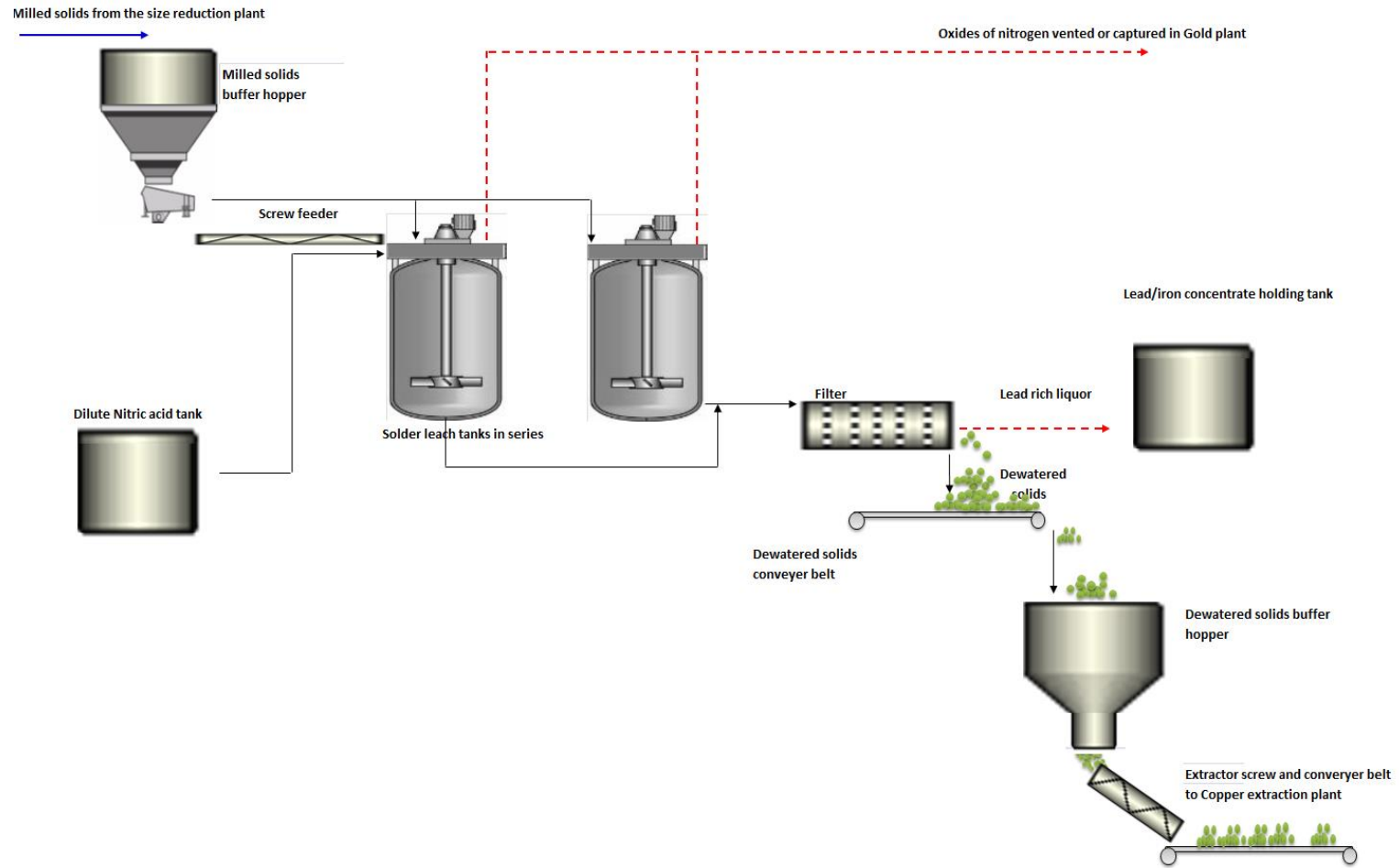
Figure D.2: Solder leaching equipment process flow diagram
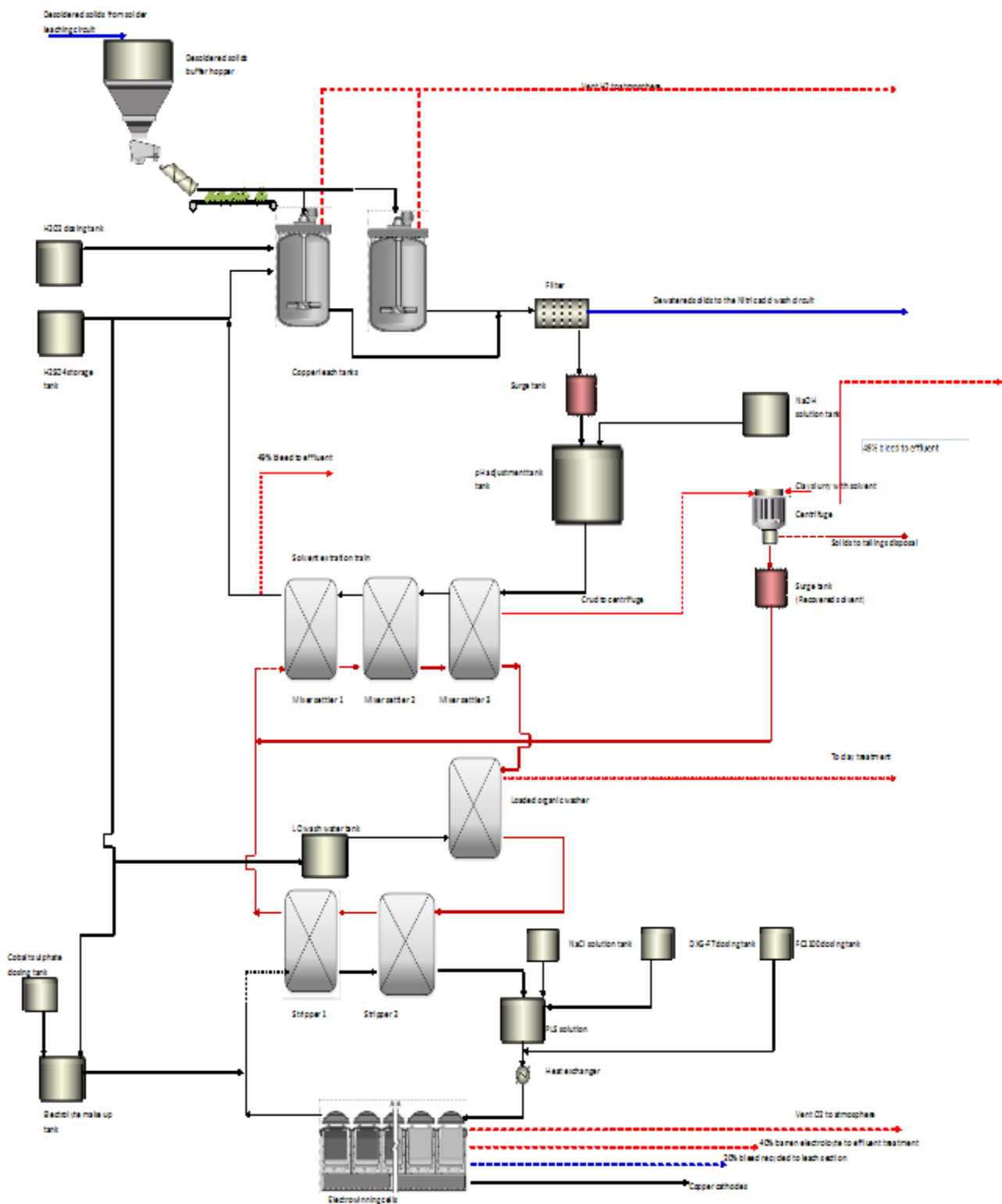
Figure D.3: Copper extraction equipment process flow diagram

Copper tracing
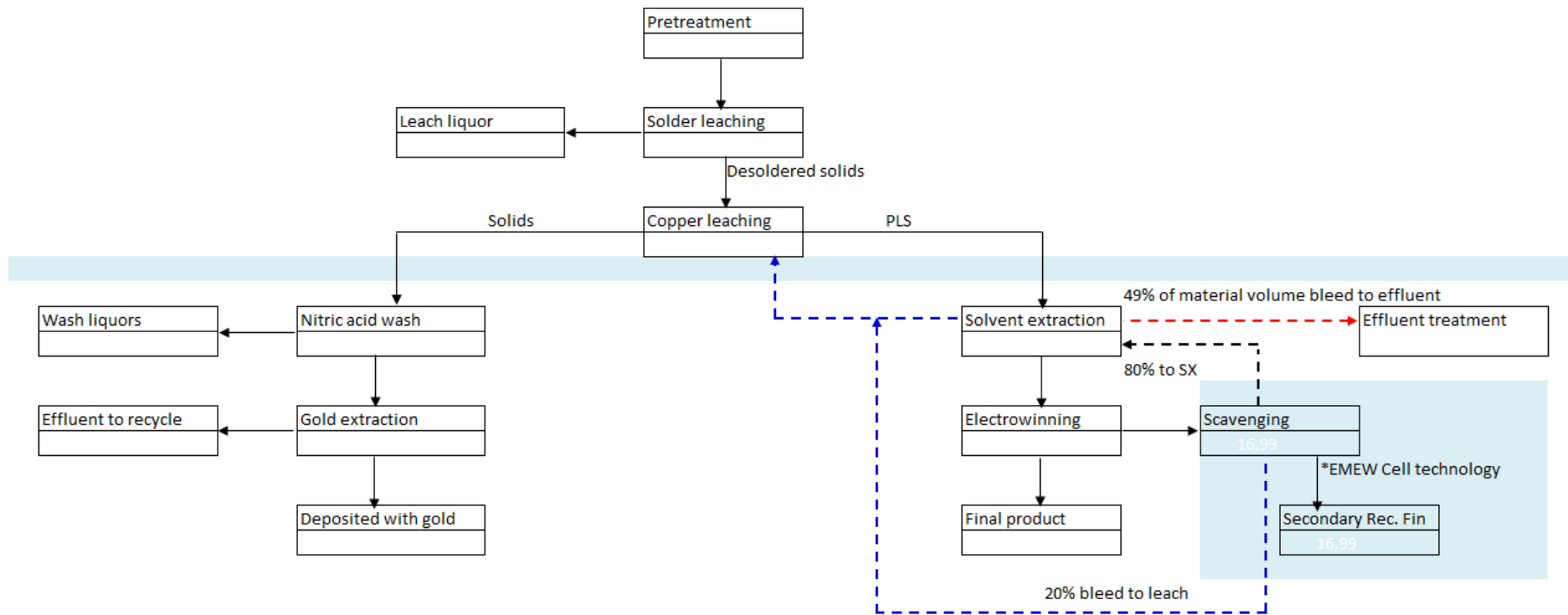
Figure D.4: Tracing the movement of copper throughout the entire scope of operations

Nitric Acid wash equipment process flow diagram
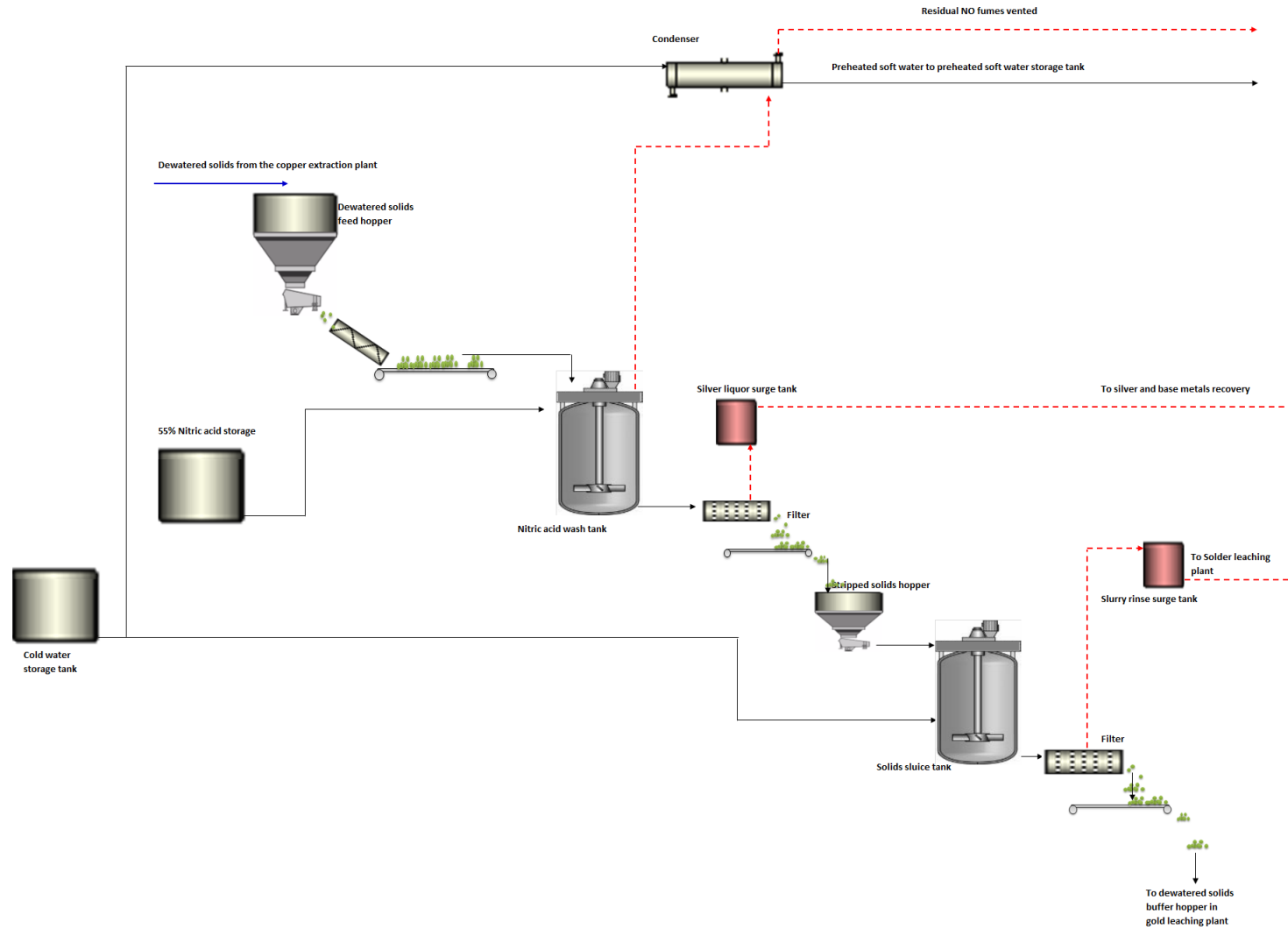
Figure D.5: Nitric acid wash equipment process flow diagram

Gold extraction equipment process flow diagram
Cyanide leaching section
[Adapted from Rossouw (2015), De Waal (2018)]
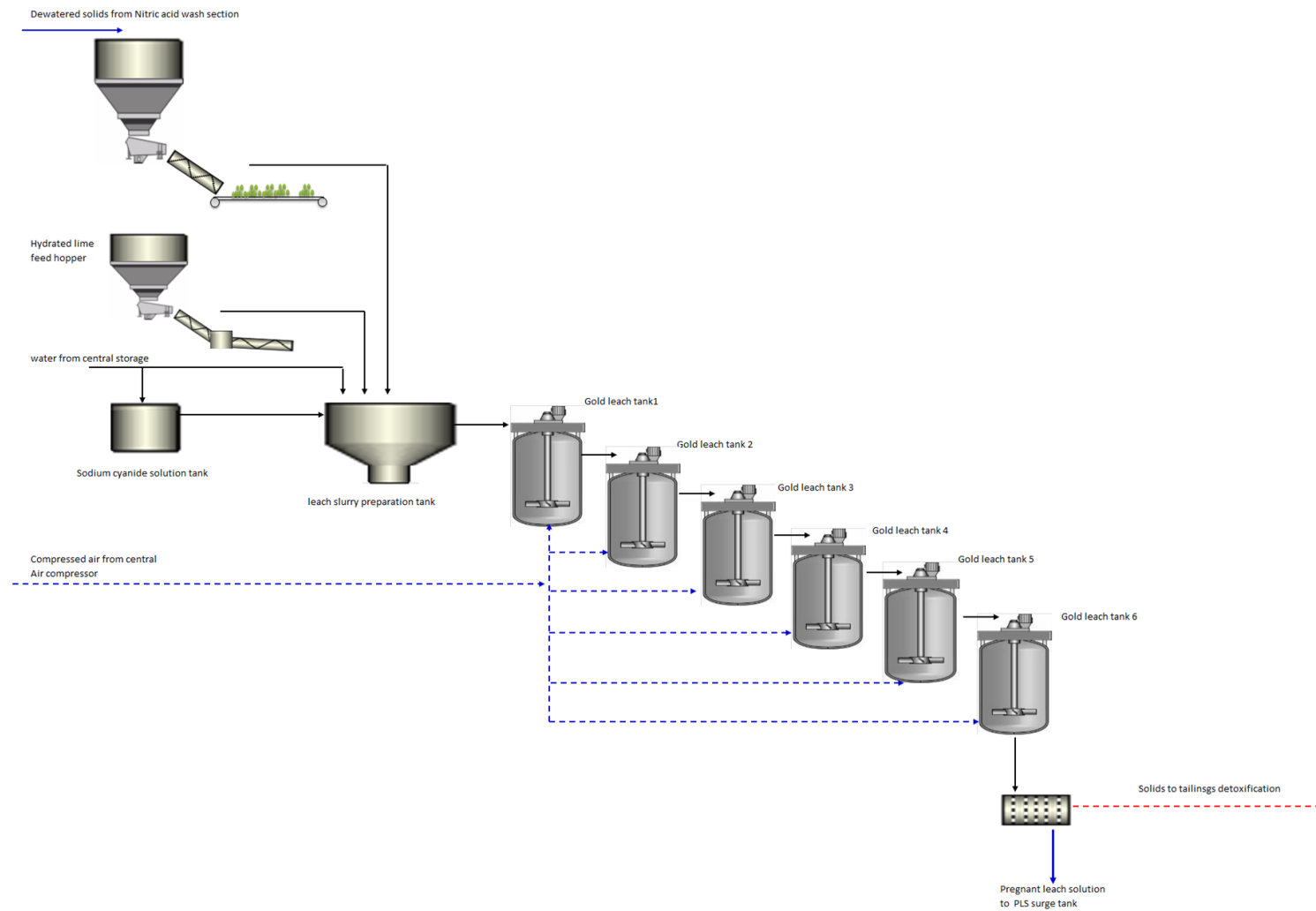
Figure D.6: Gold extraction equipment process flow diagram (Cyanide leaching section)

