

AN INDUSTRIAL ECOLOGY APPROACH TO SULPHIDE CONTAINING MINERAL WASTES TO MINIMISE ARD FORMATION

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*PART 2: Design for Disposal
and Extraction of Products of Value*



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PART 2: Design for Disposal and Extraction of Products of Value

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Water Research Commission

by

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EXECUTIVE SUMMARY

South Africa produces large amounts of fine coal waste each year that is pyrite-bearing and hence a source of acid rock drainage. Similarly, acid generating sulphide-bearing tailings from minerals processing of hard rock are generated in many regions of southern Africa, including South Africa. Acid rock drainage (ARD) represents a major potential environmental impact of mining of these sulphidic minerals and pyritic coal. To prevent its formation, appropriate handling of waste rock or interburden, coal discards, fine coal waste and hard rock tailings is required. In this study, presented as WRC K5/2231, we have addressed both the categorisation of the ARD-generating potential of the waste material and routes towards the prevention of ARD formation from the waste. Our work is informed by the over-arching premise that the re-purposing of benign mineral wastes to other uses, using an industrial ecology focus, is preferable to its long-term disposal, hence we explore this approach. The findings of this project “An Industrial Ecology Approach to Sulphide-containing Mineral Wastes to Minimise ARD Formation” are reported in two parts, presented as two separate reports:

Part 1: Characterising potential for ARD formation

Part 2: Design for Disposal and Extraction of Products of Value

In this report (Part 2), focussing on fine coal waste, we explore the potential to reduce the polluting nature and environmental burden of these wastes, thereby reducing long term liabilities, through the removal of the sulphide from the waste in a two-stage separation process, such as froth flotation, to produce a non-acid-forming bulk stream, a small-volume sulphide rich stream and a recovered coal stream. While the former two streams are typically classified as waste streams, these may be re-purposed for value-addition or for improved resource efficiency and reduced environmental burden or both. The potential of the two-stage separation process for desulphurisation has been demonstrated previously by our team in WRC K5/2215. Here, we examine the techno-economic feasibility of the process as well as its ability to minimise liability and long-term risk. Further, we look into the potential to re-purpose the three streams formed: the coal-rich stream to saleable coal, the sulphide rich stream and the sulphide lean stream.

The risk removal achieved by desulphurisation through the two-stage flotation process is evaluated using life cycle assessment (LCA) and risk assessment tools. These show a reduction in water pollution potential in terms of salinisation, ecotoxicity and metal risks. The consumption of natural resources is also decreased, leading to decreased use of land, water and fossil fuels. Based on the increased electricity consumption, there is an increase in climate change and acidification; however, this is small as the energy requirements of the separation process are typically < 1% of coal recovery. Hence the two-stage desulphurisation is beneficial in terms of risk removal and minimising environmental burden.

On techno-economic feasibility, the process shows clear potential. On considering revenue from the coal product only, the return on investment depends strongly on the coal price and the grade of the fine coal waste. The highest operating cost is that of the reagents, making the search for cheaper or lower concentration reagents important. Sensitivity analysis shows an increasing return with increasing coal price and yield and a decreasing return with increasing reagent usage and cost and other operating cost components. Under all scenarios, it is important to consider the cost of no further treatment, i.e. the ‘as is’ case of covers and barriers and of downstream ARD treatment with continuing liabilities. Improved economic return on the two-stage separation process may be achieved by further valorisation or re-purposing of the product streams, in particular the bulk low sulphur stream and the small-volume high sulphur stream. The potential of this and its feasibility forms the focus of the remainder of the report.

On considering the potential for re-purposing the high sulphur fine coal stream, we have identified a suite of possible products for manufacture from pyrite and assessed their process flow sheet and requisite starting material: sulphuric acid, paint and cement pigment, glass pigments, ferric sulphate as a coagulant, ferrous sulphate heptahydrate, use in lead refining and copper smelting, chrome (VI) reduction, soil amelioration, cemented paste backfill, enhancing heat generation for hot heap leaching and photovoltaics. While a number are not practical owing to needing high grade pyrite, the remainder were evaluated following establishment of decision-making criteria for process selection using the following categories: technical, social, environmental and economic. Using a ranking and scoring approach, four preferred options were identified and three with potential in South Africa explored further: soil amelioration, cemented paste backfill and generating ferric sulphate.

Soil amelioration using high sulphur coal waste is applicable to alkaline soils, providing slow release sulphur, other key metals and acidity. However, supply of gypsum in South Africa is cheaper when taking into account the long transport distances between the coal mines and the appropriate soils in South Africa, thus precluding its ready use as a sulphur source only. Cemented paste backfill into old mine workings serves as waste disposal, reduces surface footprints, stabilises surfaces and provides underground support. It can be used as support for the safe extraction of coal pillars. The long-term stability of the sulphide under these conditions of backfill has yet to be ascertained. Mostly, in South Africa the sulphide rich coal waste represents too small a fraction for this application. Instead, CPB should be considered with sulphide-lean tailings or the overall tailings from the stage one separation, both overcoming the volume requirement and the uncertainty about the long-term fate of the sulphide. Production of ferric sulphate as a coagulant for water treatment was also considered. Process options for its production were considered as were its uses. These uses include both water treatment within the coal industry and external to it. Potential was recognised; however, more detailed studies of its production are required to finalise the technoeconomic evaluation. This is being undertaken in WRC K5/2761 currently.

Applications highlighted for the re-purposing of benign, low sulphur tailings include use in road construction, brick-making, geopolymerisation, soil fabrication and co-disposal for ARD prevention. Many of these were detailed in WRC K5/2215. Here we consider in detail the potential use of low sulphur tailings in co-disposal with waste rock and for soil fabrication for mine site rehabilitation.

In the co-disposal application, the acid-neutralised desulphurised fine coal tailings were used for co-disposal with sulphidic waste rock and interburden, using both layered and blended packing approaches. The co-disposal reduced the permeability of the waste rock bed and hence infiltration into it. Through limiting exposure to oxygen and water and enabling neutralisation of acid produced, the acid generation associated with ARD was reduced successfully. As expected, unless acidification reactions are prevented, neutralisation by the low sulphur tailings cannot be sustained. Where loss of the covered or blended bed integrity occurred, leading to permeate flow, ARD generation followed. Hence successful co-disposal to avoid ARD formation in the long-term requires ongoing flow restriction which is governed by sustaining bed compactness and ongoing bed integrity. Owing to the dependence of this on scale, it is recommended that scale up studies be carried out, together with development of tools to guide selection of the best conditions for stable, low permeability beds.

Soils were fabricated from desulphurised fine coal waste, used in combination with degraded soil, the physical ameliorant malt extract from brewing operations and the nutrient providing amendments, anaerobic digestion sludge, algal biomass and compost. Both soils fabricated from coal waste and degraded soil in the ratios 3:1 and 1:1 were able to support cultivation of the grass *Eragrostis tef*, frequently used in mine site rehabilitation. Plant growth was augmented by both the malt extract physical ameliorant and the nutrient sources of AD sludge and algal biomass. This approach shows great potential for the avoidance of burrowing topsoil, reducing ecological effects and transport costs. Extended studies are required to optimise the soil composition and physical properties as well as to study the development of the soil and its microbial health over repeated growth cycles.

In conclusion, the study has shown the potential benefit environmentally and in terms of reduced long-term liabilities for sulphide removal from fine coal wastes, and hard rock tailings, prior to disposal. This highlights the need for decision making on processing circuits to be informed by both value recovery (non-negotiable) and quality of waste for appropriate disposal. By effective separation, the volume of waste requiring disposal with containment can be reduced substantially, with potential for substantive savings.

Further, opportunities for the re-purposing of waste streams, and associated constraints, have been demonstrated using the more readily accessible 'on-site' applications in the first instance. Such re-purposing, through co-disposal or soil fabrication or other, enhances resource efficiency, reduces the need for exploiting or natural resources, reduces land use and reduces long-term liabilities. This move towards a zero-waste or minimum-waste scenario provides substantial opportunity for enhanced mine closure practices.

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3. Kotsiopoulos A. and Harrison S.T.L. (2018). Co-disposal of benign desulfurised tailings with sulfidic waste rock to mitigate ARD generation: influence of flow and contact surface. *Minerals Engineering*, 116, 62-71. DOI: 10.1016/j.mineng.2017.03.003
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5. Amaral Filho J.R., Stander H.-M., Firpo B., Broadhurst J.L. and Harrison S.T.L. Opportunities for soil-related applications of fine coal processing wastes in South Africa. *Minerals Engineering* (under review).
6. Golela M.T., Opitz A.K.B., Smart M., Ntwampe S.K.O., Broadhurst J.L. and Harrison S.T.L. The semi-continuous UCT biokinetic test provides improved characterization of acid rock drainage potential of sulfidic waste rock by inoculated and naturally occurring microorganisms. *Minerals Engineering* (under review).
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Theses and Dissertations completed:

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ACRONYMS AND ABBREVIATIONS

ABA	Acid-Base Accounting
AF	Acid Forming
AHP	Analytical Hierarchy Process
ANC	Acid Neutralising Capacity
ARD	Acid Rock Drainage
BET	Brunauer, Emmett and Teller
C-H-S gel	Calcium-Silica-Water Gel
CPB	Cemented Paste Backfill
ICP-OES	Inductively Coupled Plasma-Optical Emission Spectroscopy
IRR	Internal Rate of Return
LCA	Life Cycle Assessment
LCIA	Life Cycle Inventory Analysis
MCDM	Multiple Criteria Decision-Making Methodologies (an umbrella term classifying a number of different types of methodologies)
MIBC	Methyl iso-butyl carbinol
MPA	Maximum Potential Acidity
NAF	Non-Acid Forming
NAG	Net Acid Generation
NAPP	Net Acid Producing Potential
NPV	Net Present Value
pa	Per annum
PAF	Potentially Acid Forming
PAX	potassium amyl xanthate
ROM	Run-of-Mine
SCE	Sequential Chemical Extraction
SETAC	Society for Environmental Toxicology and Chemistry
tpa	Tonnes per annum
UCS	Uniaxial Compressive Strength or Unconfined Compressive Strength
UNEP	United Nations Environment Program
w/c ratio	Water to cement ratio
WP	Wilting point
XRD	X-Ray Diffraction
ZAR	South African Rand