Gold extraction equipment process flow diagram
Adsorption and carbon regeneration sections
(Adapted from Rogans (2012), Rossouw (2015), De Waal (2018))
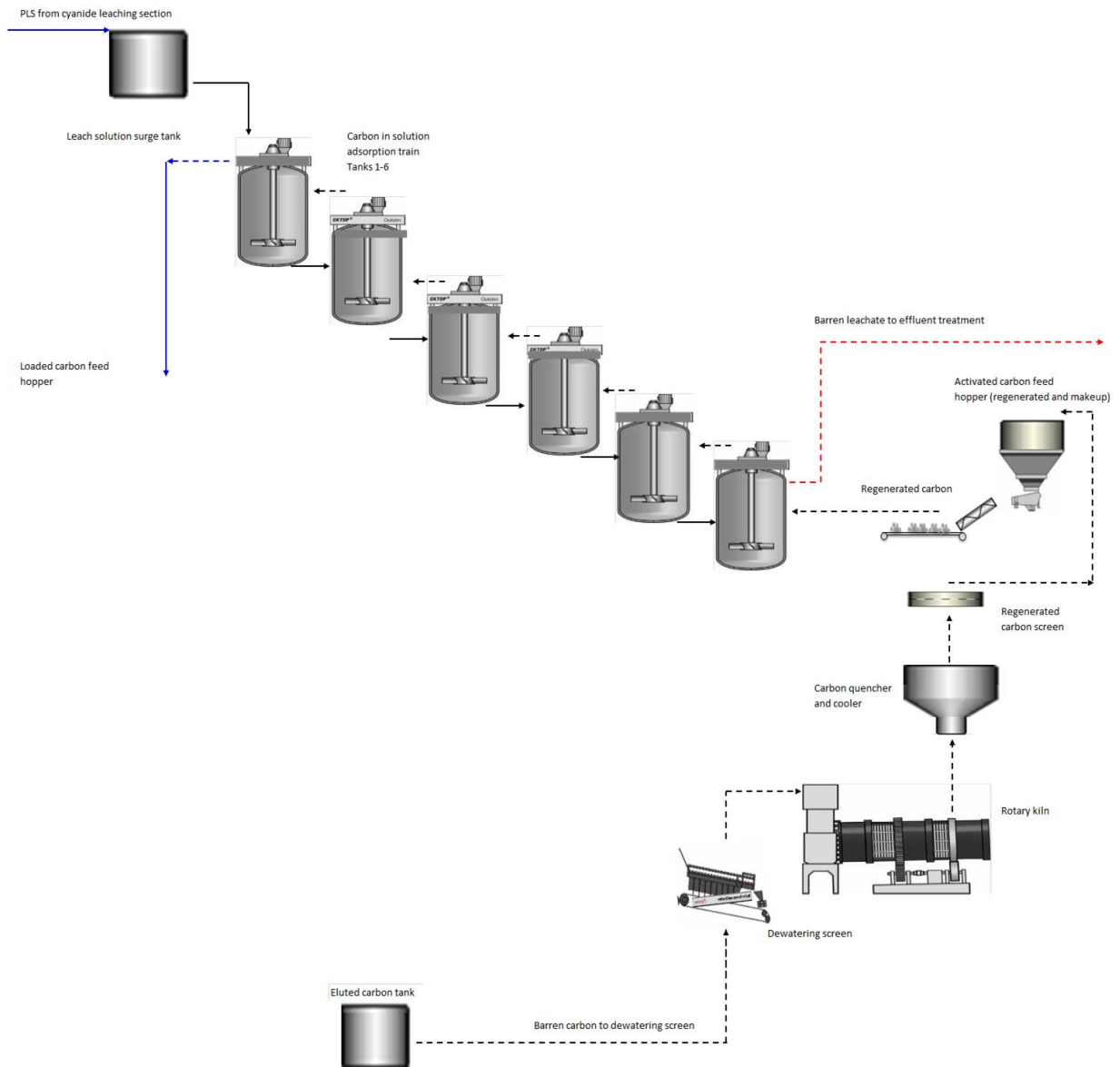
Figure D.7: Gold extraction equipment process flow diagram (Adsorption and carbon regeneration sections)

Gold extraction equipment process flow diagram
Acid wash, elution and electrowinning sections
(Adapted from Rogans (2012), Rossouw (2015), De Waal (2018))
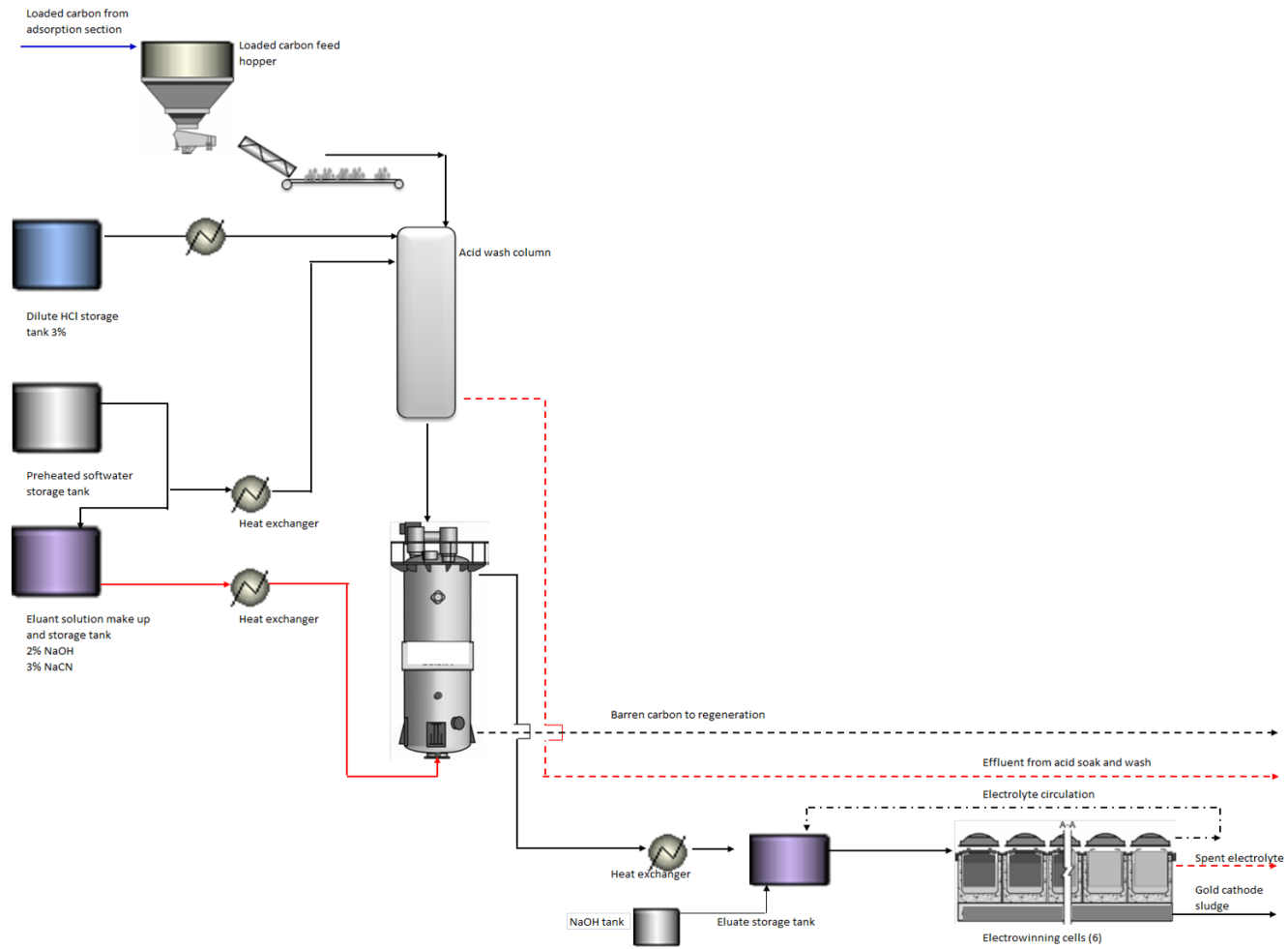
Figure D.8: Acid wash, elution and electrowinning sections

Gold tracing
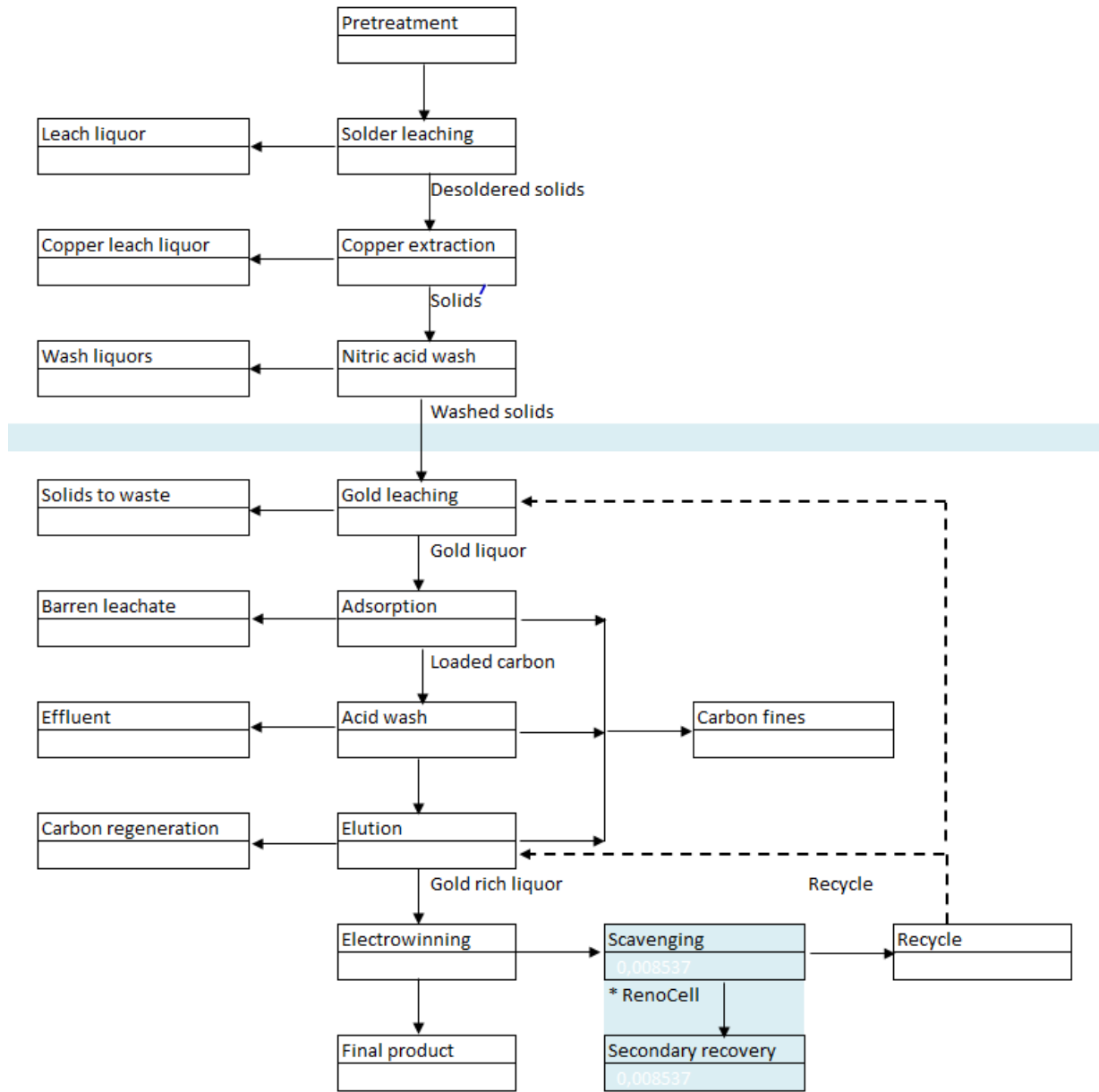
Figure D.9: Tracing the movement of gold throughout the entire scope of operations

E: Calculations for the initial evaluation of the business models

Table E.1: Collection and dismantling materials requirement and feasibility analysis

Collection and dismantling			
Storage space requirements			40% of design
Plant availability	Days	312	312
Rated operating capacity	Tonnes/yr	10 000	4 000
Proportion of PCBs in raw e-waste (Widmer *et al.,* (2005))	%	2	1
Raw e-waste requirement	Tonnes/yr	588 235	235 294
Bulk density of raw e-waste (SITA excel spread sheet calculator)	kg/m3	210	210
Material requirements per 30 days of operation	Tonnes	56 561	22 624
Storage space requirements per 30 days (for h= 4m)	m2	67 335	26 934
Manpower			
Dismantling rate per individual (Blaser & Schluep (2012))	Tonnes/month	3.50	3.50
Dismantling rate per 14 days per individual	Tonnes/fortnight	1.75	1.75
Manpower requirement to dismantle four weeks worth of material	Persons	16 160	6 464
Material supply in RSA			
Projected tonnages in South Africa yet to join recycling chain (Finlay and Lietchi (2008))	Tonnes	2 108 000	2 108 000
Projected annual growth rate (GreenCape (2017))	Tonnes/yr	322 000	322 000
Expected e-waste volumes as at December 2018 in RSA	Tonnes	5 006 000	5 006 000
Expected e-waste volumes as at December 2030 in RSA	Tonnes	12 090 000	12 090 000
By-product and waste projections			
By-product to be generated from e-waste processing (Widmer *et al.,* (2005)	Tonnes/month	55 600	22 240
Projected absorption capacity of by-products in South Africa (GreenCape (2017))*	%	15	15
Projected absorption capacity of by-products in South Africa (GreenCape (2017))*	Tonnes/month	8 340	3 336
Projected inventory build up of by-products from dismantling operations	Tonnes/month	47 260	18 904
Projected inventory build up of by-products from dismantling operations	Tonnes/yr	567 115	226 846
Projected inventory build up of by-products from dismantling operations at the end of the project life	Tonnes	11 342 308	4 536 923
Land space required for managing unprocessed by product inventory (pile heights of 4m)	m2/month	56 261	22 505
Land space required for managing unprocessed by product inventory (pile heights of 4m) at the end of the project life	m2	13 502 747	5 401 099
Potential revenue sources and streams from by-products			
Potential plastics for pyrolysis, export or composite hard material processing generated (Widmer *et al.,* (2005))	Tonnes/month	7 183	2 873
Potential metals for recycling (Widmer *et al.,* (2005))	Tonnes/month	28 450	11 380
Potential cables for recycling (Widmer *et al.* (2005))	Tonnes/month	945	378
Metals and plastics mixture for further processing (Widmer *et al.,* (2005))	Tonnes/month	2363	945
Potential source of Indium for further processing (Widmer *et al.,* (2005))	Tonnes/month	5 624	2 250

Table E.2: Annual production plan during normal operations for copper and gold metal production (models 3a & 4a)

WPCB hydrometallurgical plant																				
Annual production plan			Production summary																	
			Annual Totals (tonnes)	**3a & 4a**	3b & 4b	3c & 4c	3d & 4d													
			Gold sludge sales	0.113	0.00	0.113	0.00													
			Gold liquor sales	0.00	761.07	0.00	761.07													
			Copper LME Grade A	89.54	0.00	0.00	72.72													
			CuSO4 liquor (3.71%)	0.00	2386.44	2386.44	0.00													
			Month	Jan	Feb	Mar	April	May	June	July	Aug	Sep	Oct	Nov	Dec	Plan	BUDGET	Actual
			Days	30	28	31	30	31	30	31	31	30	31	30	31	312	312	312
			Remark	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	Normal	Normal	Year 1
Business	Gold	Gold sludge sales	tonnes	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
Models	sludge	Rand Refinery	tonnes	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
3a & 4a	>98%	Opening stocks	tonnes	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Production	tonnes	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
		Shut down	days	0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%
	Copper	Copper cathode sales	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
	LME	Trade Key.com	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
	grade A	Opening Stocks	tonnes	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	>99%	Production	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
		Shut down	days	0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%

Table E.3: Annual production plan during normal operations for copper sulphate and gold liquor production (models 3b & 4b)

WPCB hydrometallurgical plant
Annual production plan

Production summary

Annual Totals (tonnes)	3a & 4a	3b & 4b	3c & 4c	3d & 4d
Gold sludge sales	0.113	0.00	0.113	0.00
Gold liquor sales	0.00	801.04	0.00	761.07
Copper LME Grade A	72.72	0.00	0.00	72.72
CuSO4 liquor (3.71%)	0.00	2386.44	2386.44	0.00

				Month	Jan	Feb	Mar	April	May	June	July	Aug	Sep	Oct	Nov	Dec	Plan	BUDGET	Actual
				Days	30	28	31	30	31	30	31	31	30	31	30	31	312	312	312
				Remark	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	Normal	Normal	Year 1
Business	Gold	Gold liquor sales	tonnes		77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
Model	liquor	SA Precious metals	tonnes		77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
3b & 4b	142.52ppm	Opening stocks	tonnes		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Production	tonnes		77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
		Shut down	days		0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%
	Copper	Copper liquor sales	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
	liquor	Trade Key.com/SA Prec. Met.	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
	3.70%	Opening Stocks	tonnes		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Production	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
		Shut down	days		0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%

Table E.4: Annual production plan during normal operations for copper sulphate liquor and gold sludge production (models 3c & 4c)

WPCB hydrometallurgical plant
Annual production plan

Production summary					
Annual Totals (tonnes)		3a & 4a	3b & 4b	3c & 4c	3d & 4d
Gold sludge sales		0.113	0.00	0.113	0.00
Gold liquor sales		0.00	761.07	0.00	761.07
Copper LME Grade A		72.72	0.00	0.00	72.72
CuSO4 liquor (3.71%)		0.00	2386.44	2386.44	0.00

				Month	Jan	Feb	Mar	April	May	June	July	Aug	Sep	Oct	Nov	Dec	Plan	BUDGET	Actual
				Days	30	28	31	30	31	30	31	31	30	31	30	31	312	312	312
				Remark	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	Normal	Normal	Year 1
Business	Gold	Gold sludge sales	tonnes		0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
Model	sludge	Rand Refinery	tonnes		0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
3c & 4c	>98%	Opening stocks	tonnes		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Production	tonnes		0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.003	0.00	0.113	0.113	0.113
		Shut down	days		0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%
	Copper	Copper liquor sales	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
	liquor	Trade Key.com/SA Prec. Met.	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
	3.70%	Opening Stocks	tonnes		0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00	0,00
		Production	tonnes		229.47	214.17	237.11	229.47	237.11	229.47	237.11	237.11	229.47	237.11	68.84	0.00	2386.44	2386.44	2386.44
		Shut down	days		0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%		100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%

Table E.5: Annual production plan during normal operations for copper cathodes and gold liquor production (models 3d & 4d)

WPCB hydrometallurgical plant

Annual production plan

Production summary

Annual Totals (tonnes)	3a & 4a	3b & 4b	3c & 4c	3d & 4d
Gold sludge sales	0.113	0.00	0.113	0.00
Gold liquor sales	0.00	761.07	0.00	801.04
Copper LME Grade A	72.72	0.00	0.00	89.54
CuSO4 liquor (3.71%)	0.00	2386.44	2386.44	0.00

			Month	Jan	Feb	Mar	April	May	June	July	Aug	Sep	Oct	Nov	Dec	Plan	BUDGET	Actual
			Days	30	28	31	30	31	30	31	31	30	31	30	31	312	312	312
			Remark	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	plan	Normal	Normal	Year 1
Business	Gold	Gold liquor sales	tonnes	77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
Model	liquor	SA Precious metals	tonnes	77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
3d & 4d	142.52ppm	Opening stocks	tonnes	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
		Production	tonnes	77.02	71.89	79.59	77.02	79.59	77.02	79.59	79.59	77.02	79.59	23.11	0.00	801.04	801.04	801.04
		Shut down	days	0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%
	Copper	Copper cathode sales	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
	LME	Trade Key.com	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
	grade A	Opening Stocks	tonnes	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	>99%	Production	tonnes	8.61	8.04	8.90	8.61	8.90	8.61	8.90	8.90	8.61	8.90	2.58	0.00	89.54	89.54	89.54
		Shut down	days	0	0	0	0	0	0	0	0	0	0	21	31	52	52	52
		Plant availability	TU%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	30%	0%	85.71%	85.71%	85.71%

F: Calculations for economic evaluations

Table F.1: Water softener sizing and operational data sheet (Soft water requirements and process water chemistry)

Operating basis for process 400 Tonnes/year	Units	Quantity Original worked with 12,5% excess capacity	Quantity 92% availability with 400Tonnes/annum capacity	Reference data source
Soft water requirements (2 elution cycles)	m3/day	11.92	11.33	Updated mass balances,de Waal (2018))
Operating time for softener	Hours/day	22.33	21.24	DOW (1999), Commissioning data (Msuku (2012)
Resin regeneration time	Hours/day	1.67	1.59	DOW (1999), Commissioning data (Msuku (2012)
Softener tank capacity	m3	0.16	0.15	Updated mass balances (de Waal (2018))
Flow through velocity	m3/hour	0,53	0.51	Calculated
Resin loading	m3	0.05	0.05	Calculated
Resin bed voidage	m3	0.02	0.02	Ketrick (2014)
Volume for water flow	m3	0.11	0.10	Calculated
Flow through volume in softener	m3	0.13	0.12	Calculated
Recommended flow in softener lower limit	L/min m3	66.87	63.60	Technical data for softener operation Ketrick (2014)
Recommended flow in softener lower limit	L/min m3	1337.46	1272.06	Technical data for softener operation Ketrick (2014)
Softener flow for the process	L/min m3	68.41	65.07	Calculated. Resin bed will expand within normal limits of 67% of softener (Ketrick(2014))
Brine tank capacity	m3	0.59	0.56	Manjengwa (2014)
Process water chemistry				
Feed water for softening	m3/day	11.92	11.33	Local municipality (Mohotsi (2016))
Regeneration water	m3/day	1.26	1.20	
Backwash water	m3/day	0.09	0.08	DOW (1999), Commissioning data (Msuku (2012)
Brine injection and slow rinse	m3/day	0.53	0.51	DOW (1999), Commissioning data (Msuku (2012)
Rapid rinse	m3/day	0.11	0.10	DOW (1999), Commissioning data (Msuku (2012)
Brine refill	m3/day	0.53	0.51	DOW (1999), Commissioning data (Msuku (2012)
Daily water usage for running the water softener	m3/day	13.18	12.53	DOW (1999), Commissioning data (Msuku (2012)
Effluent water from regeneration	m3/day	0.73	0.69	DOW (1999), Commissioning data (Msuku (2012)
Calcium ion concentration	mg/L	17.89	17.02	Local municipality (Mohotsi (2016))
Magnesium ion concentration	mg/L	7.31	6.95	Local municipality (Mohotsi (2016))
Ca 2+ conversion factor	units	2.50	2.38	Ketrick (2014), Purolite (2014)
Mg 2+ conversion factor	units	4.18	3.98	Ketrick (2014), Purolite (2014)
Total hardness as CaCO3	mg/L	75.29	71.61	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Equivalent moles of CaCO3	moles/L	0.00	0.00	
Equivalent moles of CaCO3	moles/day	8.96	8.52	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Moles of sodium ions loaded on resin for softening	moles/day	17.93	17.05	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Moles of salt required for regenerating resin	moles/day	26.89	25.57	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Mass of salt required for regenerating resin	kgs/day	1.57	1.49	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Moles of salt required for making up new brine solution	moles/day	26.89	25.57	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Mass of salt for making up new brine solution	moles/day	1.57	1.49	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
Daily salt usage for running the water softener	kgs/day	3.14	2.99	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))

Table F.2: Water softening sizing and operational data sheet (Effluent generation and costing)

Operating basis for process 400 Tonnes/year				
	Units	Quantity Original worked with 12,5% excess capacity	Quantity 92% availability with 400Tonnes/annum capacity	Reference data source
Effluent generation streams rates				
Backwash water	m3/day	0.09	0.08	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Brine injection and slow rinse	m3/day	0.53	0.51	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Rapid rinse	m3/day	0.11	0.10	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Total water flow	m3/day	0.73	0.69	
Excess salt to effluent	kg/day	0.52	0.50	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Excess salt to effluent	mg/day	523777.37	498164.66	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Concentration of sodium chloride in effluent	mg/L	718.18	683.06	
Ca 2+ stripped from resin	mg/day	213181.00	202756.45	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Ca 2+ in regeneration streams	mg/day	13047.35	12409.34	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Total calcium to effluent	mg/day	226228.35	215165.78	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Concentration of calcium in effluent	mg/L	310.20	295.03	
Mg 2+ stripped from resin	mg/day	87131.33	82870.61	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Mg 2+ in regeneration streams	mg/day	5332.71	5071.94	Calculated (Msuku, (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Total magnesium in effluent	mg/day	92464.04	87942.55	Calculated (Msuku (2012), Ketrick (2014), Purolite (2014), Mohotsi (2016))
Concentration of magnesium in effluent	mg/L	126.78	120.58	
TDS in process water ex municipality	mg/L	146.41	139.25	Mohotsi (2016)
TDS in process water ex municipality in effluent stream	mg/day	106778.24	101556.79	Calculated (Ketrick (2014), Purolite (2014), Mohotsi (2016))
TDS in process water ex municipality in effluent stream	mg/L	146.41	139.25	
TDS exiting the softener regeneration process	**mg/L**	**1301.57**	1237.92	
Costing of softened water				
Element	**Unit**	**Unit cost**	**Usage**	
Water usage	m3/day	2.63	13,18	
Labour (basic with transport, 7 day a week minimum wage)	8 hours/shift	25.04	1.00	
Power	KWh/day	30.86	26.40	
Salt	kg/day	0.32	3.21	
Soft water produced	**m3/day**	**24.36**	**11.92**	

Table F.3: Key product output analysis for all the plants business models 3a and 4a

* crushed WPCBs were outsourced in the case of 4a

Business models 3a & 4a Material/product	Hourly production (Tonnes/hr)	Daily production (Tonnes/day)	Monthly production (Tonnes/month)	Annual production (Tonnes/annum)
Crushing and milling or outsource				
WPCBs	0.06	1.47	43.98	457.44
Solder leaching				
Desoldered solids	0.06	1.33	39.79	413.79
Solder leach	0.96	23.16	694.71	7225.02
Copper Extraction				
Filtered solids	0.04	1.00	29.98	311.75
Copper Liquor	0.32	7.65	229.47	2386.44
Copper (cathode sheets)	0.01	0.29	8.62	89.66
Nitric acid wash				
Stripped solids	0.04	1.04	31.18	324.23
Silver rich liquor	0.20	4.78	143.44	1491.76
Gold extraction				
Gold liquor	0.10	2.44	73.18	761.07
Gold sludge	0.00002	0.0004	0.0109	0.1131
Non-metallic solids	0.04	0.94	28.34	294.71

Table F.4: Key product output analysis for all the plants business models 3b and 4b

* crushed WPCBs were outsourced in the case of 4b

Business models 3b & 4b Material/product	Hourly production (Tonnes/hr)	Daily production (Tonnes/day)	Monthly production (Tonnes/month)	Annual production (Tonnes/annum)
Crushing and milling/outsourced				
WPCBs	0.06	1.47	43.98	457.44
Solder leaching				
Desoldered solids	0.06	1.33	39.79	413.79
Solder leach	0.96	23.16	694.71	7225.02
Copper Extraction				
Filtered solids	0.04	1.00	29.98	311.75
Copper Liquor	0.32	7.65	229.47	2386.44
Nitric acid wash				
Stripped solids	0.04	1.04	31.18	324.23
Silver rich liquor	0.20	4.78	143.44	1491.76
Gold extraction				
Gold liquor	0.11	2.57	77.02	801.04
Non-metallic solids	0.00	0.0003	0.010	0.105

Table F.5: Key product output analysis for all the plants business models 3c and 4c

* crushed WPCBs were outsourced in the case of 4c

Business models 3c & 4c Material/product	Hourly production (Tonnes/hr)	Daily production (Tonnes/day)	Monthly production (Tonnes/month)	Annual production (Tonnes/annum)
Crushing and milling/outsourced				
WPCBs	0.06	1.47	43.98	457.44
Solder leaching				
Desoldered solids	0.06	1.33	39.79	413.79
Solder leach	0.96	23.16	694.71	7225.02
Copper Extraction				
Filtered solids	0.04	1.00	29.98	311.75
Copper Liquor	0.32	7.65	229.47	2386.44
Nitric acid wash				
Stripped solids	0.04	1.04	31.18	324.23
Silver rich liquor	0.20	4.78	143.44	1491.76
Gold extraction				
Gold liquor	0.10	2.44	73.18	761.07
Gold sludge	0.00002	0.0004	0.0109	0.1131
Non-metallic solids	0.04	0.94	28.34	294.71

Table F.6: Key product output analysis for all the plants business models 3d and 4d

* crushed WPCBs were outsourced in the case of 4d

Business model 3d & 4d Material/product	Hourly production (Tonnes/hr)	Daily production (Tonnes/day)	Monthly production (Tonnes/month)	Annual production (Tonnes/annum)
Crushing and milling/outsourced				
WPCBs	0.06	1.47	43.98	457.44
Solder leaching				
Desoldered solids	0.06	1.33	39.79	413.79
Solder leach	0.96	23.16	694.71	7225.02
Copper Extraction				
Filtered solids	0.04	1.00	29.98	311.75
Copper Liquor	0.32	7.65	229.47	2386.44
Copper (cathode sheets)	0.01	0.29	8.62	89.66
Nitric acid wash				
Stripped solids	0.04	1.04	31.18	324.23
Silver rich liquor	0.20	4.78	143.44	1491.76
Gold extraction				
Gold liquor	0.11	2.57	77.02	801.04
Non-metallic solids	0.00	0.0003	0.010	0.105