GLOSSARY OF TERMS

Aggregate	a broad category of inert coarse granular materials including sand, gravel, crushed stone, slag, recycled concrete and geosynthetic materials an essential ingredient in concrete with water and Portland cement http://www.cement.org/cement-concrete-basics/concrete-materials/aggregates https://en.wikipedia.org/wiki/Construction_aggregate
Binder	almost any cementing material, either hydrated cement or a product of cement or lime and reactive siliceous materials. The kinds of cement and the curing conditions determine the general type of binder formed. Any material, such as asphalt or resin, which forms the matrix of concretes, mortars, and sanded grouts. Most frequently Portland cement http://www.dictionaryofconstruction.com/definition/binder.html
Bingham Plastic Fluid	a fluid which requires a certain amount of shear stress, called a yield stress, before it starts flowing; after the yield stress is reached the volumetric flow rate increases with an increase in stress https://en.wikipedia.org/wiki/Bingham_plastic
Blast Furnace Slag	obtained by quenching molten iron slag (a by-product of iron and steel-making) from a blast furnace in water or steam, to produce a glassy, granular product that is then dried and ground into a fine powder https://en.wikipedia.org/wiki/Ground_granulated_blast-furnace_slag https://www.fhwa.dot.gov/publications/research/infrastructure/structures/97148/bfs1.cfm
Brunauer, Emmett and Teller (BET) Analysis	technique for the measurement of the specific surface area of materials
Calcium-silica-water gel (C-S-H gel)	the substance that forms when ordinary Portland cement dissolves in water and precipitates again; it covers particles in the cemented material and binds them together to form a cohesive whole
Capacity Factor	ratio of actual output to potential output under continuous operation at peak conditions
Cement Clinker	a powdered material coalesced into solid/porous state by heat or pressure, but without becoming liquid, occurring as nodules in the Portland cement https://en.wikipedia.org/wiki/Sintering
Cement Kiln Dust	a fine, powdery material, portions of which contain some reactive calcium oxide, depending on the location within the dust collection system, the type of operation, the dust collection facility, and the type of fuel used http://www.engr.psu.edu/ce/courses/ce584/concrete/library/materials/Altmaterial/Cement%20Kiln%20Dust.htm
Cemented Paste Backfill (CPB)	a mixture of mine tailings, binder and water placed in a stope in an underground mining operation
Clinker	a sintered material occurring as nodules in the Portland cement
Collector	a chemical used in flotation to make the surface of a particular mineral hydrophobic so that the air bubbles will selectively adhere to that portion of the ore https://www.911metallurgist.com/blog/flotation_collectors
Curing	the process of cement hardening

Depressant	a chemical used in flotation to inhibit the flotation of the portion of the ore which should be retained in the water https://en.wikipedia.org/wiki/Mineral_processing#Froth_Flotation
Dilatant Fluid	See yield dilatant fluid
Fabricated Soil	product based on a bulk particulate material, usually a waste product, containing an appropriate amount of organic carbon, macro- and micronutrients, porosity and texture capable of supporting adequate plant growth through the addition of amendments also known as: artificial soil, manufactured soil, anthroposol, technosol, minesoil
Field Capacity	The amount of soil moisture or water content held in the soil after excess water has drained away and the rate of downward movement has decreased. This usually takes place 2-3 days after rain or irrigation in pervious soils of uniform structure and texture. https://en.wikipedia.org/wiki/Field_capacity
Fly Ash	one of the coal combustion products, composed of the fine particles that are driven out of the boiler with the flue gases https://en.wikipedia.org/wiki/Fly_ash
Frother	a chemical used in flotation to stabilise the bubbles and prevent them coalescing https://en.wikipedia.org/wiki/Mineral_processing#Froth_Flotation
Hydration	The process of cement particle dissolution in water and deposition when the pore water becomes saturated. The process of hydration leads to the deposition of different materials than the original cement components, since these are thermodynamically unstable at ambient temperature.
Life Cycle Assessment (LCA)	a technique to assess environmental impacts associated with all the stages of a product's life from raw material extraction through materials processing, manufacture, distribution, use, repair and maintenance, and disposal or recycling https://en.wikipedia.org/wiki/Life-cycle_assessment#Life_cycle_impact_assessment
Life Cycle Inventory Analysis (LCI)	the data collection portion of LCA, LCI is the straight-forward accounting of everything involved in the "system" of interest; consists of detailed tracking of all the flows in and out of the product system for a define amount of product http://www.athenasmi.org/resources/about-lca/whats-the-difference/ https://www.galvanizeit.org/sustainable-development-and-hot-dip-galvanizing/what-is-sustainable-development/life-cycle-inventory-and-life-cycle-assessment
Life Cycle Inventory Analysis (LCIA)	the "what does it mean" step where the inventory is analysed for environmental impact http://www.athenasmi.org/resources/about-lca/whats-the-difference/ https://www.galvanizeit.org/sustainable-development-and-hot-dip-galvanizing/what-is-sustainable-development/life-cycle-inventory-and-life-cycle-assessment
Mordant	a dye fixative; a substance used to set dyes on fabrics or tissue sections by forming a coordination complex with the dye which then attaches to the fabric or tissue. https://en.wikipedia.org/wiki/Mordant
non-Newtonian fluid	a fluid that does not follow Newton's Law of Viscosity. Most commonly, the viscosity (the measure of a fluid's ability to resist gradual deformation by shear or tensile stresses) of non-Newtonian fluids is dependent on shear rate or shear rate history https://en.wikipedia.org/wiki/Non-Newtonian_fluid
Pozzolan	siliceous or aluminous and siliceous material that has little or no binding properties on its own, but when added to calcium hydroxide and mixed with water it reacts to function as a binder

Pseudoplastic Fluid	See “yield pseudoplastic fluid”
Rheological	relating to the flow of matter, primarily in a liquid state, but also as 'soft solids' or solids under conditions in which they respond with plastic flow rather than deforming elastically in response to an applied force https://en.wikipedia.org/wiki/Rheology
Silica Fume	a byproduct of producing silicon metal or ferrosilicon alloys www.silicafume.org/general-silicafume.html
Sinkholes	collapse of the surface layer of soil to form a depression or hole into deeper underground spaces, may be caused by various natural processes or result from human activity, may be small or hundreds of metres in diameter or depth, usually result in disappearance of surface water https://en.wikipedia.org/wiki/Sinkhole
Sintered	fused together without melting to the point of liquefaction
Slump Test	standard concrete test: measures how much wet cemented material in a slip cone slumps when the cone is removed
Soil Ameliorant	substance or process that improves the physical properties of soil, such as structure, as well as chemical properties, such as pH and metal toxicity, and may improve the available nutrients
Stope	mining void, the empty space left after ore has been removed from the ground
Sulphate Attack	refers to the loss of strength in cemented mixtures associated with exposure to sulphate
Unconfined Compressive Strength (UCS)	The same as uniaxial compressive strength
Uniaxial Compressive Strength (UCS)	The maximum axial compressive stress that an unconfined cylindrical sample of material can withstand before breaking.
Wilting Point (WP)	The minimal point of soil moisture the plant requires not to wilt. If moisture decreases to this or any lower point a plant wilts and can no longer recover its turgidity when placed in a saturated atmosphere for 12 hours. https://en.wikipedia.org/wiki/Permanent_wilting_point
Yield Dilatant Fluid	a fluid for which increased shear stress leads to increased apparent viscosity, also known as “shear thickening” https://en.wikipedia.org/wiki/Dilatant
Yield Pseudoplastic Fluid	a fluid for which increased shear stress leads to reduced apparent viscosity, also known as “shear thinning” https://en.wikipedia.org/wiki/Shear_thinning

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1 INTRODUCTION

1.1 Context of the Project

The primary processing of hard rock ore and coal generates solid waste as unwanted, or gangue, material. The volume of these wastes is dependent on the mineral processed and its grade. In the extraction of metals from hard rock ores, the waste ore generated per unit metal recovered ranges from of the order of 1:1 for iron and aluminium through 200:1 for low grade copper ores to 100 000:1 for gold and platinum (Ayres & Ayres, 1996). In coal processing, the percentage of mined coal ore body reporting as saleable coal is of the order of 35% to 65%, depending on the deposit grade (King, 1980). Appropriate handling of the waste generated is essential to minimise negative environmental impact and to maximise resource productivity. Minimising environmental burden is particularly important where the waste rock carries sulphide mineral, leading to acid rock drainage on exposure to oxygen and water. These acidic conditions, and others, may also increase mobilisation of toxic metals and increase in salinity of associated water resources.

In this research, we focus on the development of sustainable solutions through prevention of acid mine drainage, particularly considering risk removal to provide novel and long term solutions to acid rock drainage, rather than delay of its formation. In this approach, the principle of waste minimization is highlighted and used in conjunction with industrial ecology and the circular economy to re-purpose materials that would otherwise become waste. We aim to achieve this largely through using conventional mineral processing operations.

Increased resource productivity is a key requirement for sustainable development owing to the limited availability of natural capital and the urgent need to stop considering natural capital as a 'free good'. The objective of both a circular economy and the industrial ecology is to maximize the benefit attained from natural resources while minimising associated environmental burden and resource depletion. To minimise resource depletion, complete use of resources mobilised is important, leading to the need to re-purpose waste gangue. To maximise resource efficiency, we must also minimise environmental burden, hence prevent the long-term potential for acid rock drainage generation, preferentially through risk removal.