Table F.7: Generic main equipment schedule

Guide and remarks	Equipment	Number	Capacity	Material of construction	Residence time (hours)	Temperature (Deg C)	Pressure (Kpa)
	Size reduction						
1 week x1,1 actual	WPCBs holding bin	1	9,38m3	Rigid PVC 10mm thick	168	25,15	101,325
1 shift	WPCBs crusher feedhopper	1	0,084m3	Mild steel	24	25,15	101,325
1 day	Hammer crusher	1	0,067tes/hr	MSS (250*360 mm internal chamber)	Continuous flow through	25,15	101,325
1 day	Screen (1.3 times crushing rate)	1	0,087tes/hr	Mild steel (2 deck, 1350*700*1250mm)	Continuous flow through	25,15	101,325
1 day	Crushed WPCBs buffer hopper	1	0,252m3	Mild steel	24	25,15	101,325
1 day	Ball mill	1	0,067tes/hr	MSS (1500x1000x1200)	Continuous flow through	25,15	101,325
	Solder leaching						
1 day x1,1 actual	Milled solids buffer hopper	1	0,94m3	Rigid PVC 10mm thick	24	25,15	101,325
x1,1 actual I day storage	Pregnant leach tank	1	24,78m3	Mild steel	24	33	101,325
holds material for a batch	Solder leach tanks	2	7,53m3	SS316	8	33	101,325
I day's operation 1,1 x actual	Pressure filter	1	49,57m3/hr	Mild steel	0,5	33	1600
	Copper extraction						
x 1.1 actual for 7 days	Dewatered solids buffer hopper	1	6,35m3	Mild steel	12	25,15	101,325
7 days x 1,1 actual	Hydrogen peroxide dosing tank	1	2,74m3	SS316	168 (Continuous flow through)	25,15	101,325
For 7 days	Sulphuric storage	1	4,60m3	SS316	168 (Continuous flow through)	25,15	101,325
1 week	Sodium chloride dosing tank	1	0,03m3	PVC	168 (Continuous flow through)		
One month	Cobalt sulphate dosing tank	1	0,03m3	PVC	168 (Continuous flow through)	25,15	101,325
1 week	FC-1100 dosing tank	1	0,03m3	PVC	168 (Continuous flow through)		
1 week	DXG solution tank	1	0,033m3	PVC	168 (Continuous flow through)	25,15	101,325
1 week	Electrolyte make up tank	1	27,66m3	SS316	168	25,15	101,325
1 week	Copper leach tank	2	27,79m3	SS904	8	40	101,325
1 week of material	Pressure filter	1	49,57m3/hr	SS316	1 FP	40	1600
1 week	Leach filtrate surge tank	1	2m3	SS904	1 FP	25-40	101,325
x 1.1 actual	Sodium hydroxide solution tank	1	0,703m3	Rigid PVC 10mm thick	12	25,15	101,325
1 week of material x 1,1 actual	Pregnant leach solution tank	1	52,81m3	SS316	12 FP	25,15	101,325
	Mixer settler	3	7,50m3	FRP composite	0,583333333	<40	101,325-400
Updated	Loaded organic solvent washer	1	7,50m3	FRP composite	0,583333333	<40	101,325-400
	Organic stripper	2	7,50m3	FRP composite	0,583333333	<40	101,325-400
x 1.1 actual	Electrolyte circulation tank	1	119,64m3	SS316	7,598	40	101,325
x 1.1 actual	Heat exchanger	1	3.82KW	SS316	FP	40	101,325
x 1.1 actual to 4 sig figures	Centrifuge	1	0,002t/hr	Mild steel (Dia 30cm)	FP	25,15	101,325
for handling 7 days of production, operates once a week	Cathode stripping machine	1	1 Cathode/hour	Stainless steel 316/38/220V 50Hz (manual)	1	25-143	101,325-
actual (with anode spacers)	Electrowinning cells	1	5,2t/hr	a-Sn/SS 304 anode and cathode, Glass lined mild	168	30-45	101,325
	Nitric acid wash						
7 days from copper plant	Dewatered solids feed/buffer hopper	1	4,48m3	Mild steel	24/FP	25,15	101,325
2 weeks x1,1 actual	55% nitric acid storage tank (central)	1	20,87m3	SS316	336 FP	25,15	101,325
1,1 x actual for 4 cycles each day. Sized to handle 1 day of mater	Nitric acid wash tank	1	1,72m3	SS316	5	100	101,325
x1,1 actual. Rated as tonnage of saturated steam condensed.	Condenser	1	19,51KW (54,90m3/hr)	SS316	5	35-100	101,325
3,79% vol of acid wash tank	Silver liquor surge tank	1	0,07m3	Mild steel	0,5-1	80-90	101,325
1,1 x actual for 4 cycles each day. Sized to handle 1 day of mater	Pressure filter	1	10,89m3/hr	SS316	0,25	90-100	1600
1,1 x actual for 4 cycles each day. Sized to handle 1 day of mater	Pressure filter	1	10,89m3/hr	Mild steel	0,25	25-35	1600
1,1 x actual. 7 days material	Stripped solids hopper	1	4,74m3	Mild steel	24 FP	25,15	101,325
1,1 x actual for 4 cycles each day. Sized to handle 1 day of mater	Solids sluice tank	1	1,67m3	Mild steel	1	25,15	101,325
3,79% vol of sluice tank	Slurry wash water surge tank	1	0,07m3	Mild steel	0,5-1	25,15	101,325
	Gold extraction						
1 day 1,1 actual	Dewatered solids buffer hopper	1	0,68m3	Polymer concrete (Epoxy resin 58%)	24 FP	25,15	101,325
1 day x 1,1 actual	Hydrated lime feed hopper	1	0,867m3	Polymer concrete (Epoxy resin 58%)	24 FP	25,15	101,325
1 day x 1,1 actual	Sodium cyanide dosing tank	1	8,17m3	SS316	2FP	25,15	101,325
1 day x 1,1 actual	Sodium cyanide dosing tank (Elution)	1	7,95m3	SS316	2FP	25,15	101,325
I cycle for 24 hours of material flow	Leach slurry conditioning tank	1	11,19m3	SS316	0,5	25,15	101,325
x1,1 actualx 1,104 to absorb lag time	Gold leach tanks	6	2,21m3	SS316	24	25,15	101,325
1 day's material	Pressure filter	1	10,89m3/hr	SS316	1	25,15	1600
x 1.1 holds 1 leaching campaign	Gold leach surge tank	1	10,67m3	SS316	12 maximum	25,15	101,325
x1.1 handles 12hrs worth of leachate	Adsorption tanks	6	0,912m3	Mild steel	1 hour in each, 11 hours total flow per batch	25,15	101,325
For running 2 batches x1,1 of actual	Dilute hydrochloric acid tank 3%	1	3,98m3	Hastelloy C	FP	25,15	101,325
A day's production x1,1	Preheated soft water tank	1	14,4m3	Mild steel	FP	90	101,325
A day's production x1,1	Eluant solution (make up & storage)	1	7,96m3	SS316	FP	25,15	101,325
1-2 times larger than elution x1,5 elution column (Rogans 2012)	Loaded carbon feed hopper	1	6,90m3	Mild steel	12	25,15	101,325
x 1,1 actual, for running 2 batches	Acid wash column	1	3,98m3	Hastelloy C	3	25,15	101,325
x 1,1 actual, for running 2 batches each day	Elution column	1	4,60m3	Hastelloy C	6,5	110-115	101,325-230
x 1,1 actual for handling 14 batches of material (week)	NaOH pH correction tank	1	0,59m3	Rigid PVC 10mm thick or composite	168 (Continuous flow through)	25,15	101,325
x 1,1 actual (2 bed vols)	Weak eluant recycle tank	1	3,98m3	SS316	26,5	90	101,325
x1,1 actual	Heat exchanger (electrowinning)	1	4,544KW	SS316 (Shell and tube)	24 FP	90	101,325
x 1,1 actual	Heat exchanger (eluant solution)	1	4,544KW	Mild steel (plate)	24 FP	90	101,325
x 1,1 actual	Heat exchanger (soft water)	1	4,544KW	Hastelloy C (shell and tube)	24 FP	90	101,325
x1,1 actual	Heat exchanger (dilute hydrochloric acid)	1	4,544KW	Reinforced PVC	24 FP	90	101,325
Handles 2 batches each day	Eluate storage tank/circulation (electrowinning)	1	7,96m3	SS316	5	90	101,325
x1,1 actual	Electrowinning cells	6	1,21 tes/hr	Mild steel	5	25,15	101,325
x1,1 actual	Eluted carbon tank	1	1,065m3	Mild steel	0,25	650-750	101,325
x 1,1 actual based on carbon per 12hr batch	Rotary kiln	1	1.92 tes/hr	Mild steel	0,25	25	101,325
x1,1 actual	Barren carbon dewatering screen (single deck)	1	1,32 tes/hr (1m2)	Mild steel	0,33	25	101,325
x1,3 actual	Carbon quencher/cooler	1	(2,31 tes/hr) 1,26m3	Mild steel	0,42	25	101,325
x1,1 actual	Regeneration carbon screen (double deck)	1	1,32 tes/hr (1m2)	Mild steel	0,33	25	101,325
x1,1 actual, sized to handle two batches of carbon	Activated carbon feed hopper	1	2,13m3	Mild steel	0,25	25-40	101,325
	Utilities						
7 days	Water softening unit	1	0,162m3	Mild steel	22,33 hrs operation 1,67 hours regeneration	25,15	253-405,3
x1,1 actual capacity	Brine tank	1	0,65m3	PVC	22,33 hrs	25,15	101,325
x 1.5 actual capacity	Air compressor	1	1,41m3/hr	MSS, Reciprocating, single stage	13 hrs full load, 11 hours 25% of full load	25	233,05
Recommended for the specified compressor output	Receiving cylinder	1	0,027m3	Mild steel	24 FP	35	350
	Scavenging and effluent management						
x 1,1 actual I day	Scavenging cells acid stream	1	2,43tes/hr	Equipment supplier specifications	24hrs continous electrowinning (EMEW cell technology)	25-40	101,325
x 1,1 actual I day	Scavenging cells basic gold stream	1	0,48 tes/hr	Equipment supplier specifications	24 hrs continous electrowinning (RenoCell technology)	25-40	101,325
x 1,1 actual I day	Blended effluent tank (acid stream)	1	58,26m3	SS316	24 hrs continous flow through	25-40	101,325
x 1,1 actual I day	Blended effluent tank (basic stream)	1	11,46m3	SS316	24 hrs continous flow through	25-40	101,325
x 1,1 actual per day	Barren effluent conditioning tank	1	69,72m3	Mild steel	24 hrs continous dilution and discharge or recirculation	25	101,325
x 1,1 actual I day	Polished effluent distribution tank (acid stream)	1	58,26m3	Mild steel	24 hrs continous or recirculation	25	101,325
x 1,1 actual I day	Polished effluent distribution tank (basic stream)	1	11,46m3	Mild steel	24 hrs continous or recirculation	25	101,325
1 week x1,1 actual	Spent WPCB solids hopper	1	6,78m3	Rigid PVC 10mm thick	168	25,15	101,325

Table F.8: Main equipment cost schedule for all the sections

Equipment	Quantity	Material of construction	Capacity	Reference capacity	Reference cost	Base year	Index	Factor	Estimated Unit cost	Reference data/ vendor
Main equipment list										
*FP - Feed to process, SS stainless steels series, MSS -manufacturer's specification sheet										
Size reduction										
WPCBs holding bin	1	Rigid PVC 10mm thick	9,38m3	1m3	$150,00	2010	550,8	0,6	$592,09	Improchem (Mukariri (2012)) Timmerhaus & Peters (1991)
WPCBs crusher feedhopper	1	Mild steel	0,084m3	1m3	$1000,00	2014	576,1	0,6	$985,07	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Hammer crusher	1	MSS (250*360 mm internal chamber)	0,067tes/hr	55tes/hr	$10 000,00	2018	567,5	0,6	$5 123,79	Henan Fineschina Machinery, Henan, China (Mainland)
Screen (1.3 times crushing rate)	1	Mild steel (2 deck, 1350*700*1250mm)	0,08tes/hr	0,6tes/hr (2,97 m2)	$20 800,00	2014	576,1	0,6	$10 656,71	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Crushed WPCBs buffer hopper	1	Mild steel	0,252m3	20,95m3	$13 300,00	2014	576,1	0,6	$923,53	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Ball mill	1	MSS (420mmx450mm) 911M PEPPGR426	0,067tes/hr	0,146 tes/hr	$25 000,00	2018	567,5	0,6	$15 666,54	Henan Deya Machinery Company Limited, Henan, China (Mainland)
Solder leaching										
Milled solids buffer hopper	1	Rigid PVC 10mm thick	0,94m3	1m3	$150,00	2010	550,8	0,6	$148,92	Improchem (Mukariri (2012)) Timmerhaus & Peters (1991)
Pregnant leach tank	1	Mild steel	24,78m3	75,71m3	$46 800,00	2014	576,1	0,6	$23 587,54	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Solder leach tanks	2	SS316	7,53m3	5m3	$11 500,00	2000	394,1	0,53	$20 573,33	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Pressure filter	1	Mild steel	(49,57m3/hr) 0,5 m2	18,58m2	$86 500,00	2014	576,1	0,55	$11 666,55	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991), Perry et al ., (1997)
Copper extraction										
Dewatered solids buffer hopper	1	Mild steel	6,35m3	20,95m3	$13 300,00	2014	576,1	0,6	$6 401,38	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Hydrogen peroxide dosing tank	1	SS316	2,74m3	5m3	$11 500,00	2000	394,1	0,53	$12 039,57	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Sulphuric storage	1	SS316	4,60m3	5m3	$11 500,00	2000	394,1	0,53	$15 844,00	Timmerhaus and Peters (1991), Turton et al., (2009), Van Wyk (2014)
Sodium chloride dosing tank	1	PVC	0,03m3	1m3	$150,00	2010	550,8	0,6	$18,85	Improchem (Mukariri (2012)), Timmerhaus & Peters (1991)
Cobalt sulphate dosing tank	1	PVC	0,03m3	1m3	$150,00	2010	550,8	0,6	$18,85	Improchem (Mukariri (2012)), Timmerhaus & Peters (1991)
FC-1100 dosing tank	1	PVC	0,03m3	1m3	$150,00	2010	550,8	0,6	$18,85	Improchem (Mukariri (2012)), Timmerhaus & Peters (1991)
DXG solution tank	1	PVC	0,033m3	1m3	$150,00	2010	550,8	0,6	$19,96	Improchem (Mukariri (2012)), Timmerhaus & Peters (1991)
Electrolyte make up tank	1	SS316	27,66m3	5m3	$11 500,00	2000	394,1	0,53	$41 000,09	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Copper leach tank	2	SS904	27,79m3	5m3	$11 500,00	2000	394,1	0,53	$41 102,11	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Pressure filter	1	SS316	(49,57m3/hr) 0,5 m2	18,58m2	$190 400,00	2014	576,1	0,55	$25 679,90	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991), Perry et al ., (1997)
Leach filtrate surge tank	1	SS904	2m3	5m3	$11 500,00	2000	394,1	0,53	$10 187,41	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Sodium hydroxide solution tank	1	Rigid PVC 10mm thick	0,703m3	1m3	$150,00	2010	550,8	0,6	$125,09	Improchem (Mukariri (2012)) Timmerhaus and Peters (1991)
Pregnant leach solution tank	1	SS316	52,81m3	5m3	$11 500,00	2000	394,1	0,53	$57 762,07	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Mixer settler	3	FRP composite	(7,50m3) 158 Tonnes	31 Tonnes	$75 000,00	2018	567,5	0,6	$28 466,92	Timmerhaus and Peters (1991)
Loaded organic solvent washer	1	FRP composite	(7,50m3) 158 Tonnes	31 Tonnes	$75 000,00	2018	535,3	0,6	$30 179,29	Timmerhaus and Peters (1991), Material properties http://web.mit.edu/course/3/3.11/www/modules/props.pdf 12/08/2018
Organic stripper	2	FRP composite	(7,50m3) 158 Tonnes	31 Tonnes	$75 000,00	2018	541,7	0,6	$29 822,74	Timmerhaus and Peters (1991), Material properties http://web.mit.edu/course/3/3.11/www/modules/props.pdf 12/08/2018
Electrolyte circulation tank	1	SS316	119,64m3	5m3	$11 500,00	2000	394,1	0,53	$89 099,99	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Heat exchanger	1	SS316	3,82KW	1500KW	$52 451,00	2001	394,3	0,81	$598,04	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Centrifuge	1	Mild steel (Dia 30cm)	(0,002t/hr) Dia 30cm	60,96	$14 300,00	2014	576,1	0,6	$13 133,74	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Cathode stripping machine	1	Stainless steel 316/38/220V 50Hz (manual)	3 Cathodes/hour	200 Cathodes/hour	$4 502 500,00	2018	567,5	0,6	$362 333,49	Unisam, Hunan, China (Mainland)
Electrowinning cells	1	Pb-Ca-Sn/SS 304 anode & cathode, Glass lined mild steel	5,21/hr (5,84m3)	5,84m3	$90 000,00	2014	576,1	0,6	$89 516,48	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Nitric acid wash										
Dewatered solids feed/buffer hopper	1	Mild steel	4,48m3	20,95m3	$13 300,00	2014	576,1	0,6	$5 192,49	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
55%nitric acid storage tank (central)	1	SS316	20,87m3	5m3	$11 500,00	2000	394,1	0,53	$35 314,26	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Nitric acid wash tank	1	SS316	1,72m3	5m3	$11 500,00	2000	394,1	0,53	$9 406,61	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Condenser	1	SS316	19,51KW (54,90m3/hr)	1500KW	$52 451,00	2001	394,3	0,59	$5 824,39	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Silver liquor surge tank	1	Mild steel	0,07m3	75,71m3	$46 800,00	2014	576,1	0,6	$697,08	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Pressure filter	1	SS316	10,89m3/hr (0,33m2)	18,58m2	$190 400,00	2014	576,1	0,55	$5 641,60	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991), Perry et al ., (1997) MEPS (2018)
Pressure filter	1	Mild steel	10,89m3/hr (0,33m2)	18,58m2	$86 500,00	2014	576,1	0,55	$2 563,02	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991), Perry et al ., (1997)
Stripped solids hopper	1	Mild steel	4,74m3	0.2832m3	$1800,00	2014	576,1	0,6	$9 615,05	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Solids sluice tank	1	Mild steel	1,67m3	75,71m3	$46 800,00	2014	576,1	0,6	$4 675,76	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Slurry wash water surge tank	1	Mild steel	0,07m3	75,71m3	$46 800,00	2014	576,1	0,6	$697,08	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Gold extraction										
Dewatered solids buffer hopper	1	Polymer concrete (Epoxy resin 58%)	0,68m3	1m3	$150,00	2010	550,8	0,6	$449,61	Improchem (Mukariri (2012)) Material properties http://web.mit.edu/course/3/3.11/www/modules/props.pdf 12/08/2018
Hydrated lime feed hopper	1	Polymer concrete (Epoxy resin 58%)	0,867m3	1m3	$150,00	2010	550,8	0,6	$520,17	Improchem (Mukariri (2012)) Material properties http://web.mit.edu/course/3/3.11/www/modules/props.pdf 12/08/2018
Sodium cyanide dosing tank	1	SS316	8,16m3	5m3	$11 500,00	2000	394,1	0,53	$21 482,30	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Sodium cyanide dosing tank (Elution)	1	SS316	7,95m3	5m3	$11 500,00	2000	394,1	0,53	$21 173,75	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Leach slurry conditioning tank	1	SS316	11,19m3	5m3	$11 500,00	2000	394,1	0,53	$25 379,50	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Gold leach tanks	6	SS316	2,2 m3	5m3	$11 500,00	2000	394,1	0,53	$10 189,41	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Pressure filter	1	SS316	10,89m3/hr (0,33m2)	18,58m2	$86 500,00	2014	576,1	0,6	$2 139,18	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus and Peters (1991)
Gold leach surge tank	1	SS316	9,67m3	5m3	$11 500,00	2000	394,1	0,53	$24 747,44	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Adsorption tanks	6	Mild steel	0,912m3	30m3	$216 837,00	2013	567,6	0,47	$41 976,62	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Dilute hydrochloric acid tank 3%	1	Hastelloy C	3,98m3	5m3	$11 500,00	2000	394,1	0,53	$14 673,75	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Preheated soft water tank	1	Mild steel	14,4m3	75,71m3	$46 800,00	2014	576,1	0,6	$17 030,97	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Eluant solution (make up & storage)	1	SS316	7,96m3	5m3	$11 500,00	2000	394,1	0,53	$21 187,86	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Loaded carbon feed hopper	1	Mild steel	6,90m3	20,95m3	$13 300,00	2014	576,1	0,6	$6 728,51	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus and Peters (1991)
Acid wash tank	1	Hastelloy C	3,98m3	2,6m3	$5 366,00	2001	394,3	0,3	$8 775,31	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Elution column	1	Hastelloy C	4,60m3	2,6m3	$5 366,00	2001	394,3	0,3	$9 164,84	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
NaOHpH correction tank	1	Rigid PVC 10mm thick or composite	0,59m3	1m3	$150,00	2010	550,8	0,6	$112,61	Improchem (Mukariri (2012)) Timmerhaus and Peters (1991)
Weak eluant recycle tank	1	SS316	3,98m3	5m3	$11 500,00	2000	394,1	0,53	$14 673,75	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Heat exchanger (electrowinning)	1	SS316 (Shell and tube)	4,544KW	1500KW	$52 451,00	2001	394,3	0,81	$688,31	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Heat exchanger (eluant solution)	1	Mild steel (plate)	4,544KW	1500KW	$52 451,00	2001	394,3	0,81	$688,31	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Heat exchanger (soft water)	1	Hastelloy C (shell and tube)	4,544KW	1500KW	$52 451,00	2001	394,3	0,81	$688,31	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Heat exchanger (dilute hydrochloric acid)	1	Reinforced PVC	4,544KW	1500KW	$52 451,00	2001	394,3	0,81	$688,31	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Eluate storage tank/circulation (electrowinning)	1	SS316	7,96m3	5m3	$11 500,00	2000	394,1	0,53	$21 187,86	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Electrowinning cells	6	Mild steel	1,211es/hr	5tes/hr	$42 092,00	2013	567,6	0,8	$13 436,68	Timmerhaus & Peters (1991), Turton et al., (2009), Van Wyk (2014)
Eluted carbon tank	1	Mild steel	1,065m3	75,71m3	$46 800,00	2014	576,1	0,6	$3 569,70	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Rotary kiln	1	Mild steel	1,92 tes/hr	4,55tes/hr	$360 000,00	1978	218,8	0,7	$590 415,59	Zanitsch & Lynch (1978), Van Wyk (2014)
Barren carbon dewatering screen (single deck)	1	Mild steel	1,32 tes/hr (1m2)	2,97m2	$20 900,00	2014	576,1	0,6	$10 714,22	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Carbon quencher/cooler	1	Mild steel	(2,31tes/hr) 1,26m3	75,71m3	$46 800,00	2014	576,1	0,6	$3 948,61	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Regeneration carbon screen (double deck)	1	Mild steel	1,32 tes/hr (1m2)	0,6tes/hr (2,97 m2)	$20 800,00	2014	576,1	0,6	$10 662,95	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Activated carbon feed hopper	1	Mild steel	2,13m3	20,95m3	$13 300,00	2014	576,1	0,6	$3 323,82	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Utilities										
Water softening unit	1	Mild steel	0,162m3	3,53m3	$19 212,22	2012	584,6	0,6	$235,82	Improchem (Mukariri (2012)), Timmerhaus and Peters (1991)
Brine tank	1	PVC	0,65m3	1m3	$150,00	2010	550,8	0,6	$119,35	Improchem (Mukariri (2012)), Timmerhaus and Peters (1991)
Air compressor	1	MSS, Reciprocating, single stage	1,4 1m3/hr/ 0,45Hp	125Hp	$88 700,00	2014	576,1	0,9	$123,76	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Receiving cylinder	1	Mild steel	0,027m3/ 80,59kg	9071,85kg	$76 700,00	2014	576,1	0,62	$4 901,46	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Scavenging and effluent management										
Scavenging cells acid stream	1	Equipment supplier specifications	2,43tes/hr	5tes/hr	$42 092,00	2013	567,6	0,8	$23 628,28	Timmerhaus and Peters (1991), Turton et al., (2009), Van Wyk (2014)
Scavenging cells basic gold stream	1	Equipment supplier specifications	0,48 tes/hr	5tes/hr	$42 092,00	2013	567,6	0,8	$6 455,65	Timmerhaus and Peters (1991), Turton et al., (2009), Van Wyk (2014)
Blended effluent tank (acid stream)	1	SS316	58,26m3	5m3	$11 500,00	2000	394,1	0,53	$60 848,45	Timmerhaus and Peters (1991), Turton et al., (2009), Van Wyk (2014)
Blended effluent tank (basic stream)	1	SS316	11,46m3	5m3	$11 500,00	2000	394,1	0,53	$25 702,24	Timmerhaus and Peters (1991), Turton et al., (2009), Van Wyk (2014)
Barren effluent conditioning tank	1	Mild steel	69,72m3	75,71m3	$46 800,00	2014	576,1	0,6	$43 876,94	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Polished effluent distribution tank (acid stream)	1	Mild steel	58,26m3	75,71m3	$46 800,00	2014	576,1	0,6	$39 395,28	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Polished effluent distribution tank (basic stream)	1	Mild steel	11,46m3	75,71m3	$46 800,00	2014	576,1	0,6	$14 850,21	Online estimator (http://www.matche.com/equipcost/Bin.html), Timmerhaus & Peters (1991)
Spent WPCB solids hopper	1	Rigid PVC 10mm thick	6,78m3	1m3	$150,00	2010	550,8	0,6	$487,31	Improchem (Mukariri (2012)), Timmerhaus & Peters (1991)