While our initial proof of concept of the potential to remove ARD risk through sulphide separation and to re-purpose the resultant waste fractions and recovered fraction of value focussed on both hard rock and coal wastes (Harrison et al. 2013), in this project we focus on sulphur-bearing wastes from coal mining in South Africa. These coal wastes contain significant energetic value due to the presence of residual coal, as well other potentially useable minerals such as pyrite, kaolinite and limestone (Amaral Filho, et al., 2013; North, et al., 2015). Apart from a loss of mined resources when disposed of as wastes, the disposal of these wastes poses a significant risk to the surrounding environment, particularly through the generation of dust and acid rock drainage (ARD) arising from the oxidative dissolution of pyrite and other sulphidic minerals (Kontopoulos, 1998), as well as contributing negatively to land-use.

To remove the risk of ARD generation from these coal wastes, removal of the sulphidic fraction from the waste material by separation or reaction is desirable, thereby removing the ARD generation risk. On achieving a sulphide, or sulphate, rich stream (from separation or reaction respectively) and a benign tailings fraction, potential exists for their re-purposing. Improved separation for recovery of value together with re-using the waste is preferred over the current typical disposal to waste rock and tailings dumps. A zero waste strategy is the ideal. An increasing desire to avoid liabilities on mine closure as well as a move towards enhanced resource efficiency provides motivation to develop waste management approaches that remove risks in perpetuity, whilst simultaneously creating opportunities for value recovery and re-purposing of so-called wastes as feedstock for other uses.

1.2 Acid Rock Drainage

ARD is well recognised as a significant problem in South Africa (The Council for Geoscience, 2010; Feris & Kotzé, 2014). Oxidation of sulphidic minerals results in production of acid and sulphate which cause increased salinity and acidity of water run-off, leachate, rebound water and water bodies. These acidic conditions, rich in dissolved iron and sulphuric acid also augment continued sulphide leaching and the mobilization of metals, with concomitant contamination of the environment. Following mining of sulphide-bearing ores, the generation of ARD often becomes a long-lived problem, extending for tens or hundreds of years after mine closure. This highlights the importance of developing processes for handling and disposal of waste materials in order to prevent, or at least minimise, the generation of ARD. This study focuses on ARD prevention from sulphide-bearing coal wastes, with associated re-purposing of the waste materials.

Wastes from the extraction and beneficiation of sulphide-bearing coal deposits may contain significant quantities of sulphide minerals, particularly pyrite (FeS_2). ARD generation results when these sulfidic minerals react with oxygen and water, releasing iron and sulfuric acid into solution (Rohwerder, et al., 2003; McCarthy, 2011). In the presence of acid, acidophilic microorganisms catalyse the oxidation of Fe^{2+} ions, producing Fe^{3+} ions which serve as leach agents to enhance the leaching process. Biologically-mediated ferric iron leaching occurs much faster than chemical leaching (Rohwerder, et al., 2003; McCarthy & Pretorius, 2009; Parker & Robertson, 1999). Similarly, microbially catalysed oxidation of the reduced sulphur causes acidity and dissolved sulphate. ARD drives metal mobility. Further it degrades water resources and agricultural land due to the acidity and salinity of resulting mine waters and their potential metal content (McCarthy & Pretorius, 2009).

Formation of ARD requires the reaction of sulphide, water and oxygen and is catalysed by microorganisms. Hence, the long-term prevention of its formation requires the removal of these components from the reaction surface. Most typically, exclusion of oxygen and water is sought; however, this is required on an ongoing basis into perpetuity. Conversely, removal of sulphide presents a permanent solution in which the risk of ARD formation is removed completely. This is the approach proposed in this report, with subsequent re-purposing.

1.3 South African Coal Waste

South Africa has a generous, yet limited, supply of readily extractable coal in Limpopo, Mpumalanga, Gauteng and the Free State, with depleted reserves in KwaZulu-Natal and the Eastern Cape provinces (Hancox & Götz, 2014; Eberhard, 2011; Jeffrey, 2005). Most of the commercially mineable resource is contained in the Permian-aged Vryheid formations of the Ecca Group which have been found to be rich in both inertinite and vitrinite (Hancox & Götz, 2014; Snyman & Botha, 1993; Hutton & Mandile, 1996).

The run-of-mine (ROM) coal contains combustible coal, gangue minerals, impurities and unwanted mining fragments. In order to meet the market quality specifications, the ROM coal is usually beneficiated, resulting in separation of saleable coal from the unwanted or waste material. In a typical South African processing plant, coal wastes are disposed in dump deposits and dams as discards and fine coal slurry. Coal discards are residues generated from the washing of crushed coal using dense medium and gravity separation techniques, whilst fine coal or slurry waste is typically a mixture of fine (-500 μm to +150 μm) to ultrafine (-150 μm) particulates (Horstall, et al., 1986) including coal, inorganic minerals and process water that is purged from the coal processing plant. Analysis of the particle size distribution of South African coal slurry wastes shows that the samples present a D_{50} lower than 250 μm (Iroala, 2014; Kazadi Mbamba, 2011), indicating suitability to be processed by flotation (Kawatra, 1995).

Despite having a relatively high calorific value and significant potential for coal recovery (Reddick, 2006), historically the ultrafine coal slurry is discarded as a waste because it is expensive to dewater. Industrial figures estimate that the total local industry-wide coal waste produced makes up 27.7% of

ROM coal (Chamber of Mines of South Africa, 2007). The National Inventory of Discard and Duff Coal of 2001 (DME, 2001) estimated the coal slurry produced to account for 21% of total coal waste. Based on this information, it has been estimated that between 14 Mt and 18 Mt of coal slurry was discarded in 2006 in the South African coal mining industry (Reddick, 2006). These figures are likely to have increased since then, due to increased coal production (Chamber of Mines of South Africa, 2014) and declining ore grades. Apart from representing a loss of revenue with respect to the coal content, provision for slurry deposits has been estimated to contribute 3% to the capital cost of establishing an open pit coal mine, equivalent to 10% of the capital cost of the processing plant (Mohutsiwa & Musingwini, 2015).

Both coal discards and coal slurry waste typically contain pyrite, creating an ARD risk. Despite the relatively low sulphur content (1.1 to 2.3%) of the coal slurries studied by both Kazadi Mbamba, et al. (2012) and Kotelo (2013), these coal slurries were acid generating. In South African coal fields, ARD is commonplace and presents a significant problem (Bell, et al., 2001; McCarthy & Pretorius, 2009), with spoil heaps not only generating ARD but also forming blemishes on the landscape and resulting in air pollution from spontaneous combustion (Bell, et al., 2001). Prediction, prevention and containment of ARD is of primary concern in waste disposal sites at both active and abandoned mines. In this report we consider prevention of ARD by sulphide separation with associated re-purposing of the coal waste fractions. In the companion report (Part A), we address characterisation of SRD generation potential.

1.4 The Two-Stage Flotation Process for Prevention of ARD

1.4.1 The two-stage separation process

To recover value and reduce the ARD risk associated with sulfidic mineral tailings, researchers at the University of Cape Town have developed a two-stage separation process (Hesketh, et al., 2010; Kazadi Mbamba, et al., 2012; Harrison, et al., 2013; Iroala, 2014) which, using physical separation techniques such as flotation or reflux classification, separates a fine sulfidic (coal) mine waste stream into a recovered valuable product stream, a concentrated sulphide-rich stream which is typically acid generating and a non-acid generating sulphide-lean stream which is relatively benign. This process has been shown to be effective with fine coal waste and hard rock or base metal tailings. The ongoing focus of this study is on fine coal waste.

The sulphide-rich stream has a reduced volume and a concentrated sulphide content, while the bulk waste volume resides in the sulphide-lean stream. In the case of coal waste rich in coal, a considerable portion of the original volume may be contained in the third fraction, a saleable coal fraction. The technological feasibility of this approach of complete removal of the risk of ARD formation from mine waste fractions by sulphide separation has been demonstrated (Harrison, et al., 2010; Harrison, et al., 2013; Hesketh, et al., 2010; Kazadi Mbamba, et al., 2012; Kazadi Mbamba, et al., 2013). It is apparent from these studies that the prevention of ARD from the coal fines waste fraction by sulphide removal has good potential. The two-stage flotation circuit, which generates a saleable product stream and two streams with valorisation potential, enables mines to enhance resource productivity and potentially utilise, or re-purpose, the waste (Kazadi Mbamba, et al., 2012; Castelo-Branco, et al., 1999), while removing the long term risk of ARD generation potential.

To further validate the implementation of this approach to minimising the risk of ARD formation from the fine coal mine wastes, rigorous environmental and techno-economic analysis is valuable. An environmental impact study was conducted on the treatment of fine coal waste from South African coal processing (Chapter 1). The case study uses a Life Cycle Inventory (LCI) followed by a Life Cycle Impact Assessment (LCIA). This is followed by a detailed techno-economic assessment (Chapter 3) of this separation process for fine coal wastes. The assessment is based on previously reported experimental work of the Centre for Bioprocess Engineering Research (CeBER) at UCT. It includes criteria arising from two-stage flotation of four different coal wastes, with four different reagent profiles.

1.4.2 Applications of the sulphide-rich and sulphide-lean streams

Initially the desulphurisation of fine coal slurries was treated as an end-of-pipe treatment; however, it is recognised that the integration of the treatment step into the value recovery process is ideal as decisions can be made to optimise value recovery and minimise burden in an integrated manner, producing a bulk fraction of benign tailings and a small fraction of sulphide-rich material for containment. This aids to mitigate the negative effect of the mine waste through risk removal of the sulphide-containing fraction, thereby reducing or avoiding ARD generation potential. To some extent, it is possible to partially valorise the waste stream by sale of the coal concentrate. Further, re-purposing of the benign and sulphide rich fractions diminishes the impact of the bulk stream by condensing volume and removing acid-producing potential.