Table F.9: Process usage ratios used for the calculations

Material/chemical	Remarks	Usage ratio	Units	Commercial Purity (%)	Strength for process	Unit Price	Source
Crushing and milling							
WPCBs		4350.60	Tes/Tes Au	100%(Categories 2, 3 and 4. Low, medium and high grade)	100%	Average	NGOs, Private suppliers, informal traders, governement offices
Power		312.65	KWh/Tes WPCBs	3 phase AC for industry	350-380V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Water		0.00	m3/Tes WPCBs	Municipal potable 100%	Potable 100%	R38,72/Klitre metered	City of Johannesburg tarrifs 2017/2018
Solder leaching							
WPCBs		4353542.94	Kg/Tes Au	100%(Categories 2, 3 and 4. Low, medium and high grade)	100%	Internal transfer price ()	WPCBs pretreatment plant
Nitric acid		933.39	kg/Tes solids	55%Industrial grade	0,0593	US$350/tonne	Shijiazhuang Xinlongwei Chemical Co., Ltd, China
Water		13.98	m3/ Tes solids	Municipal	Potable 100%	R38,72/Klitre metered	City of Johannesburg tarrifs 2017/2018
Power		121.65	KWh/ Tes solids	3 phase AC for industry	220V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Copper extraction							
WPCBs		6080.78	kg/Tes Cu	100%(Categories 2, 3 and 4. Low, medium and high grade)	100%	Internal transfer price	Solder leaching plant
Sulphuric acid start up	Start up	4623.12	Kg/Tes Cu	98%Industrial grade	16,4%	US290/tonne	
Sulphuric acid	Steady state	2713.95	Kg/Tes Cu	98%Industrial grade	16,4%	US290/tonne	Shijiazhuang Xinlongwei Chemical Co., Ltd, China
Water	Process, SX wash and cathode stripp	40.64	m3/Tes Cu	Municipal potable 100%	Potable 100%	R38,72/K litre metered	City of Johannesburg tarrifs 2017/2018
Kerosene	Once off loading per annum	115.56	m3	Aviation Kerosene JP54, sulphonated 99,96%min	68%blend	US$315/Tonne	LLC Alliance (Alibaba.com)
LIX 984N	Once off loading per annum	54.38	m3	100%(1: 1 volume blend of LIX 860N-I and LIX 84-I)	32%blend	US$11,14/kg	BASF, (Mwale (2009))
Solvent losses		0.37	kg/Tes Cu	32%LIX 984N 68%Kerosene	32:68 LIX:Kerosene	US$3,78/kg	Calculated
Sodium chloride		0.77	kg/Tes Cu	(Industrial grade) 99%	4,19%	US$100/tonne	Lianyungang Huaihua International Trade Co.,Ltd (FOB), China (Alibaba.com)
Hydrogen peroxide		917.52	kg/Tes Cu	Industrial grade 50%	30,00%	US$580/tonne	Zhengzhou Double Tree Enterprise Co., Ltd (Alibaba. Com)
Sodium Hydroxide		30.89	kg/Tes Cu	Industrial grade 97%>	7,55%	US$400/tonne	Henan Bright Commercial Co., Ltd, China (Alibaba.com)
Cobalt Sulphate		0.49	kg/Tes Cu	Industrial grade >96,96%	12,50%	US$7,60/kg	Sincere Union- Import & Export Co., Ltd, Hebei, China (Alibaba.com)
Bentonite clay	For treatment bled raffinate to efflue	340.7	kg/Tes Cu	Industrial grade	2,00%	US0,45/kg	Shandong Jiqi International Trade Co., Ltd. China (Alibaba.com)
DXG-F7/Guar gum	Grain modifier	0.15	kg/Tes Cu	Yellow powder, maximum humidity 14%, solubilty 90%(min)	0,83%	US$2/kg	MARKHOR (Albaba.com)
FC-1100 /Eagles FC-1100	Acid mist suppressant	0.16	L/Tes Cu	55%Fluorochemicals, 45%water (FC-1100), 50%Eagles 1100	10%	US$200/kg	3M mining chemicals/ Henan, China (Mainland)(Alibaba.com)
Polypropylene macrospher	Acid mist suppressant, once off load	2.72	kg	2000 units per kg. White, hollow.	100%polypropylene, dia 20mm	US$485,24/kg	K-Mac plastics (http://k-mac-plastics.com/polypropylene-plastic-balls.htm)
Power		3507.87	KWh/Tes Cu	50 Hz, 3 phase AC for industry	350-380V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Nitric acid wash							
WPCBs		2966883.00	kg/Tes Au	100%(Categories 2, 3 and 4. Low, medium and high grade)	100%	Internal transfer price	Copper extraction plant
Nitric acid		426.54	kg/Tes solids	55%Industrial grade	55%	US$350/tonne	Shijiazhuang Xinlongwei Chemicals, China
Water		10.78	m3/Tes solids	Municipal potable 100%	Potable 100%	R38,72/Klitre metered	City of Johannesburg tarrifs 2017/2018
Compressed air		3.75	m3/Tes solids	21%Oxygen, 79%Nitrogen, 2,3 bars	21%O2, 79%N2, 2.3 bars	US$6,56/m3	https://www.hydraulicspneumatics.com/air-compressors/determine-cost-compressed-air-your-plant
Power		159.82	KWh/Tes solids	3 phase AC for industry	220V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Gold extraction							
WPCBs		3086935.23	kg/Tes Au	100%(Categories 2, 3 and 4. Low, medium and high grade)	100%	Internal transfer price	Nitric acid wash plant
Sodium cyanide		649207.20	kg/Tes Au	98%NaCN briquettes	0,025%leaching, 3%elution	US$1650/tonne	ex China Alibaba.com
Sodium hydroxide		601251.66	kg/Tes Au	>98%NaOH flakes	2%elution, pH correction of eluate 5(	US$400/tonne	Henan Bright Commercial Co., Ltd, China (Alibaba.com)
Hydrated lime		103101.95	kg/Tes Au	Industrial grade 95%	>95%	US130/tonne	Zibo Qilu Chemicals Company China (Alibaba.com)
Water		33513.39	m3/Tes Au	Municipal water (Carbon regeneration, leaching, eluate pH cor	Potable 100%	US$2,63/Klitre metered	City of Johannesburg tarrifs 2017/2018
Water		35382.77	m3/Tes Au	Soft water (Acid washing and elution)	<1ppm Total hardness	Internal transfer price ()	Water softening section
Purchased Oxygen		393.68	kg/Tes Au	100%	100%	R41,29/Kg	AFROX
Total Oxygen gas		484.60	kg/Tes Au	Enriched air and purchased oxygen	100%	R41,29/Kg	AFROX
Activated carbon	Once off loading per annum for 2 bat	218.21	kg	100%coconut shell, 1,2-1,5 mm dia	100%	US$1000/tonne	Chifeng Fuyue Activated Carbon Factory, China (Alibaba. Com)
Activated carbon losses		155.50	kg/Tes Au	Carbon fines	<0,65mm dia, <30g/t	US$1000/tonne	ex China Alibaba.com
Peas coal		4033463.98	kg/Tes Au	Peas , anthracite. Calorific value 27,4 -29,5 M J/kg, <8%Sulphu	Minimum calorific value 27,4M J/Kg	US$106,70/tonne	https://tradingeconomics.com/commodity/coal
Hydrochloric acid		644218.78	kg/Tes Au	Industrial grade 33%	3%	US$250/tonne	Zouping Changshan Town Zefeng Fertilizer Factory, China (Alibaba.com)
Compressed air		33569.93	m3/Tes Au	21%Oxygen, 79%Nitrogen, 2,3 bars	21%Oxygen, 79%Nitrogen, 2,3 bars	US$6,56/m3	https://www.hydraulicspneumatics.com/air-compressors/determine-cost-compressed-air-your-plant
Power		1161986.83	KWh/Tes Au	3 phase AC for industry	350-380V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Water softening							
Water for soft water		35382.77	m3/Tes Au	Municipal potable 100%	Potable 100%	US$2,63/Klitre metered	City of Johannesburg tarrifs 2017/2018
Water for regeneration		3750.08	m3/Tes Au	Municipal potable 100%	Potable 100%	US$2,63/Klitre metered	City of Johannesburg tarrifs 2017/2018
Sodium chloride		0.27	kg/m3 soft water	Industrial course salt, 99%	10%brine	US$150/tonne	Xinle City Hongyuan Carbon Black Sales Center, Hebei, China
Power		2,215469515894742.2	KWh/m3 soft wate	3 phase AC for industry	220V	R17,19/KWh	Eskom business rate tarrifs 2018/2019 (VAT included), Business rate 3
Resin (DOWEX Marathon	Once off loading per annum	65.00	kg/annum	Cationic resin Styrene-DVB gel with a sulphonic acid active gr	100%	US$3,88/kg	https://www.servapure.com/Dowex-Marathon-C-Softening-Na-Form-Cation-Resin_p_5546.html
Air compression							
Power		0.24	KWh/m3 air	21%Oxygen, 79%Nitrogen, 2,3 bars	21%Oxygen, 79%Nitrogen, 2,3 bars	US$1,17/KWh	https://www.hydraulicspneumatics.com/air-compressors/determine-cost-compressed-air-your-plant

Table F.10: Summary of the effluent streams from the different plants (Effluent flows for model 3a used for illustration kgs/annum)

Material	Size reduction	Solder leaching	Copper extraction		Nitric acid wash	Gold extraction		Soft water generation	Compressed air section	Summary of entire process
			Raffinate bleed	SX LO wash water		Barren leachate	Ex HCl acid wash			
H2O	0	6463314	949288	4844062	1396508	756451	555635	56886	2231	**15024374**
HNO3	0	256042	0	0	0	0	0	0	0	**256042**
NaCl	0	0	5	0	0	0	0	41	0	**46**
Pb(NO3)2	0	14563	0	0	312	0	0	0	0	**14876**
Sn(NO3)2	0	2664	0	0	0	0	0	0	0	**2664**
Ni(NO3)2	0	9788	0	0	7946	0	0	0	0	**17733**
Zn(NO3)2	0	229	0	0	855	0	0	0	0	**1085**
Fe(NO3)3	0	110650	0	0	25740	0	0	0	0	**136390**
Al(NO3)3	0	11400	0	0	49703	0	0	0	0	**61104**
AgNO3	0	39	0	0	1295	0	0	0	0	**1334**
Cu(NO3)2	0	549	0	0	9398	0	0	0	0	**9946**
Solvent	0	0	0	0	0	0	0	0	0	**0**
H2SO4	0	0	54266	7078	0	0	0	0	0	**61345**
Al2(SO4)3	0	0	4570	0	0	0	0	0	0	**4570**
FeSO4	0	0	8093	39	0	0	0	0	0	**8132**
NiSO4	0	0	4640	9415	0	0	0	0	0	**14055**
ZnSO4	0	0	5176	0	0	0	0	0	0	**5176**
PbSO4	0	0	0	0	0	0	0	0	0	**0**
SnSO4	0	0	0	0	0	0	0	0	0	**0**
Ag2SO4	0	0	0	0	0	0	0	0	0	**0**
CoSO4	0	0	0	24818	0	0	0	0	0	**24818**
DXG-F7	0	0	1	0	0	0	0	0	0	**1**
FC-1100	0	0	1	0	0	0	0	0	0	**1**
HCl	0	0	0	0	0	0	8459	0	0	**8459**
FeCl3	0	0	0	0	0	0	120	0	0	**120**
AlCl3	0	0	0	0	0	0	3	0	0	**3**
ZnCl2	0	0	0	0	0	0	0	0	0	**0**
NiCl2	0	0	0	0	0	0	1	0	0	**1**
PbCl2	0	0	0	0	0	0	0	0	0	**0**
NaOH	0	0	0	0	0	22	0	0	0	**22**
NaCN	0	0	0	0	0	0	0	0	0	**0**
NaAu(CN)2	0	0	0	0	0	1	0	0	0	**1**
NaAg(CN)2	0	0	0	0	0	0	0	0	0	**0**
Carbon fines	0	0	0	0	0	4	0	0	0	**4**
NaAu(CN)2 on carbon fines	0	0	0	0	0	0	0	0	0	**0**
Total material flow	**0**	**6869237**	**1026042**	**4885412**	**1491758**	**756478**	**564216**	**56927**	**2231**	**15652302**
Approximate pH range	**n/a**	**3-4,5**	**0,2-1**	**1,8-2,0**	**4-6,0**	**10-11,0**	**0,3-1**	**6,8-7**	**7**	
TDSmg/L	**0**	**21819**	**22406**	**7065**	**68206**	**31**	**221**	**1302**	**0**	**17293**

Table F.11: Annual schedule of material requirements for the entire process (based on business model 3a)

Materials	Commercial purity/strength/pressure	Strength for the process	Units	Size reduction	Solder leaching	Copper extraction	Nitric acid wash	Gold extraction	Soft water generation	Compressed air section	The entire operation
Waste PCBs (100)	100% (Categories 2, 3 and 4. Low, medium and high grade)	100%	kg	457140.86	457450.09	442170.12	311746.30	324360.83	0.00	0.00	**457140.86**
Nitric acid (%)	55% Industrial grade	0.0593% SL, 55%NA	kg	0.00	776311.08	0.00	354758.06	0.00	0.00	0.00	**1131069.15**
Sulphuric acid	98% Industrial grade	16%	kg	0.00	0.00	45.82	0.00	0.00	0.00	0.00	**45.82**
Sulphuric acid steady state	98% Industrial grade	0.164	kg	0.00	0.00	197347.20	0.00	0.00	0.00	0.00	**197347.20**
Kerosene	Aviation Kerosene JP54, sulphonated 99,96% min	68% blend	kg	0.00	0.00	115.56	0.00	0.00	0.00	0.00	**115.56**
LIX 984N	100% (1 : 1 volume blend of LIX 860N-I and LIX 84-I)	32% blend	kg	0.00	0.00	54.38	0.00	0.00	0.00	0.00	**54.38**
Solvent losses (crud)	32% LIX 984N 68% Kerosene	32:68 LIX:Kerosene	kg	0.00	0.00	26.81	0.00	0.00	0.00	0.00	**26.81**
Sodium chloride	(Industrial grade) 99%	4.19%	kg	0.00	0.00	56.56	0.00	0.00	1000.42	0.00	**56.56**
Hydrogen peroxide	Industrial grade 50%	30.00%	kg	0.00	0.00	133436.71	0.00	0.00	0.00	0.00	**133436.71**
Sodium Hydroxide	>98% NaOH flakes	2% elution, 7.55% Cu pH correction	kg	0.00	0.00	57.13	0.00	64466.06	0.00	0.00	**64523.19**
Cobalt Sulphate	Industrial grade >96,96%	12.50%	kg	0.00	0.00	36.97	0.00	0.00	0.00	0.00	**36.97**
Bentonite clay	Industrial grade	2.00%	kg	0.00	0.00	24774.39	0.00	0.00	0.00	0.00	**24774.39**
DXG-F7/Guar gum	Yellow powder, maximum humidity 14%, solubilty 90% (min)	0.83%	kg	0.00	0.00	10.91	0.00	0.00	0.00	0.00	**10.91**
FC-1100 /Eagles FC-1100	55% Fluorochemicals, 45% water (FC-1100), 50% Eagles 1100	10.00%	kg	0.00	0.00	11.63	0.00	0.00	0.00	0.00	**11.63**
Polypropylene macrospheres	100	100	kg	0.00	0.00	2.72	0.00	0.00	0.00	0.00	**2.72**
Sodium cyanide	98% NaCN briquettes	0.025% leaching, 3% elution	kg	0.00	0.00	0.00	0.00	69607.84	0.00	0.00	**69607.84**
Hydrated lime	Industrial grade 95%	>95%	kg	0.00	0.00	0.00	0.00	11403.66	0.00	0.00	**11403.66**
Water	Municipal poTable 100%	PoTable 100%	m3	0.00	6395.65	2028.51	3360.46	3521.43	0.00	0.00	**15306.05**
Soft Water	Soft water (Acid washing and elution)	<1ppm Total hardness	m3	0.00	0.00	0.00	0.00	3717.86	0.00	0.00	**3717.86**
Purchased Oxygen	100%	100%	kg	0.00	0.00	0.00	0.00	41.37	0.00	0.00	**41.37**
Total Oxygen gas	Enriched air and purchased oxygen	100%	kg	0.00	0.00	0.00	0.00	50.92	0.00	0.00	**50.92**
Activated carbon	100% coconut shell, 1,2-1,5 mm dia	100%	kg	0.00	0.00	0.00	0.00	218.21	0.00	0.00	**218.21**
Activated carbon losses	Carbon fines	<0.65mm dia, <30g/t	kg	0.00	0.00	0.00	0.00	49.51	0.00	0.00	**49.51**
Hydrochloric acid	Industrial grade 33%	3%	kg	0.00	0.00	0.00	0.00	205125.82	0.00	0.00	**205125.82**
Resin (DOWEX Marathon C)	Cationic resin Styrene-DVB gel with a sulphonic acid active group	100%	kg	0.00	0.00	0.00	0.00	0.00	65.00	0.00	**0.00**
Utilities											
Power	50 Hz, 3 phase AC for industry	220V/350-380V	KWh	143020.80	55648.32	255078.28	49823,28	122096.18	8236.80	1144.03	**625666.87**
Fuel (Peas coal)	Peas , anthracite. Calorific value 27,4 -29,5 MJ/kg, <8% Sulphur	Minimum calorific value 27.4MJ/Kg	kg	0.00	0.00	0.00	0.00	423817.68	0,00	0.00	**423817.68**
Water for softening	Municipal poTable 100%	PoTable 100%	m3	0.00	0.00	0.00	0.00	0.00	3717.86	0.00	**0.00**
Water for regeneration	Municipal poTable 100%	PoTable 100%	m3	0.00	0.00	0.00	0.00	0.00	1170024.14	0.00	**0.00**
Compressed air (bar)	Ambient air 1,01 bars	Ambient air 2.3 bars	m3	0.00	0.00	0.00	1216.98	3527.37	0.00	0.00	**4744.36**