Further, in this project we further improve resource efficiency by both decreasing burden and increasing value, the latter through developing a combined industrial metabolism and circular-economy approach. The foremost objective is to eliminate the risk of ARD entirely by desulphurisation of the fine coal slurry to produce a sulphide-rich stream. This stream is either disposed of with complete containment or re-purposed for appropriate value generation. Similarly, re-purposing of the sulphide-lean fraction may enhance resource efficiency. Removing the environmental and social impact of stock piling or dumping of both the sulphide-lean and sulphide-rich waste through re-purposing is desired. The significant additional benefit inherent in the two “secondary” output streams is recognised.

On desiring to use all exit streams as (potential) product streams, realistic options for the use of both the sulphide-rich concentrate and benign (sulphide-lean) tailings fractions are required and addressed in Chapters 4 and 5. Harrison et al. (2013) reviewed the potential opportunities for use of these streams. The sulphide-rich stream should be applied where additional acid is an advantage or as a source of sulphur- and iron-containing products. The sulphide-lean stream can be used in applications where a neutral or even alkaline stream is required; however, it is mainly be used for its particulate material nature. As examples, the sulphide-rich stream can, for instance, be used in ferrous sulphate production (Vigânico, et al., 2011) and mine backfill (Benzaazoua, et al., 2000), while the sulphide-lean material could be used as a construction material (Argane, et al., 2016) or as covers for waste rock dumps (Kotsiopoulos & Harrison, 2015).

1.4.3 The approach of this project

The approach of separating the sulphide fraction from the bulk fine coal slurry to avoid environmental pollution in the form of ARD and re-purposing components of the ‘waste’ streams to provide new product components, underpinned by industrial ecology forms the central thrust of this report. The separation process is positioned in a holistic framework based on the move towards zero waste and zero harm. Complete resource use enables resource efficiency, while striving for zero waste ideals. The separation, and removal, of ARD-forming sulphides drives towards the zero impact disposal principles, necessitating the prevention of ARD. An inclusive decision-making framework is embedded into the methodology adopted, using multiple criteria and accessing broad-based literature review, local consultation and intensive primary context-specific research to arrive at an informed, comprehensive, contextual evaluation of desulphurisation of coal wastes using the two-stage flotation (or equivalent) separation process, followed by options for use or processing or both of all three exit streams for the elimination of negative social and environmental impact of the unprocessed waste, together with an optimal resource recovery opportunity.

2 ENVIRONMENTAL ASSESSMENT OF THE IMPACT OF SULPHIDE SEPARATION BY FLOTATION

Also presented as:

Fundikwa B.M., Broadhurst J.L. and Harrison S.T.L. (2019). *An environmental assessment of the use of froth flotation for recovery and sulfur removal from ultrafine coal waste in South Africa. Minerals Engineering, in press.*

2.1 Introduction to Environmental Assessment Case Study

There are a number of potential environmental risks and liabilities associated with the direct disposal of untreated coal processing wastes into slurry dams, the key ones being the generation of acid rock drainage (ARD), spontaneous combustion, land transformation, dust emissions, and the metal contamination and salinisation of groundwater sources (Bell, et al., 2001; Kotelo, 2013; Sonjica, 2010; Pone, et al., 2007). Of particular concern in the South African coal mining context, is the extensive acid drainage pollution in the Olifants River catchment area arising from the long-term oxidation of pyritic sulphur, typically present in geological coal deposits and processing waste dumps (Mangena & Brent, 2006; Karadad-Nelson, 2010; Sonjica, 2010). Furthermore, coal processing waste dumps typically contain significant quantities of potentially useable coal, thus representing a loss of mined resources. This is particularly the case for the ultrafine slurry waste streams from South African collieries, which typically have a calorific value of > 20 MJ/kg (DME, 2001).

The two-stage flotation process, developed at University of Cape Town (UCT) and illustrated diagrammatically in Figure 2-1, is aimed at recovering coal from fine coal processing waste in the form of a saleable product, whilst simultaneously removing acid forming pyritic sulphur, thereby effectively mitigating the risk of ARD formation (Broadhurst & Harrison, 2015; Kazadi Mbamba, et al., 2012; Harrison, et al., 2013; Harrison, et al., 2010).

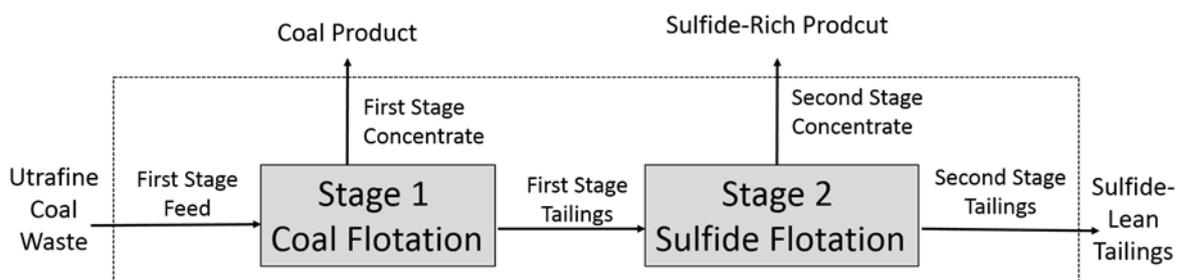


Figure 2-1 Proposed two-stage flotation process for the processing of fine coal waste

It is furthermore proposed that the sulphide-lean and sulphide-rich streams arising from this process be re-purposed as feed for other uses, thereby further reducing the waste burden and enhancing overall resource efficiency of the coal processing sector (Amaral Filho, et al., 2016; Broadhurst & Harrison, 2015; Harrison, et al., 2013). The technical feasibility of the two-stage flotation process has been established on a laboratory-scale using a number of fine coal wastes from various sources. Results of these studies have shown that it is possible to recover as much as 85% of the coal and remove up to 60% of the sulphide sulphur, effectively generating a non-acid forming sulphide-lean stream in all cases (Broadhurst & Harrison, 2015; Kazadi Mbamba, et al., 2012). Based on this laboratory testwork, Jera (2013) established a framework for an order-of-magnitude cost estimate ($\pm 30\%$ to $\pm 50\%$ accuracy) for a fictitious two-stage coal flotation desulphurisation plant to treat nominal coal fines from a dump of an abandoned mine in the Witbank/Middleburg coal field (Jera, 2013). This study showed the potential

economic viability of the desulphurisation process for removing ARD risks, when combined with value recovery.

To date studies on the environmental implications of the two-stage flotation process for the treatment of fine coal waste have been limited to ARD generating potential, with little consideration having been given to broader environmental consequences. As indicated in a previous base-metal sulphide tailings case study (Broadhurst, et al., 2015), desulphurisation flotation for the removal of acid rock drainage risks has implications in terms of human and eco-toxicity, global warming and terrestrial acidification, land use, as well as the depletion of non-renewable resources such as water and minerals. Although life cycle assessment (LCA) provides a potentially suitable tool for evaluating the systemic environmental benefits and burdens of process interventions such as the two-stage flotation process, the study by Broadhurst et al. (2015) showed that standard LCA models are still deficient and inconsistent in terms of their ability to adequately and reliably assess the impacts associated with solid waste deposits. This pertains in particular to water-related impacts, including aqueous acidification, salinisation and heavy metal contamination. Recognition of these deficiencies has prompted the development of additional indicators and new impact assessment methods to address some of the limitations associated with current LCA tools, whilst still maintaining the systemic nature and systematic procedural framework of the LCA approach (Hermann, et al., 2007; Leske & Buckley, 2004; Ridoutt & Pfister, 2013). The study by Broadhurst et al. (2015) also highlighted the need for more reliable model input data, particularly in terms of electricity consumption and mobility of tailings components.

This study uses a combination of life cycle and risk-based assessment tools to evaluate the broader environmental consequences of the proposed two-stage flotation process for the case of an ultrafine coal waste slurry sample from the Waterberg region in South Africa.

2.2 Methodology

2.2.1 Case study scenarios

Two scenarios were developed in this study (Figure 2-2). In the first scenario, termed the base case, coal ultrafine slurry generated from the coal beneficiation process is fed into a thickener, where excess water is recovered and recycled back to the coal processing plant, together with dam return water, and the underflow slurry is disposed of in a slurry dam. In the second scenario, the two-stage flotation process, the coal ultrafine slurry is initially passed through the first froth flotation stage, where a coal-rich concentrate is recovered and dewatered by means of a filter press. The tailings stream from the first flotation stage is then fed to the second froth flotation stage, where a sulphide-rich concentrate is recovered and dewatered, also by means of a filter press. Lastly, the tailings from the second froth flotation stage are dewatered in a thickener and filter press, before being disposed of in a tailings impoundment. As in the base-case scenario, all water from the dewatering processes is recycled back to the coal beneficiation plant together with slurry dam return water. In this case study, it is assumed that the coal and sulphide-rich concentrates can be sold as products for downstream use, for example as a source of electricity and in sulphuric acid production respectively.