Table F.12: Schedule of detailed labour requirements (Overall schedule based on business model 3a)

Manpower requirements for operating configuration 2

Roaster structure - 4 shift system

Position	Man hours/day	Man-hours/year & individual	Complement for 4 shift roaster	Total man-hours per year	Skill level	Average hourly wage	Total labour costs
Executive management							
General manager	8	1784	1	1784	Specialised	$15.50	$29 031.50
Administration department							
Administration manager	8	1784	1	1784	Specialised	$9.77	$18 299.21
IT administrator	8	1784	1	1784	Skilled	$4.05	$7 585.80
Supplies and logistics officer	8	1784	1	1784	Skilled	$5.53	$10 357.69
Warehouse and dispatch clerk	8	1784	1	1784	Semiskilled	$2.93	$5 487.89
Buyer	8	1784	1	1784	Skilled	$5.05	$9 458.65
Accountant	8	1784	1	1784	Specialised	$5.54	$10 382.71
Finance and salaries administrator	8	1784	1	1784	Skilled	$5.54	$10 382.71
Personnel and training officer	8	1784	1	1784	Skilled	$4.55	$8 522.15
Sales officer	8	1784	1	1784	Skilled	$5.05	$9 458.65
Personnel assistant and receptionist	8	1784	1	1784	Semiskilled	$1.65	$3 090.45
Production department							
Production manager	8	2014	1	2014	Specialised	$9.77	$20 546.31
Process superintendent	8	2060	2	4120	Skilled	$6.04	$25 422.36
Process foremen	8	1784	1	1784	Skilled	$5.69	$10 657.37
Analytical chemist	8	1784	1	1784	Skilled	$6.04	$11 312.92
SHEQ systems officer	8	1784	1	1784	Skilled	$5.69	$10 657.37
Shift laboratory technician	8	2496	8	19968	Skilled	$5.19	$103 633.92
Shift drivers	8	2496	4	9984	Unskilled	$2.95	$29 452.80
Fork lift drivers	8	2496	1	2496	Unskilled	$2.95	$7 363.20
Shift artisans (engineering)	8	2496	4	9984	Skilled	$6.95	$69 388.80
General hands	8	1784	3	5352	Unskilled	$1.59	$8 509.68
Size reduction plant operators	8	2496	8	19968	Semiskilled	$3.13	$62 500.64
Solder leaching plant operators	8	2496	8	19968	Semiskilled	$3.13	$62 500.64
Copper extraction plant operators	8	2496	8	19968	Semiskilled	$3.13	$62 500.64
Nitric acid wash plant operators	8	2496	8	19968	Semiskilled	$3.13	$62 500.64
Gold extraction plant operators	8	2496	8	19968	Semiskilled	$3.13	$62 500.64
Effluent and solid waste treatment plant operators	8	2496	12	29952	Semiskilled	$3.13	$93 750.96
Operating labour							**$406 254.15**
Other labour							**$362 376.11**
Supervisory and labour							**$56 626.04**
TOTAL WAGE BILL							**$825 256.30**

Production schedule - operating schedule 1

Basis 400 Tonnes WPCBs per annum

Day	M	T	W	T	F	S	S
Start up	a						
Week 1		a					
			a	a	a	a	a
			a				
				a	a		
Steady state	b						
Week 2		b					
	a	a	a/b	b	b	b	b
			b				
				b	b		
Week 3	c						
		c					
	b	b	b/c	c	c	c	c
			c				
				c	c		
Week 4	d						
		d					
	c	c	c/d	d	d	d	d
			d				
				d	d		
Week 5	e						
		e					
	d	d	d/e	e	e	e	e
			e				
				e	e		

Operation key

		Utilization %
Size reduction		14.29
Solder leaching		14.29
Copper extraction		100.00
Nitric acid wash		14.29
Gold extraction		28.57

Batch codes   a,b,c,d….

From start up first batch	a
Second batch	b
Third batch	c
Fourth batch	d
Fifth batch	e
……..	…….

Figure F.1: Production operating schedule for the first configuration from start up

Production schedule - operating schedule 2

Basis 400 Tonnes WPCBs per annum

Day	M	T	W	T	F	S	S
Start up	a	a	a	a	a	a	a
Week 1							a
Week 2	b	b	b	b	b	b	b
	a	a	a	a	a	a	b
						a	a
						a	a
Steady state	c	c	c	c	c	c	c
Week 3	b	b	b	b	b	b	c
	a	a	a	a	a	a/b	b
	a	a	a	a	a	b	b
						a	a
Week 4	d	d	d	d	d	d	d
	c	c	c	c	c	c	d
	b	b	b	b	b	b/c	c
	b	b	b	b	b	c	c
	a	a	a	a	a	b	b
Week 5	e	e	e	e	e	e	e
	d	d	d	d	d	d	
	c	c	c	c	c	c/d	d
	c	c	c	c	c	d	d
	b	b	b	b	b	c	c

Operation key

		Utilization %
Size reduction		100
Solder leaching		100
Copper extraction		100
Nitric acid wash		100
Gold extraction		100

Batch codes   a,b,c,d….

From start up first batch	a
Second batch	b
Third batch	c
Fourth batch	d
Fifth batch	e
……..	…….

Figure F.2: Production operating schedule for the second configuration from start up

Production schedule - operating schedule 3
Basis 400 Tonnes WPCBs per annum

Day	M	T	W	T	F	S	S
Start up	a						
Week 1		a					
			a	a	a	a	a
			a	a	a	a	a
Steady state	b						
Week 2		b					
	a	a	a/b	b	b	b	b
	a	a	b	b	b	b	b
			a	a	a	a	a
Week 3	c						
		c					
	b	b	b/c	c	c	c	c
	b	b	c	c	c	c	c
	a	a	b	b	b	b	b
Week 4	d						
		d					
	c	c	c/d	d	d	d	d
	c	c	d	d	d	d	d
	b	b	c	c	c	c	c
Week 5	e						
		e					
	d	d	d/e	e	e	e	e
	d	d	e	e	e	e	e
	c	c	d	d	d	d	d

Operation key

		Utilization %
Size reduction		14,29
Solder leaching		14,29
Copper extraction		100,00
Nitric acid wash		100,00
Gold extraction		100,00

Batch codes   a,b,c,d….

From start up first batch	a
Second batch	b
Third batch	c
Fourth batch	d
Fifth batch	e
……..	…….

Figure F.3: Production operating schedule for the third configuration from start up

Table F.13: Capital cost distribution by plant in each business model

Business model	Size reduction plant (US$)	Solder leaching plant (US$)	Copper extraction plant (US$)	Nitric acid wash circuit (US$)	Gold extraction circuit (US$)	Soft water production unit (US$)	Compressed air unit (US$)	Effluent and solid waste management plant (US$)	Total capital investment (US$)
3a	195 427	465 021	5 970 240	483 698	6 984 490	21 699	55 330	1 484 069	15 659 976
3b	195 427	465 021	915 413	483 698	825 268	0	55 330	751 518	3 691 676
3c	195 427	465 021	1 019 572	483 698	6 984 490	21 699	55 330	1 227 522	10 452 761
3d	195 427	465 021	5 970 240	483 698	825 268	0	55 330	1 033 859	9 028 845
4a	0	465 021	5 968 218	483 698	6 984 490	21 699	55 330	1 484 069	15 464 549
4b	0	465 021	915 413	483 698	825 268	0	55 330	751 518	3 496 249
4c	0	465 021	915 413	483 698	6 984 490	21 699	55 330	1 227 522	10 153 174
4d	0	465 021	5 970 240	483 698	825 268	0	55 330	1 033 859	8 833 418

Table F.14: Overall capital cost distribution for each business model

Business model	3a	3b	3c	3d	4a	4b	4c	4d
Fixed capital	10 665 504	2 525 046	7 054 890	6 152 941	10 534 126	2 393 669	6 923 513	6 021 564
Working capital	1 909 327	451 340	1 262 592	1 101 152	1 886 243	428 255	1 239 508	1 078 067
Process non consumables	3 087	0	2 250	1 962	2 432	0	2 250	1 962
Equipment capitalized costs	3 072 471	715 290	2 128 534	1 767 696	3 031 506	674 325	1 983 410	1 726 731
Capitalized costs- Process materials	9 587	0	4 494	5 093	6 489	0	4 494	5 093

Table F.15: Breakdown of annual operating costs for each of the business models

Business model	Direct costs (US$/yr)	Indirect costs (US$/yr)	General operating expenses (US$/yr)	Contingency (US$/yr)	Total operating costs (US$/yr)
3a	5 975 558	724 675	3 416 689	381 611	10 498 533
3b	3 907 702	496 300	2 751 957	209 688	7 365 647
3c	4 842 028	621 772	2 890 377	281 312	8 635 490
3d	5 116 899	599 203	3 299 507	314 366	9 329 975
4a	6 363 107	633 233	3 190 015	310 318	10 496 672
4b	4 294 953	404 858	2 525 159	138 404	7 363 374
4c	5 229 422	530 330	2 663 692	210 029	8 633 472
4d	5 504 154	507 761	3 072 710	243 082	9 327 706

Table F.16: Analysis of individual plant operating costs

Plant	3a	3b	3c	3d	4a	4b	4c	4d
Size reduction	$1 858 652	$1 858 651	$1 858 492	$1 858 651	$0.00	$0.00	$0.00	$0.00
Solder leaching	$3 600 443	$3 600 443	$3 600 265	$3 600 443	$3 598 833	$3 598 833	$3 598 833	$3 598 833
Copper extraction	$6 456 799	$4 937 127	$4 971 687	$6 460 410	$6 455 241	$4 935 201	$4 969 973	$6 458 484
Nitric acid wash	$7 999 538	$6 276 700	$6 336 167	$8 003 538	$7 997 748	$6 274 542	$6 334 248	$8 001 384
Gold extraction	$10 748 753	$7 615 173	$8 885 028	$9 580 197	$10 746 748	$7 612 756	$8 882 878	$9 577 783
Softened water	$46 168	$0.00	$48 785	$0.00	$46 168	$0.00	$48 785	$0.00
Compressed air	$7 234	$5 172	$7 235	$5 202	$7 234	$5 172	$7 234	$5 202
Effluent treatment plant	$1 607 464	$902 444	$992 138	$1 517 770	$1 607 464	$902 444	$992 138	$1 517 770
Operating cost drivers								
Raw materials	19.13%	23.81%	23.94%	18.79%	25.99%	33.58%	32.27%	26.50%
Effluent and waste treatment	15.31%	12.25%	11.49%	16.27%	15.31%	12.26%	11.49%	16.27%
Sales and distribution	20.47%	23.16%	20.83%	22.20%	19.28%	21.47%	19.38%	20.86%
Research and dev	10.24%	11.58%	10.41%	11.10%	9.64%	10.73%	9.69%	10.43%
Cumulative of op costs	**65.15%**	**70.81%**	**66.66%**	**68.35%**	**70.22%**	**78.04%**	**72.83%**	**74.06%**

Table F.17: Accounting for copper distribution in each of the business models

Destination	3a	3b	3c	3d	4a	4b	4c	4d
Final product	78.07%	96.31%	96.32%	78.07%	78.07%	96.31%	96.32%	78.07%
Co-deposit with gold	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Scavenging	18.24%	0.00%	0.00%	18.24%	18.24%	0.00%	0.00%	18.24%
Tailings (to polishing)	3.69%	3.69%	3.68%	3.69%	3.69%	3.69%	3.68%	3.69%

Table F.18: Accounting for the distribution of gold in each business model

Destination	3a	3b	3c	3d	4a	4b	4c	4d
Final product	90.34%	95.00%	90.34%	95.00%	90.34%	95.00%	90.34%	95.00%
Scavenging (recovery)	0.55%	0.00%	0.55%	0.00%	0.55%	0.00%	0.55%	0.00%
Tailings (polishing and recovery)	6.32%	5.00%	6.32%	5.00%	6.32%	5.00%	6.32%	5.00%
Lock up in process	2.79%	0.00%	2.79%	0.00%	2.79%	0.00%	2.79%	0.00%

Table F.19: Analysis of operating costs versus gross revenue of each business model 3a to 4d

Model	Gross revenue	Total Annual operating costs
3a	4 432 134	10 498 533
3b	3 828 305	7 365 647
3c	4 525 641	8 635 490
3d	3 734 798	9 329 975
4a	4 432 134	10 496 672
4b	3 828 305	7 363 374
4c	4 525 640	8 633 472
4d	3 734 798	9 327 706

Table F.20: Comparison of discounted cumulative cash flows for each of the models (3a-4d)

Comparison of discounted cumulative cash flows for each of the models (3a-4d)								
	All cumulative discounted cash flows in US$							
Year (n)	3a	3b	3c	3d	4a	4b	4c	4d
0	0	0	0	0	0	0	0	0
0	0	0	0	0	0	0	0	0
0	-15 656 223	-3 691 676	-10 448 353	-9 028 845	-15 460 796	-3 496 249	-10 148 767	-8 833 418
0	-16 616 118	-3 918 930	-11 083 293	-9 582 609	-16 408 867	-3 711 679	-10 771 883	-9 375 358
1	-19 975 356	-6 007 881	-13 343 741	-12 834 691	-19 770 131	-5 802 393	-13 034 257	-12 629 205
2	-22 958 693	-7 863 078	-15 351 244	-15 722 863	-22 755 268	-7 659 155	-15 043 469	-15 518 945
3	-25 608 193	-9 510 677	-17 134 105	-18 287 847	-25 406 366	-9 308 144	-16 827 850	-18 085 322
4	-27 961 213	-10 973 909	-18 717 464	-20 565 808	-27 760 805	-10 772 611	-18 412 558	-20 364 519
5	-30 050 928	-12 273 404	-20 123 644	-22 588 864	-29 851 781	-12 073 203	-19 819 936	-22 388 673
6	-31 906 804	-13 427 485	-21 372 472	-24 385 539	-31 708 776	-13 228 258	-21 069 828	-24 186 324
7	-33 555 005	-14 452 424	-22 481 555	-25 981 165	-33 357 972	-14 254 061	-22 179 856	-25 782 816
8	-35 018 772	-15 362 671	-23 466 531	-27 398 239	-34 822 622	-15 165 077	-23 165 671	-27 200 660
9	-36 318 743	-16 171 062	-24 341 288	-28 656 742	-36 123 377	-15 974 150	-24 041 173	-28 459 846
10	-37 473 247	-16 888 993	-25 118 159	-29 774 418	-37 278 577	-16 692 687	-24 818 706	-29 578 129
11	-38 498 560	-17 526 587	-25 808 098	-30 767 026	-38 304 509	-17 330 819	-25 509 233	-30 571 275
12	-39 409 141	-18 092 834	-26 420 832	-31 648 560	-39 215 639	-17 897 544	-26 122 489	-31 453 288
13	-40 217 827	-18 595 718	-26 965 001	-32 431 450	-40 024 813	-18 400 852	-26 667 122	-32 236 603
14	-40 936 021	-19 042 329	-27 448 277	-33 126 734	-40 743 440	-18 847 840	-27 150 810	-32 932 264
15	-41 573 849	-19 438 964	-27 877 475	-33 744 216	-41 381 653	-19 244 809	-27 580 373	-33 550 081
16	-42 140 303	-19 791 215	-28 258 645	-34 292 601	-41 948 449	-19 597 358	-27 961 868	-34 098 764
17	-42 643 371	-20 104 049	-28 597 162	-34 779 621	-42 451 820	-19 910 456	-28 300 673	-34 586 049
18	-43 090 145	-20 381 877	-28 897 798	-35 212 144	-42 898 863	-20 188 518	-28 601 565	-35 018 806
19	-43 486 925	-20 628 615	-29 164 793	-35 596 267	-43 295 883	-20 435 465	-28 868 788	-35 403 138
20	-43 661 431	-20 805 697	-29 284 288	-35 834 822	-43 472 752	-20 614 882	-28 990 635	-35 644 029