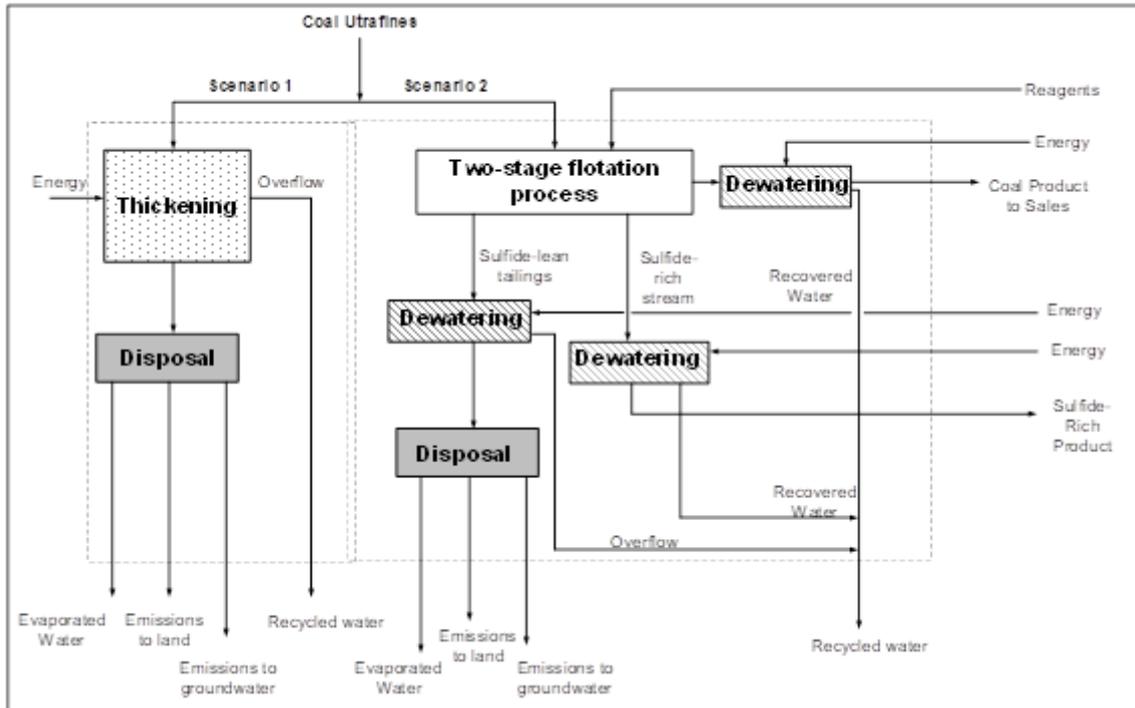


Figure 2-2 Coal ultrafine slurry management scenarios

In both cases, dissipative water loss occurs through evaporation, seepage and entrainment during disposal of the final tailings streams. Seepage water contains acid, metals and salts mobilised through the reaction of the ash-forming minerals in the tailings, as well as residual flotation chemicals in the case of the second (two-stage flotation process) scenario. In this way the tailings deposit is considered as a unit reactor having its own emissions and residue (unreacted tailings).

2.2.2 Inventory analysis

The overall system boundary for the two-stage flotation scenario is outlined in Figure 2-3. In both scenarios, the system boundary includes the production of electricity and flotation reagents as background processes, with the foreground processes (see Figure 2-1), including dewatering, two-stage flotation (scenario 2 only) and tailings disposal. Downstream utilisation of the coal and sulphide-rich product streams were not included, i.e. the system is “cradle-to-gate”. The functional unit used in this project was 100 t/hr coal ultrafine waste (dry basis), at a solids content of 10%. This is consistent with the conceptual flowsheet established by Jera (2013), with a capacity of 720,000 tons per annum at a plant life and availability of 15 years and 82% respectively.

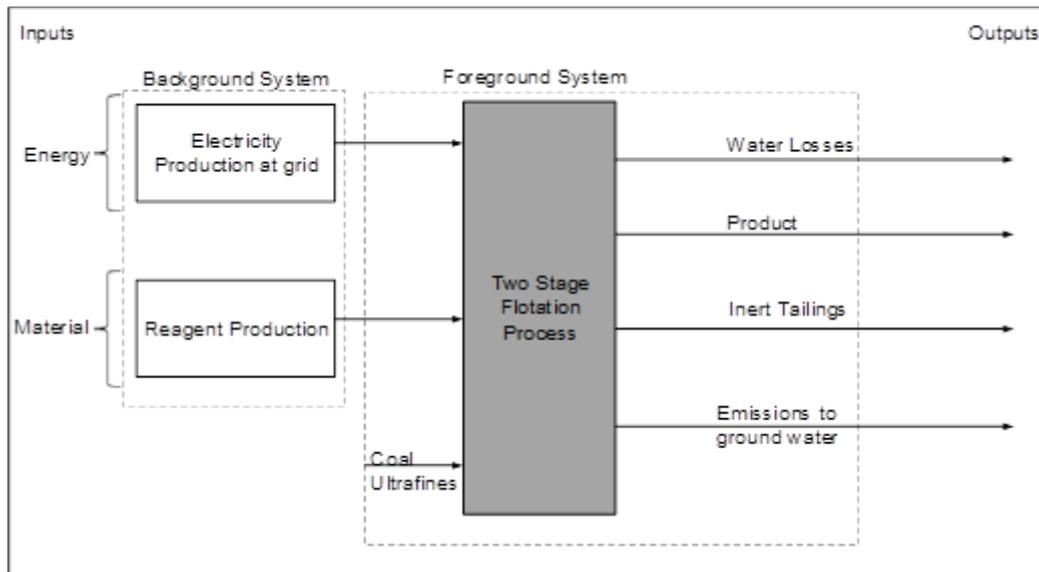


Figure 2-3 System boundary for the two-stage flotation process (scenario 2)

Data for the foreground system was derived from in-house laboratory-scale testwork conducted by Iroala (2014) on a sample of coal ultrafine slurry from the Waterberg coal fields in South Africa. Table 2-1 summarises the key experimental data for the two-stage flotation process (scenario 2).

Table 2-1 Laboratory-scale data for the two-stage flotation process (Iroala, 2014)¹

	Feed	Coal concentrate	Sulphide-rich concentrate	Sulphide-lean tailings
Total Solids Department (%)	100	30.2	2	68
Sulphide sulphur ²	1.07	1.14	12.8	0.69
Sulphates	0.0280	0.03	0.36	0.02
Stream Composition (%)				
Organic sulphur	0.822	1.50	15.1	0.10
Total sulphur	1.92	2.66	28.3	0.81
Ash	49.1	24.3	49.1	62.29
Fixed carbon and volatile matter ²	49.0	73.0	22.6	36.90
NAPP (kg/t H ₂ SO ₄) ³	52	-	532	-101
<ol style="list-style-type: none"> 1. Flotation conducted at the following reagent dosages: 1.4 kg/t of Nalflote 9858 (coal collector in stage 1); 0.28 kg/t of methyl isobutyl carbinol (frother in stages 1 and 2); 2.33 kg/t of sodium isobutyl xanthate (sulphur collector in stage 2); and 0.93 kg/t of Dextrin (coal suppressant in stage 2). 2. Both the recovery of coal (30.2%) and removal of sulphide sulphur (25%) are relatively low in comparison to previous case studies with other coal wastes (Kazadi Mbamba et al. 2012 and 2013; Broadhurst and Harrison, 2015). 3. Based on the standard Acid Base Accounting (ABA) test (Smart et al., 2002). In accordance with the results of this test the feed and the sulfide-rich concentrate are acid forming and the sulphide-lean tailings non-acid forming. 				

Tailings from both scenarios were subjected to leaching under test conditions consistent with the net acid generation (NAG) test for ARD prediction (Smart et al., 2002), and the leachates submitted for analysis to determine the concentration of potentially “available” metals under disposal conditions.

Although Nalflote 9858 was used as a coal collector in the laboratory-scale case study by Iroala (2014), for the purpose of this study, oleic acid (another commonly used coal flotation reagent) was used as a proxy for Nalflote 9858. This was due to the lack of information in terms of the chemical compositions and production inputs and outputs for the Nalflote reagent. Background data for coal-based electricity production and oleic acid was derived from the Ecoinvent database (version 2.2), whilst that for SIBX (sodium isobutyl xanthate) was derived from a recent in-house LCA study of the local production of this reagent at Senmin’s International’s production facility in Sasolburg, South Africa (Kunene, 2014).

2.2.3 Impact assessment

The inventory data was translated into meaningful environmental indicators, in accordance with relevant LCA impact categories (Table 2-2).

Table 2-2 LCA-based impact categories and characterisation factors

Impact category	Characterisation Factor	Indicator
Climate Change	Global Warming Potential for each greenhouse gas (kg CO ₂ /kg emission) ¹	kg CO ₂ equivalents
Terrestrial Acidification	Acidification Potential (AP) for each emission to air (kg SO ₂ /kg emission) ¹	kg SO ₂ equivalents
Fossil Fuel Depletion	Fossil Fuel Depletion Potential for each extraction of fossil fuels ¹	kg of oil equivalents
Water Footprint	Stress weighted Global Consumptive Water Footprint ²	m ³ /year
Land Use	Characterization factors for land occupation/land transformation ¹	m ² /year
Salinity	Total Salinity Potential for a groundwater ³	Total Dissolved Salts kg equivalents
Eco-Toxicity ⁴	Eco-toxicity potentials for a freshwater ¹	kg 1,4 Dichlorobenzene equivalents

1. Taken from Goedkoop et al. (2015)
 2. Taken from Pfister et al. (2009)
 3. Taken from Leske and Buckley (2004)
 4. Excludes iron and sulphate due to lack of characterisation factors

The ranking and scoring protocol, developed by Broadhurst and Petrie (2010), was also used to identify the tailings components of potential significance in terms of their hazardous nature and chemical behaviour. This protocol entails the calculation of a risk potential factor in accordance with Equation 2-1.

$$RPF_i = \frac{(AC_i)^2}{ARC_i \times BC_i} \quad \text{Equation 2-1}$$

Where: RPF_i is the risk potential factor for constituent i ; AC_i is the “environmentally” available concentration (mg/kg); ARC_i is the acceptable risk concentration (mg/kg) and BC_i is the background concentration

In this study, leachable concentrations, under conditions consistent with the NAG test described by Smart et al. (Smart, et al., 2002), were used as a proxy for available concentrations (AC_i); average crustal abundance values (Broadhurst & Petrie, 2010) as a proxy for background concentrations (BC_i); and Australian drinking water quality guidelines (Australian Government, 2016) as a proxy for acceptable risk concentrations (ARC_i). Risk potential factors for each of the individual metals were summated to provide a net indication of potential environmental risk associated with the release of soluble metals from the tailings impoundments. Vanadium and titanium were excluded from this assessment, due to the lack of drinking water quality guidelines.

2.3 Results

2.3.1 Inventory analysis

Table 2-3 summarises the key outputs for the two coal ultrafine waste management scenarios, taking into account both the foreground and background processes.

Table 2-3 Summary of key inventory data for the two coal waste treatment scenarios

Description	Units	Flows per 100 tons of ultrafine coal waste			
		Base case	Two stage flotation		
Environmental Emissions	Acid (H ⁺)	kg	106	0.005	
	Sulfate (S)	kg	2640	1153	
	Ground water Emissions	Iron (Fe)	kg	148.1	83.9
		Aluminium (Al)	kg	18.5	13.1
		Titanium (Ti)	kg	9.54	5.73
		Trace metals	kg	6.20	3.33
	Xanthate	kg	0	0.51	
	Air Emissions	Carbon dioxide	kg	12	2597
		Sulfur dioxide	kg	0.08	27.8
	Land Emissions	Solid waste	tons	100	68
Non-recoverable resources	water	m ³	112.5	27.2	
	Coal	t	47.0	27.5	
	Natural Gas	kg	0	62.7	

Key assumptions and calculations:

- Acid generation is calculated in accordance with the NAPP values obtained from the Acid Base Accounting predictive test
- Available metal concentrations were derived from leach tests consistent with NAG tests, with trace metals including Zn, Cr, Ba, V, Mn, Ni, Cu, As, Pb and U.
- 80% of the non-organic sulfur in the tailings is mobilised as soluble sulfate under disposal conditions (Broadhurst and Petrie, 2010).
- 97% of the xanthate feed reports to the concentrate during flotation. The remaining 3% departs to the tailings storage facility as soluble xanthate, and is partially emitted to the environment during disposal.
- The solids content in the tailings thickener underflow is 60% and that of the filtered concentrate, 90%. Distribution of thickener underflow water is as follows: 25% dam return water; 40% evaporation; 30% entrainment; 5% seepage (after Bleiwas, 2012).

A comparison of the inventory data indicates that, relative to the base case scenario, the two-stage flotation process results in a significant decrease in total solid waste (31.9%) and water-related emissions, including acid (>99%), sulphate (56.3%), iron (43.3%), aluminium (29.2%) and trace metals (46.2%). The two stage flotation does, however, result in an increase in atmospheric emissions of sulphur dioxide and carbon dioxide in background processes. Carbon dioxide emissions can be mainly attributed to the use of fossil-fuel based electricity in the flotation process (81.3% of total), with smaller contributions from the production of xanthate (16.5%) and oleic acid (2.2%). Sulphur dioxide emissions associated with the two stage flotation process can be attributed to both fossil-fuel based electricity (49.6%) and xanthate (50.4%) production. First-order calculations of energy consumption have indicated that the two stage flotation scenario uses 1760 kWh of energy per 100 tons of ultrafine coal waste, in comparison with 30.4 kWh for the base case scenario. Energy consumption in the two stage flotation process can, furthermore, be attributed mainly to the flotation reactors (54%) and air compressors (26%). Relative emissions are illustrated diagrammatically in Figure 2-4.

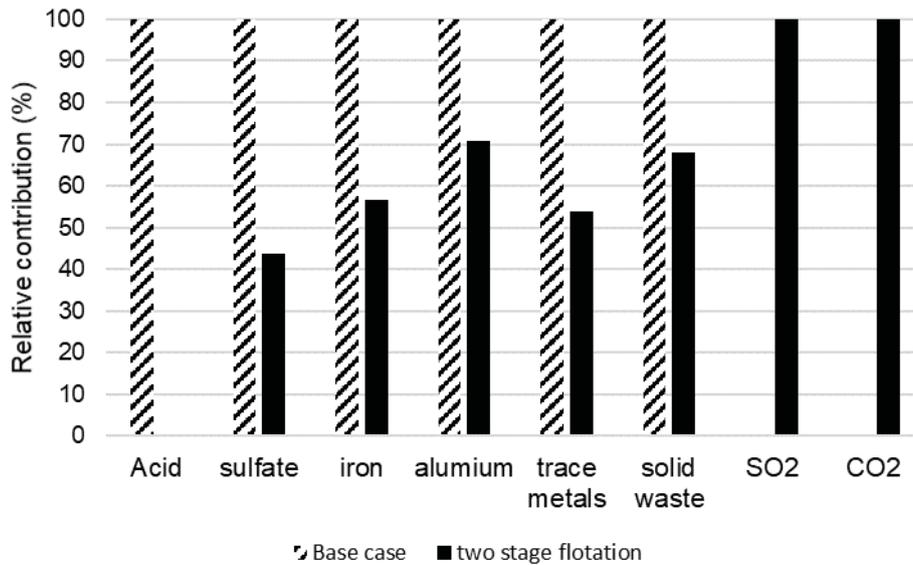


Figure 2-4 Relative emissions for the two scenarios for management of coal ultrafine waste

The results in Table 2-3 also indicate that the two-stage flotation process results in a significant decrease in the loss of non-recoverable resources, including water (75.4%) and coal (41.6%). Water consumption can be attributed largely to dissipative water losses during tailings disposal (>99.9% and 56.3% in the case of the base case and two-stage flotation scenarios respectively), although entrainment of water in product streams also contributes significantly to the net water consumption in the case of the two-stage flotation scenario (33.2%). The significant difference in water loss between the two scenarios is, however, mainly due to the incorporation of filtration units in the two-stage flotation process flowsheets. This results in an increase in the solids content of the final tailings from 40% to 70%, and a process water recovery of 97% compared to that of 87% for the base case scenario, which employs only a conventional thickener for tailings dewatering. The addition of a filtration circuit in the base case scenario would reduce water losses by more than 70%, resulting in a net loss in water resources of only 14% more than the two-stage flotation scenario. Similarly, coal losses occur mainly as a result of the land disposal of coal-bearing tailings, although in the case of the two-stage flotation scenario, coal consumption can also be partially attributed to the production of electricity (approximately 10% of total coal consumption).

2.3.2 Impact assessment

As indicated in Table 2-2, the coal waste management scenarios were evaluated in accordance with a number of potentially relevant LCA-based impact categories, including those that are related to resource consumption (fossil fuel depletion, natural land transformation, consumptive water footprint) and those related to environmental emissions (climate change, terrestrial acidification, aqueous salinisation and eco-toxicity). In addition, potential risk factors were used as a measure of the potential risks associated with the leaching of metals and sulphate from the final tailings under disposal conditions.

2.3.2.1 Resource consumption impacts

The relative resource consumption-related impacts for the two coal waste management scenarios are presented in Figure 2-5. Relative to the base case scenario, the two-stage flotation process results in a decrease in the transformation of land, depletion of fossil fuel and, in particular, the water footprint. Consistent with the inventory results, the resource consumption-related impacts can be largely attributed to the waste management (foreground) processes, and more specifically the land disposal of final tailings. However, in the case of the two-stage flotation process, generation of fossil-fuel based electricity and production of flotation reagents also make some contribution (7% and 11% respectively)

to fossil fuel depletion. As indicated previously, the water footprint of the base case scenario could be reduced quite considerably by dewatering the final tailings in a filter press.

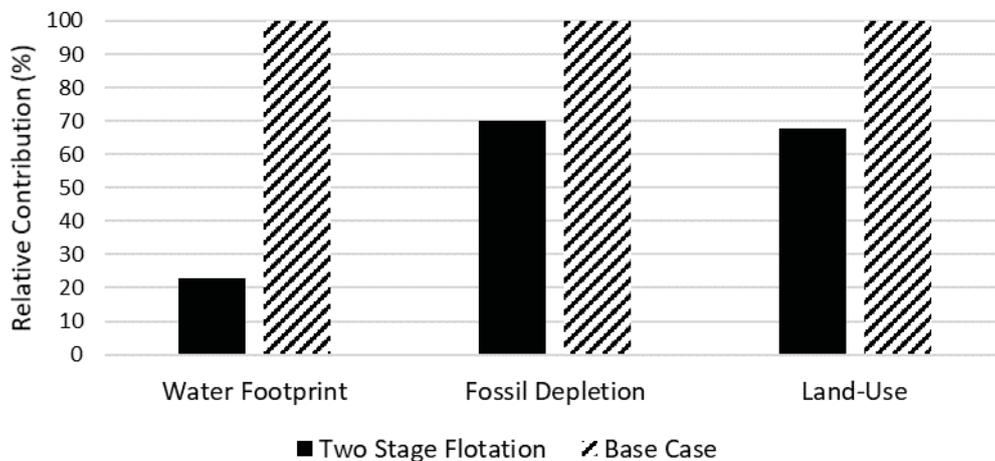


Figure 2-5 Relative resource consumption impacts for the two coal disposal scenarios

2.3.2.2 Environmental emission impacts

Figure 2-6 shows that the two stage flotation process also results in a significant decrease in the water-related salinisation (60%) and eco-toxicity (54%) impacts, as well as potential metal risk factors (29.6%). These impacts are attributed almost exclusively to land disposal of the final process tailings. A more detailed analysis indicates that eco-toxicity impacts are due largely to vanadium, copper, nickel, zinc and manganese (Figure 2-7), with vanadium having a particularly high characterisation factor (Goedkoop et al., 2009). No characterisation factors were available for iron, which, in accordance with the risk potential factors, was the only metal to have a sufficiently high leachable concentration levels to pose a risk, albeit a low one ($RPF/1000 < 1$). The low ($RPF/1000 = 0.1-1$) to negligible ($RPF/1000 < 0.1$) risk posed by the metals in the final tailings from both management processes (Table 2-4) can be attributed, at least partially, to their low solubility at the circum-neutral pH values (pH 6-7) of the leach solutions obtained under NAG test conditions. Of greater environmental significance is the potential release of soluble sulphate under disposal conditions. In accordance with the risk potential classification protocol proposed by Broadhurst and Petrie (2010), sulphate poses a low risk in the tailings from the two-stage flotation process and a moderate risk in the tailings arising from the base case scenario.