Table F.21: Calculations for plots of capitalization ratio (CR) against time for each of the models (3a-4d)

Recapitalization risk assessment for each business model								
	Plot of capitalization ratio over the life of the project							
Year (n)	3a	3b	3c	3d	4a	4b	4c	4d
0								
0								
1	-6.49	-8.40	-6.26	-7.24	-6.51	-8.60	-6.57	-7.29
2	-7.46	-10.99	-7.21	-8.87	-7.49	-11.36	-7.58	-8,96
3	-8.32	-13.30	-8.04	-10.32	-8.36	-13.80	-8.48	-10.44
4	-9.08	-15.34	-8.79	-11.60	-9.14	-15.98	-9.28	-11.76
5	-9.76	-17.16	-9.45	-12.74	-9.83	-17.90	-9.99	-12.93
6	-10.36	-18.77	-10.03	-13.76	-10.44	-19.62	-10.62	-13.97
7	-10.90	-20.20	-10.56	-14.66	-10.98	-21.14	-11.17	-14.89
8	-11.37	-21.48	-11.02	-15.45	-11.46	-22.49	-11.67	-15.71
9	-11.80	-22.61	-11.43	-16.16	-11.89	-23.69	-12.11	-16.43
10	-12.17	-23.61	-11.79	-16.80	-12.27	-24.75	-12.50	-17.08
11	-12.50	-24.50	-12.12	-17.36	-12.61	-25.70	-12.85	-17.65
12	-12.80	-25.29	-12.40	-17.85	-12.91	-26.54	-13.16	-18.16
13	-13.06	-26.00	-12.66	-18.29	-13.17	-27.29	-13.44	-18.61
14	-13.30	-26.62	-12.89	-18.69	-13.41	-27.95	-13.68	-19.02
15	-13.50	-27.18	-13.09	-19.03	-13.62	-28.54	-13.90	-19.37
16	-13.69	-27.67	-13.27	-19.34	-13.81	-29.06	-14.09	-19.69
17	-13.85	-28.11	-13.43	-19.62	-13.97	-29.53	-14.26	-19.97
18	-14.00	-28.49	-13.57	-19.86	-14.12	-29.94	-14.41	-20.22
19	-14.12	-28.84	-13.69	-20.08	-14.25	-30.31	-14.54	-20.44
20	-14.18	-29.09	-13.75	-20.21	-14.31	-30.57	-14.61	-20.58

Table F.22: Calculations for plots of capital rate of return ratio (CRR) against time for each of the models (3a-4d)

Comparison of capital rate of return ratio for each of the models (3a-4d)								
	CRRs over the life of the project for each model							
Year (n)	3a	3b	3c	3d	4a	4b	4c	4d
0								
0	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00	-1.00
1	-1.20	-1.53	-1.20	-1.34	-1.20	-1.56	-1.21	-1.35
2	-1.38	-2.01	-1.39	-1.64	-1.39	-2.06	-1.40	-1.66
3	-1.54	-2.43	-1.55	-1.91	-1.55	-2.51	-1.56	-1.93
4	-1.68	-2.80	-1.69	-2.15	-1.69	-2.90	-1.71	-2.17
5	-1.81	-3.13	-1.82	-2.36	-1.82	-3.25	-1.84	-2.39
6	-1.92	-3.43	-1.93	-2.54	-1.93	-3.56	-1.96	-2.58
7	-2.02	-3.69	-2.03	-2.71	-2.03	-3.84	-2.06	-2.75
8	-2.11	-3.92	-2.12	-2.86	-2.12	-4.09	-2.15	-2.90
9	-2.19	-4.13	-2.20	-2.99	-2.20	-4.30	-2.23	-3.04
10	-2.26	-4.31	-2.27	-3.11	-2.27	-4.50	-2.30	-3.15
11	-2.32	-4.47	-2.33	-3.21	-2.33	-4.67	-2.37	-3.26
12	-2.37	-4.62	-2.38	-3.30	-2.39	-4.82	-2.43	-3.35
13	-2.42	-4.75	-2.43	-3.38	-2.44	-4.96	-2.48	-3.44
14	-2.46	-4.86	-2.48	-3.46	-2.48	-5.08	-2.52	-3.51
15	-2.50	-4.96	-2.52	-3.52	-2.52	-5.18	-2.56	-3.58
16	-2.54	-5.05	-2.55	-3.58	-2.56	-5.28	-2.60	-3.64
17	-2.57	-5.13	-2.58	-3.63	-2.59	-5.36	-2.63	-3.69
18	-2.59	-5.20	-2.61	-3.67	-2.61	-5.44	-2.66	-3.74
19	-2.62	-5.26	-2.63	-3.71	-2.64	-5.51	-2.68	-3.78
20	-2.63	-5.31	-2.64	-3.74	-2.65	-5.55	-2.69	-3.80

Table F.23: Calculations for plots of interest recovery period (IRP) against time for each of the models (3a-4d)

Comparison of IRP for each of the business models (3a-4d)								
	All cumulative undiscounted cash flows in US$							
Year (n)	3a	3b	3c	3d	4a	4b	4c	4d
0								
0								
1	-16 616 118	-3 918 930	-11 083 293	-9 582 609	-16 408 867	-3 711 679	-10 771 883	-9 375 358
2	-20 398 620	-6 271 089	-13 628 557	-13 244 453	-20 193 651	-6 065 823	-13 319 316	-13 039 190
3	-24 181 121	-8 623 248	-16 173 822	-16 906 297	-23 978 434	-8 419 967	-15 866 748	-16 703 022
4	-27 963 623	-10 975 407	-18 719 086	-20 568 141	-27 763 217	-10 774 111	-18 414 181	-20 366 854
5	-31 746 125	-13 327 566	-21 264 350	-24 229 985	-31 548 000	-13 128 255	-20 961 614	-24 030 686
6	-35 528 626	-15 679 725	-23 809 614	-27 891 829	-35 332 783	-15 482 399	-23 509 047	-27 694 517
7	-39 311 128	-18 031 884	-26 354 878	-31 553 673	-39 117 567	-17 836 542	-26 056 479	-31 358 349
8	-43 093 629	-20 384 043	-28 900 143	-35 215 517	-42 902 350	-20 190 686	-28 603 912	-35 022 181
9	-46 876 131	-22 736 202	-31 445 407	-38 877 361	-46 687 133	-22 544 830	-31 151 345	-38 686 013
10	-50 658 632	-25 088 361	-33 990 671	-42 539 205	-50 471 916	-24 898 974	-33 698 778	-42 349 845
11	-54 441 134	-27 440 521	-36 535 935	-46 201 049	-54 256 699	-27 253 118	-36 246 210	-46 013 676
12	-58 223 635	-29 792 680	-39 081 200	-49 862 893	-58 041 483	-29 607 262	-38 793 643	-49 677 508
13	-62 006 137	-32 144 839	-41 626 464	-53 524 737	-61 826 266	-31 961 406	-41 341 076	-53 341 340
14	-65 788 638	-34 496 998	-44 171 728	-57 186 581	-65 611 049	-34 315 550	-43 888 508	-57 005 172
15	-69 571 140	-36 849 157	-46 716 992	-60 848 424	-69 395 832	-36 669 693	-46 435 941	-60 669 004
16	-73 353 641	-39 201 316	-49 262 256	-64 510 268	-73 180 615	-39 023 837	-48 983 374	-64 332 836
17	-77 136 143	-41 553 475	-51 807 521	-68 172 112	-76 965 399	-41 377 981	-51 530 807	-67 996 667
18	-80 918 645	-43 905 634	-54 352 785	-71 833 956	-80 750 182	-43 732 125	-54 078 239	-71 660 499
19	-84 701 146	-46 257 793	-56 898 049	-75 495 800	-84 534 965	-46 086 269	-56 625 672	-75 324 331
20	-88 483 648	-48 609 952	-59 443 313	-79 157 644	-88 319 748	-48 440 413	-59 173 105	-78 988 163

Table F.24: A summary of the additional gross revenue from sales of palladium, silver and tin at a business efficiency of 70% scenario 1

Model	Gross revenue scenario 2	Tin, silver, and palladium revenue	Total Annual operating costs
3a	6 466 509	1 630 278	10 498 533
3b	5 632 226	1 630 278	7 365 647
3c	6 460 238	1 630 278	8 635 490
3d	5 638 497	1 630 278	9 329 975
4a	6 466 509	1 630 278	10 496 672
4b	5 632 226	1 630 278	7 363 374
4c	6 460 238	1 630 278	8 633 472
4d	5 638 497	1 630 278	9 327 706

Table F.25: A summary of the additional gross revenue from sales of silver, tin and palladium at a business efficiency of 70% scenario 2

Model	Gross revenue scenario 2	Tin, silver, and palladium revenue	Total Annual operating costs
3a	11 758 120	6 921 890	10 498 533
3b	10 923 838	6 921 890	7 365 647
3c	11 751 850	6 921 890	8 635 490
3d	10 930 108	6 921 890	9 329 975
4a	11 758 120	6 921 890	10 496 672
4b	10 923 838	6 921 890	7 363 374
4c	11 751 850	6 921 890	8 633 472
4d	10 930 108	6 921 890	9 327 706

Table F.26: Effluent outflows from the respective sections per annum (kgs per annum)

Business model	Size reduction	Solder leaching	Copper extraction	Nitric acid wash	Gold extraction	Soft water generation	Compressed air section
3a	0	6869237	5911454	1491758	1320695	56927	2231
3b	0	6869237	0	1491758	0	0	2231
3c	0	6869237	0	1491758	1320695	56927	2231
3d	0	6869237	5911454	1491758	0	0	2231
4a	0	6869237	5911454	1491758	1320695	56927	2231
4b	0	6869237	0	1491758	0	0.00	2231
4c	0	6869237	0	1491758	1320695	56927	2231
4d	0	6869237	5911454	1491758	0	0	2231
Approximate pH range	n/a	3-4.5	0.2-1 &1.8-2.0	4-6.0	0.3-1.0 & 10-11.0	6.8-7	7.00
TDSppm	0	21819	22406 & 7065	68205.71	31 & 221	1302	0

Table F.27: The expected solid waste outflows originating directly from the process (kgs per annum).

Business model	Size reduction	Solder leaching	Copper extraction	Nitric acid wash	Gold extraction	Soft water generation	Compressed air section
3a	0	0	5144	0	316348	0	0
3b	0	0	0	0	316348	0	0
3c	0	0	0	0	316348	0	0
3d	0	0	5144	0	316348	0	0
4a	0	0	5144	0	316348	0	0
4b	0	0	0	0	316348	0	0
4c	0	0	0	0	316348	0	0
4d	0	0	5144	0	316348	0	0

G: Calculations for the sensitivity analysis

The sensitivity analysis was done with the broad objective of determining at which points variable the factors would improve the performance of the remaining models in group 4. An examination of all these factors revealed that the performance of the business models could only be improved significantly by increasing the gold content. Detailed in this appendix are the data sheets used in plotting the graphs discussed in chapter 8.

Changes in total capital investment

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.1: Sensitivity of models to changes in total capital invested

Sensitivity of models to changes in total capital investment				
NPV (US$)				
Change in initial capital invested (%)	4a	4b	4c	4d
-50	-36 402 410	-19 124 507	-24 661 660	-31 875 617
-40	-37 816 478	-19 422 582	-25 527 455	-32 629 299
-30	-39 230 546	-19 720 657	-26 393 250	-33 382 982
-20	-40 644 615	-20 018 732	-27 259 045	-34 136 664
-10	-42 058 683	-20 316 807	-28 124 840	-34 890 347
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-44 886 820	-20 912 956	-29 856 430	-36 397 711
20	-46 300 889	-21 211 031	-30 722 225	-37 151 394
30	-47 714 957	-21 509 106	-31 588 020	-37 905 076
40	-49 129 026	-21 807 181	-32 453 815	-38 658 759
50	-50 543 094	-22 105 256	-33 319 610	-39 412 441
60	-51 957 163	-22 403 331	-34 185 406	-40 166 124
70	-53 371 231	-22 701 406	-35 051 201	-40 919 806
80	-54 785 300	-22 999 481	-35 916 996	-41 673 489
90	-56 199 368	-23 297 555	-36 782 791	-42 427 171
100	-57 613 437	-23 595 630	-37 648 586	-43 180 854

Changes in operating costs

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.2: Sensitivity of models to changes in annual operating costs

Sensitivity of models to changes in annual operating costs				
	NPV (US$)			
Change in annual operating costs (%)	4a	4b	4c	4d
-50	-16 276 191	-1 536 603	-6 621 570	-11 476 223
-40	-21 715 503	-5 352 259	-11 095 383	-16 309 785
-30	-27 154 815	-9 167 915	-15 569 196	-21 143 346
-20	-32 594 127	-12 983 570	-20 043 009	-25 976 907
-10	-38 033 440	-16 799 226	-24 516 822	-30 810 468
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-48 912 064	-24 430 537	-33 464 448	-40 477 590
20	-54 351 376	-28 246 193	-37 938 261	-45 311 151
30	-59 790 689	-32 061 849	-42 412 074	-50 144 712
40	-65 230 001	-35 877 504	-46 885 887	-54 978 273
50	-70 669 313	-39 693 160	-51 359 700	-59 811 835
60	-76 108 625	-43 508 816	-55 833 513	-64 645 396
70	-81 547 937	-47 324 471	-60 307 326	-69 478 957
80	-86 987 250	-51 140 127	-64 781 139	-74 312 518
90	-92 426 562	-54 955 783	-69 254 952	-79 146 079
100	-97 865 874	-58 771 438	-73 728 765	-83 979 640

Changes in CAPEX and OPEX

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.3: Sensitivity of business models to changes in CAPEX and OPEX

Sensitivity of models to changes in CAPEX and OPEX				
NPV (US$)				
Change in CAPEX and OPEX (%)	4a	4b	4c	4d
-90	18 207 674	16 408 693	19 065 837	14 641 163
-80	11 354 294	12 294 963	13 726 229	9 053 920
-70	4 500 913	8 181 232	8 386 621	3 466 676
-60	-2 352 468	4 067 502	3 047 013	-2 120 568
-50	-9 205 848	-46 229	-2 292 595	-7 707 811
-40	-16 059 229	-4 159 959	-7 632 203	-13 295 055
-30	-22 912 610	-8 273 690	-12 971 811	-18 882 298
-20	-29 765 990	-12 387 421	-18 311 419	-24 469 542
-10	-36 619 371	-16 501 151	-23 651 027	-30 056 785
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-50 326 133	-24 728 612	-34 330 243	-41 231 273
20	-57 179 513	-28 842 343	-39 669 851	-46 818 516
30	-64 032 894	-32 956 073	-45 009 459	-52 405 760
40	-70 886 275	-37 069 804	-50 349 067	-57 993 003
50	-77 739 655	-41 183 534	-55 688 675	-63 580 247
60	-84 593 036	-45 297 265	-61 028 283	-69 167 490
70	-91 446 417	-49 410 995	-66 367 892	-74 754 734
80	-98 299 797	-53 524 726	-71 707 500	-80 341 978
90	-105 153 178	-57 638 456	-77 047 108	-85 929 221
100	-112 006 559	-61 752 187	-82 386 716	-91 516 465

Changes in annual effluent treatment costs

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.4: Sensitivity of models to changes in annual effluent treatment costs

Sensitivity of models to changes in annual effluent and waste treatment costs				
	NPV (US$)			
Change in effluent and waste treatment costs (%)	4a	4b	4c	4d
-50	-39 301 554	-18 266 188	-26 413 729	-31 711 273
-40	-40 135 794	-18 735 927	-26 929 110	-32 497 824
-30	-40 970 033	-19 205 666	-27 444 491	-33 284 376
-20	-41 804 273	-19 675 404	-27 959 873	-34 070 927
-10	-42 638 512	-20 145 143	-28 475 254	-34 857 478
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-44 306 991	-21 084 620	-29 506 016	-36 430 580
20	-45 141 231	-21 554 359	-30 021 398	-37 217 131
30	-45 975 471	-22 024 098	-30 536 779	-38 003 682
40	-46 809 710	-22 493 836	-31 052 160	-38 790 234
50	-47 643 950	-22 963 575	-31 567 541	-39 576 785
60	-48 478 189	-23 433 314	-32 082 923	-40 363 336
70	-49 312 429	-23 903 052	-32 598 304	-41 149 887
80	-50 146 668	-24 372 791	-33 113 685	-41 936 438
90	-50 980 908	-24 842 530	-33 629 066	-42 722 989
100	-51 815 148	-25 312 268	-34 144 448	-43 509 541

Changes in copper prices

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.5: Sensitivity of models to changes in copper metal price (fixed gold/ gold liquor)

Sensitivity of models to changes in copper metal price (fixed gold/ gold liquor)				
	NPV (US$)			
Change in copper price (%)	4a	4b	4c	4d
-50	-44 840 977	-21 966 860	-30 342 614	-37 012 254
-40	-44 567 332	-21 696 464	-30 072 218	-36 738 609
-30	-44 293 687	-21 426 069	-29 801 822	-36 464 964
-20	-44 020 042	-21 155 673	-29 531 427	-36 191 319
-10	-43 746 397	-20 885 277	-29 261 031	-35 917 674
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-43 199 107	-20 344 486	-28 720 240	-35 370 384
20	-42 925 462	-20 074 090	-28 449 844	-35 096 739
30	-42 651 817	-19 803 695	-28 179 448	-34 823 094
40	-42 378 172	-19 533 299	-27 909 052	-34 549 449
50	-42 104 527	-19 262 903	-27 638 657	-34 275 804
60	-41 830 882	-18 992 507	-27 368 261	-34 002 159
70	-41 557 237	-18 722 112	-27 097 865	-33 728 514
80	-41 283 591	-18 451 716	-26 827 470	-33 454 869
90	-41 009 946	-18 181 320	-26 557 074	-33 181 224
100	-40 736 301	-17 910 925	-26 286 678	-32 907 579