Table 2-4 Risk potential factors for tailings constituents

Component	Risk Potential Factors (RPF)	
	Base case	Two-stage flotation
Sulphate	1507	760
Iron (Fe)	146	102
Aluminium (Al)	2.1	2.3
Arsenic (As)	0.03	0.03
Lead, zinc, chromium, manganese, cobalt, nickel, copper, vanadium, barium, uranium	<0.01	0.01

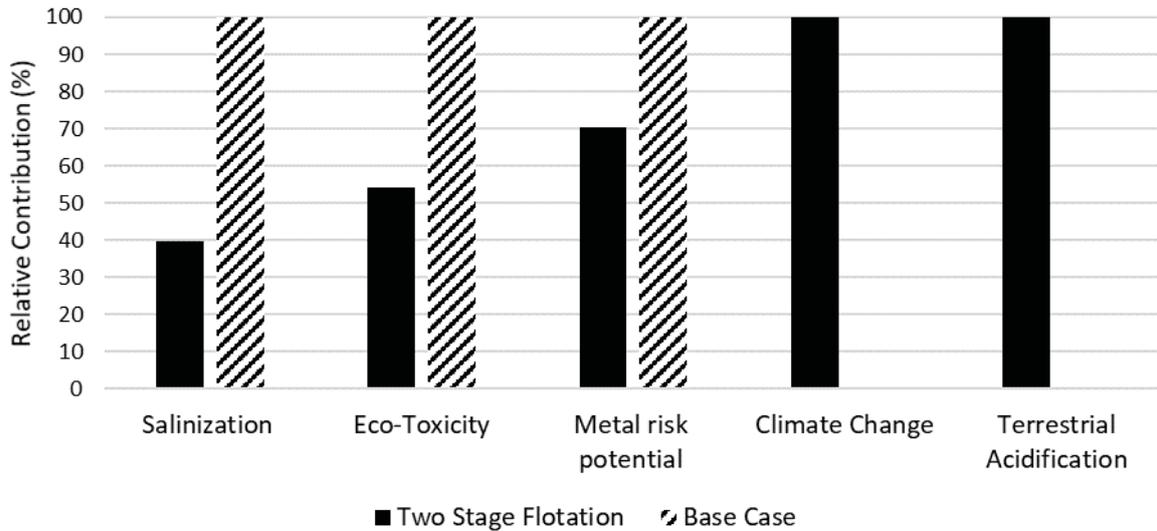


Figure 2-6 Relative environmental emission impacts for the two coal waste management scenarios

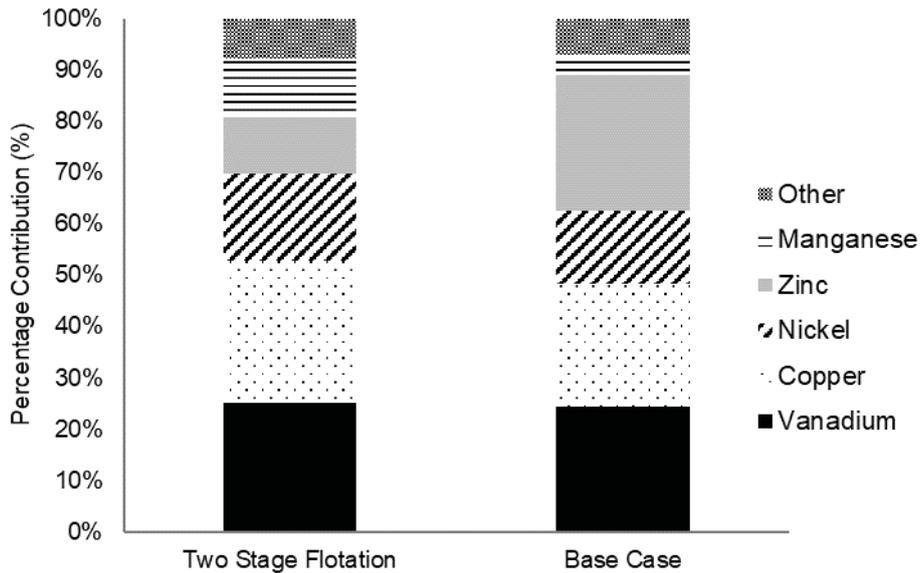


Figure 2-7 Metal contributions to eco-toxicity impacts

Despite the positive effect in terms of water-related impacts, results in Figure 2-6 indicate that the two stage flotation process results in a significant increase in climate change and terrestrial acidification impacts. Consistent with the inventory results, these impacts can be attributed exclusively to SO₂ and CO₂ emissions arising from background production of electricity (88% and 58% of climate change and terrestrial acidification impacts respectively) and, to a lesser extent, flotation reagents (12% and 42% of the climate change and acidification impacts respectively).

2.4 Interpretation and discussion

2.4.1 Sensitivity analyses

A sensitivity analysis was conducted to demonstrate the effect of key process variable on the relevant impact categories for the two-stage flotation scenario.

2.4.1.1 Impact of electricity consumption on climate change and terrestrial acidification

The results of the impact assessment indicated that climate change and terrestrial acidification impacts can be largely attributed to the consumption of fossil-fuel based electricity. A sensitivity analysis (Figure 2-8) confirms the significance of electricity consumption in terms of environmental performance, with a 43% decrease in electricity consumption (equivalent to 1 MWh/100 t of coal ultrafine coal tailings) resulting in a decrease of 38% and 26% in climate change and aqueous acidification impacts respectively.

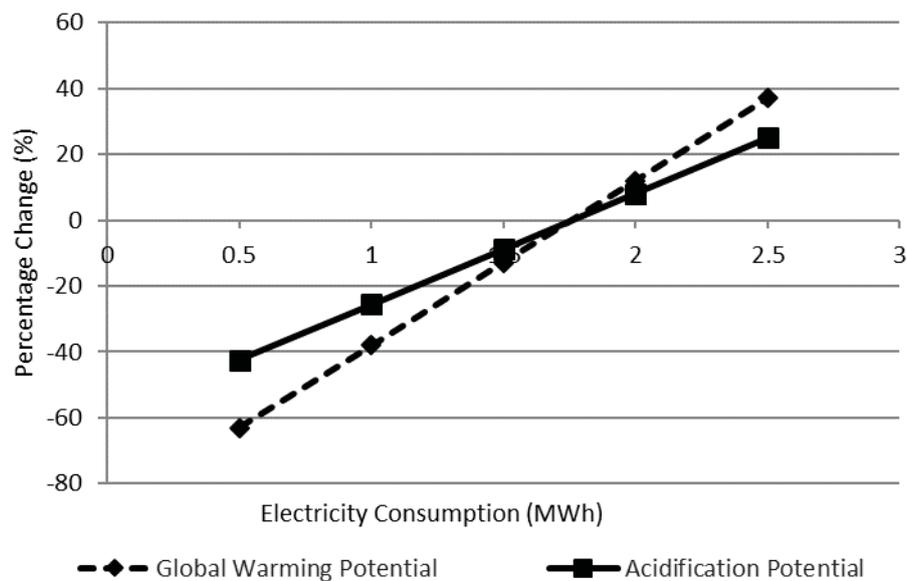


Figure 2-8 Effect of varying electricity consumption on climate change, terrestrial acidification and fossil fuel depletion for the toe-stage flotation scenario

2.4.1.2 Impact of coal yield on land use and fossil fuel depletion

The coal yield during flotation (30.2%) for the selected case study was low in comparison to previous studies (up to 86%). The results in Figure 2-9 indicate that coal yield has a significant effect on fossil fuel depletion and, to a lesser extent, land transformation. Increasing the coal yield to 55% (82% increase) will result in a more than 91% decrease in fossil fuel depletion and 36% decrease in natural land transformation impacts. At a coal flotation yield of <10%, the fossil fuel depletion impact for the two-stage flotation scenario will become higher than that for the base case scenario, due to the consumption of coal in the production of electricity consumed in the flotation circuit.