Changes in gold prices

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.6: Sensitivity of models to changes in gold metal price (fixed copper/copper liquor)

Sensitivity of models to changes in gold metal price (fixed copper/copper liquor)				
	NPV (US$)			
Change in gold price (%)	4a	4b	4c	4d
-50	-54 635 054	-29 631 830	-40 152 937	-44 660 977
-40	-52 402 594	-27 828 440	-37 920 477	-42 857 588
-30	-50 170 133	-26 025 051	-35 688 017	-41 054 198
-20	-47 937 673	-24 221 661	-33 455 556	-39 250 808
-10	-45 705 212	-22 418 271	-31 223 096	-37 447 419
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-41 240 291	-18 811 492	-26 758 175	-33 840 639
20	-39 007 831	-17 008 102	-24 525 714	-32 037 250
30	-36 775 371	-15 204 713	-22 293 254	-30 233 860
40	-34 542 910	-13 401 323	-20 060 793	-28 430 470
50	-32 310 450	-11 597 933	-17 828 333	-26 627 081
60	-30 077 989	-9 794 543	-15 595 873	-24 823 691
70	-27 845 529	-7 991 154	-13 363 412	-23 020 301
80	-25 613 068	-6 187 764	-11 130 952	-21 216 911
90	-23 380 608	-4 384 374	-8 898 491	-19 413 522
100	-21 148 147	-2 580 985	-6 666 031	-17 610 132

Changes in metal prices

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.7: Sensitivity of models to changes in metal prices

Sensitivity of models to changes in metal prices				
	NPV (US$)			
Change in product prices (%)	**4a**	**4b**	**4c**	**4d**
-50	-56 003 279	-30 983 809	-41 504 916	-46 029 203
-40	-53 497 174	-28 910 023	-39 002 060	-43 952 168
-30	-50 991 068	-26 836 238	-36 499 204	-41 875 133
-20	-48 484 963	-24 762 452	-33 996 347	-39 798 099
-10	-45 978 857	-22 688 667	-31 493 491	-37 721 064
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-40 966 646	-18 541 096	-26 487 779	-33 566 994
20	-38 460 541	-16 467 311	-23 984 923	-31 489 960
30	-35 954 435	-14 393 525	-21 482 067	-29 412 925
40	-33 448 330	-12 319 740	-18 979 211	-27 335 890
50	-30 942 224	-10 245 955	-16 476 354	-25 258 855
60	-28 436 119	-8 172 169	-13 973 498	-23 181 821
70	-25 930 013	-6 098 384	-11 470 642	-21 104 786
80	-23 423 908	-4 024 598	-8 967 786	-19 027 751
90	-20 917 802	-1 950 813	-6 464 930	-16 950 716
100	-18 411 697	122 972	-3 962 074	-14 873 682

Changes in copper sales

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.8: Sensitivity of models to changes in copper sales volume (with fixed gold sales)

Sensitivity of models to changes in copper sales volumes (with fixed gold sales)				
	NPV (US$)			
Change in copper sales (%)	4a	4b	4c	4d
-100	-46 209 202	-23 318 839	-31 694 592	-38 380 480
-90	-45 935 557	-23 048 443	-31 424 196	-38 106 834
-80	-45 661 912	-22 778 047	-31 153 801	-37 833 189
-70	-45 388 267	-22 507 651	-30 883 405	-37 559 544
-60	-45 114 622	-22 237 256	-30 613 009	-37 285 899
-50	-44 840 977	-21 966 860	-30 342 614	-37 012 254
-40	-44 567 332	-21 696 464	-30 072 218	-36 738 609
-30	-44 293 687	-21 426 069	-29 801 822	-36 464 964
-20	-44 020 042	-21 155 673	-29 531 427	-36 191 319
-10	-43 746 397	-20 885 277	-29 261 031	-35 917 674
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-43 199 107	-20 344 486	-28 720 240	-35 370 384
20	-42 925 462	-20 074 090	-28 449 844	-35 096 739
30	-42 651 817	-19 803 695	-28 179 448	-34 823 094
40	-42 378 172	-19 533 299	-27 909 052	-34 549 449
50	-42 104 527	-19 262 903	-27 638 657	-34 275 804
60	-41 830 882	-18 992 507	-27 368 261	-34 002 159
70	-41 557 237	-18 722 112	-27 097 865	-33 728 514
80	-41 283 591	-18 451 716	-26 827 470	-33 454 869
90	-41 009 946	-18 181 320	-26 557 074	-33 181 224
100	-40 736 301	-17 910 925	-26 286 678	-32 907 579

Movement of gold sales

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.9: Sensitivity of models to changes in gold sales volumes (with fixed copper sales)

Sensitivity of models to changes in gold sales volumes (with fixed copper sales)				
	NPV (US$)			
Change in gold sales (%)	4a	4b	4c	4d
-100	-65 797 356	-38 648 779	-51 315 240	-53 677 926
-90	-63 564 896	-36 845 389	-49 082 779	-51 874 536
-80	-61 332 435	-35 041 999	-46 850 319	-50 071 147
-70	-59 099 975	-33 238 609	-44 617 858	-48 267 757
-60	-56 867 515	-31 435 220	-42 385 398	-46 464 367
-50	-54 635 054	-29 631 830	-40 152 937	-44 660 977
-40	-52 402 594	-27 828 440	-37 920 477	-42 857 588
-30	-50 170 133	-26 025 051	-35 688 017	-41 054 198
-20	-47 937 673	-24 221 661	-33 455 556	-39 250 808
-10	-45 705 212	-22 418 271	-31 223 096	-37 447 419
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-41 240 291	-18 811 492	-26 758 175	-33 840 639
20	-39 007 831	-17 008 102	-24 525 714	-32 037 250
30	-36 775 371	-15 204 713	-22 293 254	-30 233 860
40	-34 542 910	-13 401 323	-20 060 793	-28 430 470
50	-32 310 450	-11 597 933	-17 828 333	-26 627 081
60	-30 077 989	-9 794 543	-15 595 873	-24 823 691
70	-27 845 529	-7 991 154	-13 363 412	-23 020 301
80	-25 613 068	-6 187 764	-11 130 952	-21 216 911
90	-23 380 608	-4 384 374	-8 898 491	-19 413 522
100	-21 148 147	-2 580 985	-6 666 031	-17 610 132

Movement of copper and gold sales

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.10: Sensitivity of models to changes in copper and gold sales volumes

Sensitivity of models to changes in sales volumes				
	NPV (US$)			
Change in sales (%)	4a	4b	4c	4d
-100	-68 533 807	-41 352 736	-54 019 197	-56 414 376
-90	-66 027 701	-39 278 950	-51 516 340	-54 337 342
-80	-63 521 596	-37 205 165	-49 013 484	-52 260 307
-70	-61 015 490	-35 131 379	-46 510 628	-50 183 272
-60	-58 509 385	-33 057 594	-44 007 772	-48 106 237
-50	-56 003 279	-30 983 809	-41 504 916	-46 029 203
-40	-53 497 174	-28 910 023	-39 002 060	-43 952 168
-30	-50 991 068	-26 836 238	-36 499 204	-41 875 133
-20	-48 484 963	-24 762 452	-33 996 347	-39 798 099
-10	-45 978 857	-22 688 667	-31 493 491	-37 721 064
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-40 966 646	-18 541 096	-26 487 779	-33 566 994
20	-38 460 541	-16 467 311	-23 984 923	-31 489 960
30	-35 954 435	-14 393 525	-21 482 067	-29 412 925
40	-33 448 330	-12 319 740	-18 979 211	-27 335 890
50	-30 942 224	-10 245 955	-16 476 354	-25 258 855
60	-28 436 119	-8 172 169	-13 973 498	-23 181 821
70	-25 930 013	-6 098 384	-11 470 642	-21 104 786
80	-23 423 908	-4 024 598	-8 967 786	-19 027 751
90	-20 917 802	-1 950 813	-6 464 930	-16 950 716
100	-18 411 697	122 972	-3 962 074	-14 873 682

Changes in WPCB prices

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.11: Sensitivity of models to changes in WPCB purchase prices

Sensitivity of models to changes in WPCB purchase prices				
	NPV (US$)			
Change in WPCB prices (%)	4a	4b	4c	4d
-100	-28 028 884	-5 171 014	-13 546 767	-20 200 161
-90	-29 573 271	-6 715 401	-15 091 154	-21 744 548
-80	-31 117 658	-8 259 787	-16 635 541	-23 288 935
-70	-32 662 044	-9 804 174	-18 179 928	-24 833 322
-60	-34 206 431	-11 348 561	-19 724 315	-26 377 708
-50	-35 750 818	-12 892 948	-21 268 701	-27 922 095
-40	-37 295 205	-14 437 334	-22 813 088	-29 466 482
-30	-38 839 592	-15 981 721	-24 357 475	-31 010 869
-20	-40 383 978	-17 526 108	-25 901 862	-32 555 255
-10	-41 928 365	-19 070 495	-27 446 248	-34 099 642
0	-43 472 752	-20 614 882	-28 990 635	-35 644 029
10	-45 017 139	-22 159 268	-30 535 022	-37 188 416
20	-46 561 525	-23 703 655	-32 079 409	-38 732 803
30	-48 105 912	-25 248 042	-33 623 796	-40 277 189
40	-49 650 299	-26 792 429	-35 168 182	-41 821 576
50	-51 194 686	-28 336 816	-36 712 569	-43 365 963
60	-52 739 073	-29 881 202	-38 256 956	-44 910 350
70	-54 283 459	-31 425 589	-39 801 343	-46 454 736
80	-55 827 846	-32 969 976	-41 345 729	-47 999 123
90	-57 372 233	-34 514 363	-42 890 116	-49 543 510
100	-58 916 620	-36 058 749	-44 434 503	-51 087 897

Changes in copper recoveries

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.12: Sensitivity of models to changes in copper recoveries

Sensitivity of models to changes in copper recoveries				
	NPV (US$)			
Copper recovery (% per year)	4a	4b	4c	4d
0	-46 209 202	-23 318 839	-31 694 592	-38 380 480
10	-45 925 073	-23 038 083	-31 413 837	-38 096 350
20	-45 640 944	-22 757 327	-31 133 081	-37 812 221
30	-45 356 814	-22 476 572	-30 852 325	-37 528 091
40	-45 072 685	-22 195 816	-30 571 570	-37 243 962
50	-44 788 555	-21 915 061	-30 290 814	-36 959 832
60	-44 504 426	-21 634 305	-30 010 059	-36 675 703
70	-44 220 296	-21 353 550	-29 729 303	-36 391 574
80	-43 936 167	-21 072 794	-29 448 548	-36 107 444
90	-43 652 038	-20 792 038	-29 167 792	-35 823 315
100	-43 367 908	-20 511 283	-28 887 036	-35 539 185

Changes in gold recoveries

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.13: Sensitivity of models to changes in gold recoveries

Sensitivity of models to changes in gold recoveries				
	NPV (US$)			
Gold recovery (% per year)	4a	4b	4c	4d
0	-65 797 356	-38 648 779	-51 315 240	-53 677 926
10	-63 500 823	-36 827 173	-49 018 706	-51 856 320
20	-61 204 289	-35 005 567	-46 722 172	-50 034 714
30	-58 907 755	-33 183 961	-44 425 638	-48 213 109
40	-56 611 221	-31 362 356	-42 129 105	-46 391 503
50	-54 314 688	-29 540 750	-39 832 571	-44 569 897
60	-52 018 154	-27 719 144	-37 536 037	-42 748 291
70	-49 721 620	-25 897 538	-35 239 504	-40 926 686
80	-47 425 086	-24 075 933	-32 942 970	-39 105 080
90	-45 128 553	-22 254 327	-30 646 436	-37 283 474
100	-42 832 019	-20 432 721	-28 349 902	-35 461 868

Changes in gold and copper recoveries

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.14: Sensitivity of models to changes in both copper and gold recoveries

Sensitivity of models to changes in gold and copper recoveries				
	NPV (US$)			
Gold & Copper recovery (% per year)	4a	4b	4c	4d
0	-68 533 807	-41 352 736	-54 019 197	-56 414 376
10	-65 953 144	-39 250 374	-51 441 907	-54 308 641
20	-63 372 480	-37 148 013	-48 864 618	-52 202 906
30	-60 791 817	-35 045 652	-46 287 329	-50 097 171
40	-58 211 154	-32 943 290	-43 710 039	-47 991 436
50	-55 630 491	-30 840 929	-41 132 750	-45 885 701
60	-53 049 828	-28 738 568	-38 555 461	-43 779 965
70	-50 469 165	-26 636 206	-35 978 171	-41 674 230
80	-47 888 502	-24 533 845	-33 400 882	-39 568 495
90	-45 307 838	-22 431 484	-30 823 593	-37 462 760
100	-42 727 175	-20 329 122	-28 246 303	-35 357 025

Changes in capacity utilization

The base case has been highlighted in grey. Where the base case lies between two percentages the range is highlighted in grey

Table G.15: Sensitivity of models to changes in capacity utilization

Sensitivity of models to changes in capacity utilization				
	NPV (US$)			
Capacity utilization (% per year)	4a	4b	4c	4d
10	-26 467 609	-11 018 651	-18 885 978	-20 709 122
20	-28 357 069	-12 084 899	-20 008 717	-22 368 556
30	-30 246 529	-13 151 147	-21 131 457	-24 027 991
40	-32 135 990	-14 217 394	-22 254 197	-25 687 425
50	-34 025 450	-15 283 642	-23 376 937	-27 346 859
60	-35 914 910	-16 349 890	-24 499 676	-29 006 293
70	-37 804 371	-17 416 138	-25 622 416	-30 665 727
80	-39 693 831	-18 482 386	-26 745 156	-32 325 161
90	-41 583 292	-19 548 634	-27 867 895	-33 984 595
100	-43 472 752	-20 614 882	-28 990 635	-35 644 029

H: Waste management landscape of South Africa

Table H.1: Waste management landscape 1989 to 2017 (Godfrey & Oelofse, 2017)

Age	Key highlights
Land-filling as of 1989	South African waste management still reliant on land-filling as the method of choice. Promoted by low gate fees to access landfills. 90% of all waste generated in the country reporting to land-filling. Policy development and regulation of landfills stimulated by impacts to environment and societies. Birth of the Environmental Conservation Act (Act 73 of 1989). Provided guidelines for handling and classification of waste. Active licensing of landfills.
Emergence of recycling	Stimulus driven by social needs and demand for specific resources. Push by government to increase waste recycling White paper on IP & WM and the first National Waste Management Summit (NWMS). Polokwane declaration published. Recycling targets defined and ratified however not legalised. Publication of guidelines by the Department of Environmental Affairs and Tourism (DEAT) to assist in the implementation of the NWMS. Growth in the waste recycling sector and reuse characterised by the significant input of the informal sector. Government spearheaded efforts to define formalised structures to effectively capture input of the informal sector. This was through promotion of cooperatives that suffered due to operational setbacks.
Flood of regulation	Publication of the white paper on Integrated Pollution and Waste Management (IP & WM). In 2009 the first legislation on waste management was published. Provided mechanism to regulate every aspect of waste management and exploitation of secondary resources in the value chain. Objectives aimed to control the sector and impacts of poor waste management efforts, promote value realisation from recycling activities. 41 other acts related to waste management were documented. Legislative acts also supported by local municipal bylaws. First talk about EPR being adopted in SA
Drive for EPR	Alternative policy instruments published to support EPR drive New tax measures instituted which became an area of debate. After 2009 shift from voluntary to mandatory EPR schemes Government expressed desire for EPR schemes for the lighting, WEEE and the packaging and paper sectors.

Table H.2: Legislation in South Africa affecting e-waste management practices (Finlay, 2005; GIZ, 2013)

Legislation	Description
The National Environmental Management Act (Act 107 of 1998) (NEMA)	Pillars for waste management are specified in the operating context of this act. Elements of EPR are however not succinctly defined. The act embodies aspects related to the polluter pays principle, re-use, and waste reduction, recycling and proper disposal.
The Constitution of the Republic of South Africa (Act 106 of 1996)	Provides overarching policy on the preservation of the environment.
The National Environmental Management: Waste Act (Act 59 of 2008)	Provides for the regulation and control of all waste and includes electronic and electrical waste.
The Hazardous Substances Act	This Act is concerned with the regulation of the management of hazardous substances, storage and disposal of hazardous waste.
The Health Act	Supporting Act that advances healthy living and occupational conditions.
The White Paper on Integrated Pollution and Waste Management	This paper is concerned with the allocation of environmental and waste management powers. The holistic waste management of water is embodied in the core function of the National Waste Management strategy. Other focal areas include the waste information system, general waste collection, waste treatment and disposal and capacity building and educational activities.
(ECA) The Environment Conservation Act. Act 73 of 1989	Works in collaboration with the Department of Environmental Affairs and Tourism (DEAT) for the administration of the disposal of waste. This act covers the protection of the environment.
The Occupational Health and Safety Act (OHSA) (Act 85 of 1993)	Aspects of health and safety concerning workplaces are covered in this act.
Municipal Services Act (Act 32 of 2000)	Effective local governance is specified within this act.
Second –hand Goods Bill	Provides for the regulation of the trade in second-hand goods. Objectively it limits the trade in stolen goods while promoting ethical standards in the second-hand sector.
Atmospheric Pollution Prevention Act, 45 of 1965	Specific smelting, melting or burning processes require a registration certificate. These processes include lead, copper, waste incineration, cadmium, metal recovery, mercury and glass processes.
Air Quality Act, 39 of 2004	This act aims to improve air quality standards and control. It has a significant bearing on smelter operations.
Precious Metals legislation	The trade and smelting activities of precious metals are highly regulated in South Africa. This act covers gold, silver, PGMs
National Pricing Strategy for Waste Management	Was instituted in 2016 with the realisation that undercharging of waste management services does nothing to encourage recycling and reuse of materials. The charges for waste management will be determined in South Africa
Industrial Waste Management Plan IndWMP	The first draft notice was published in 2015. Listed sectors are required to submit waste management plans within two years (12 months for registration and 12 months for submission). Recyclers are required to register with the minister.