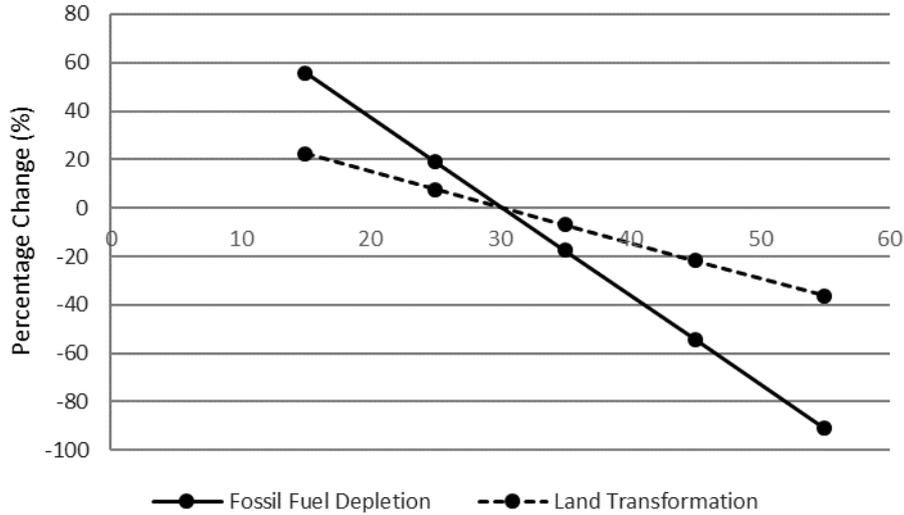


Figure 2-9 Effect of variation of coal yield on fossil fuel depletion and natural land transformation for the two-stage flotation scenario

2.4.2 South African scenario analysis

In order to fully understand the environmental implications of the two stage flotation process, the results of this study have been interpreted in the context of the South African situation. South Africa produces approximately 220 Mt of coal and 11 Mt of ultrafine coal waste (dry basis) annually (DME, 2001). At a flotation yield of 30.2%, the two-stage flotation process thus has the potential to generate an additional 3.3 Mtpa of coal (constituting approximately 1.5% annual production) and 66 440 TJ of additional energy per annum from mined and processed coal resources. Even at this conservative coal flotation yield, this represents a significant increase in resource efficiency. Furthermore, with a reduction in the amount of tailings disposed to land and the incorporation of filtration units for tailings dewatering, the proposed two-stage flotation plant will result in a 86% reduction of the approximately 12.3 Mt water that is currently lost through the disposal of ultrafine tailings in South Africa annually (based on a dissipative water loss of 1.13 t/t of coal ultrafine waste). The two-stage process will also result in a decrease in the degradation of groundwater quality in the vicinity of coal waste deposits, particularly with respect to salinization.

Whilst the two-stage flotation process results in an increase in electricity consumption which has negative implications in terms of climate change and terrestrial acidification, the amount of additional electricity consumed equates to approximately 1% of the additional electricity that can potentially be generated through the combustion of recovered coal. Furthermore, the potential greenhouse gas (GHG) emissions associated with the electricity consumed by the treatment of 11 Mtpa of ultrafine coal waste will amount to < 0.1% of the current annual GHG emissions in South Africa (DEAT, 2009).

2.5 Conclusions Regarding Environmental Impact

This study has shown that, relative to conventional thickening and disposal, the proposed two-stage flotation process for the management of coal ultrafine waste results in a decrease in water pollution-related impacts (including aqueous acidification, salinisation, eco-toxicity and metal risks), as well as consumption of natural resources (including land, water and fossil fuel). Of particular significant to South Africa, a water-scarce country which relies extensively on coal as a source of energy, is the net reduction in the loss of mined coal resources and, in particular, water typically associated with the conventional management of the ultrafine coal slurry waste stream. The two-stage flotation process also results in an increase in climate change and terrestrial acidification impacts, due mainly to the gaseous emissions (CO₂ and SO₂ respectively) associated with the production of electricity and, to a lesser extent, reagents used in the flotation operations. However, the additional energy consumed within the process equates to <1% of the potential additional energy generated through the recovery of

coal, whilst the associated emission of CO₂ is projected to be < 0.1% of the current national greenhouse gas emissions per annum.

A sensitivity analysis indicated, furthermore, that an increase in coal yield during flotation will result in a significant decrease in net fossil fuel depletion and increase in associated energy recovery, whilst a reduction in process energy consumption will have a substantial impact on both climate change and terrestrial acidification impacts.

A first-order risk assessment has also shown that the final tailings from the two-stage process contains relatively low levels of soluble metals and is likely to pose a low to negligible risk to the environment and human health. The downstream utilisation of these tailings will further improve the resource efficiency and reduce the overall waste burden associated with the coal processing operations and is the subject of on-going investigations.

3 ECONOMIC ASSESSMENT OF SULPHIDE SEPARATION BY FLOTATION

The technological feasibility to mitigate ARD from fine waste fractions using flotation has been demonstrated for both coal mine wastes and hard rock mine wastes previously (Kazadi Mbamba, et al., 2013; Harrison, et al., 2013; Hesketh, et al., 2010; Kazadi Mbamba, et al., 2012). The proposed two stage flotation process yields a valuable mineral product, a low volume sulphide rich fraction and a high volume sulphide lean tailings fraction. The mineral product is a saleable commodity and the sulphide rich and sulphide lean products have revenue generating potential subject to further investigation (Harrison, et al., 2013). This potential is further explored as part of this project (Chapters 4 and 5) and may increase the economic feasibility of the process. This is particularly important in cases of abandoned mines where capital to finance remediation is not easily accessible.

Demonstration of the viability of the two-stage flotation process, and its potential to generate value from coal mining waste, is investigated here. In keeping with procedures as outlined by Jera (2013) and in Harrison et al. (2013) for the two-stage flotation of coal waste, order of magnitude estimates for four coal desulphurisation plants were applied to develop the profitability analysis. The net present value (NPV) and internal rate of return (IRR) for this analysis was prepared using data obtained from bench-scale experiments (Amaral Filho, et al., 2012; Magabane & Naidoo, 2011; Iroala, 2014; Kazadi Mbamba, 2011). A sensitivity analysis with respect to key variables is shown for these studies to determine the long-term viability of this project in fluctuating markets.

3.1 Basis for Plant Design

Four case studies (A-D) were investigated, using identical criteria for the plant design but each were based on experimental results for the separation of fine coal, sulphide-rich fraction and sulphide-lean fraction of different coal waste streams using laboratory flotation tests. All plant equipment was based on a feed rate of 100 t/h nominal coal ultrafines from coal mine waste streams, with similar operating conditions (Table 3-1), except for the retention time in the second stage.

Table 3-1 Design basis for all plants

Parameter	Basis
Feed throughput	100 tph
Specific gravity of ore (Wills and Napier-Munn, 2006)	1.4 t/m ³
Feed moisture (Metso, 2011)	8%
Design bulk density (Metso, 2011)	0.8 t/m ³
Plant availability	82%
Operating hours	8 400 hpa
Plant capacity	720 000 tpa
First Stage Flotation	
Conditioning time	5 min
Retention time	5 min
Second Stage Flotation	
Conditioning time	5 min
Retention time (Plant B)	20 (5) min
Assumed based on information from lab-based flotation studies	

Four sets of results from experiments previously conducted in the laboratory using four different coal waste samples from the Witbank and Waterberg coal fields were worked into the four different scenarios for plant design and operation. The four different coal mine wastes from South African mines were used as feed are described in Table 3-2. Coal Wastes A, B and D, corresponding to feed streams to Plants A, B and D respectively, had similar ash-to-combustibles ratios between ca. 65-68% combustibles. Coal Waste C (Plant C) had a much lower level of combustibles at only 50%. The total sulphur present in the different fine coal waste streams varied from 0.7% to 1.11%.

Table 3-2 Characterisation of Coal Wastes 1-4 used in experimental flotation scenarios for Plants A-D respectively

Designation of Coal Waste	Coal Waste A	Coal Waste B	Coal Waste C	Coal Waste D
Coal Waste Ash Content (%)	32.6	31.8	50	34.4
Coal Waste Total Sulphur (%)	0.7	0.75	1.11	1.08
References	Amaral Filho, et al. (2012)	Magabane and Naidoo (2011)	Iroala (2014)	Kazadi Mbamba (2011)

The reagent usage and the resultant production of clean coal concentrate, sulphide-rich concentrates and benign tailings for each of the scenarios are presented in Table 3-3.

Plant A and Plant B are based on the treatment of Coal Waste A and Coal Waste B respectively. The coal flotation stage in each of these plants used 2.79 kg/t oleic acid as collector and 0.28 kg/t methyl isobutyl carbinol (MIBC) as frother. Plant C, treating Coal Waste C, used the first stage reagents Nalflote 9858 (1.4 kg/t) and MIBC (0.11 kg/t). Plant D, treating Coal Waste D, used 1.86 kg/t dodecane and 0.11 kg/t MIBC as first stage reagents.

For the second stage, Plants A and D used the same reagent mix of 2.33 kg/t potassium amyl xanthate (PAX) collector, 0.11 kg/t MIBC frother and 0.93 kg/t of the coal depressant dextrin. Plant B used the same reagents at different dosages, 0.7 kg/t PAX, 0.22 kg/t MIBC and 0.93 kg/t dextrin. Plant C, in turn, uses the same dosages as Plants A and D, but replaced PAX with SIBX as collector.

Table 3-3 Data from laboratory flotation experiments for basis of Plants A-D

Scenario	Plant A	Plant B	Plant C	Plant D
Coal Waste	Coal Waste A	Coal Waste B	Coal Waste C	Coal Waste D
FIRST STAGE FLOTATION				
Collector	Oleic Acid	Oleic Acid	Nalflote 9858	Dodecane
Frother	MIBC	MIBC	MIBC	MIBC
collector (kg/t)	2.79	2.79	1.4	1.86
frother (kg/t)	0.28	0.28	0.11	0.11
Coal Concentrate Yield (% of feed)	45.00%	65.00%	30.20%	19.70%
Coal Concentrate Ash Content (%)	20.90%	17.63%	24.50%	13.50%
Coal Concentrate Total Sulphur (%)	0.60%	0.65%	1.28%	0.48%
1st Stage Tailings (% of feed)	55.00%	35.00%	69.80%	80.30%
1st Stage Tailings Ash Content (%)	43.30%	55.06%	61.10%	38.80%
1st Stage Tailings Total Sulphur (%)	0.80%	0.89%	0.92%	0.92%

Scenario	Plant A	Plant B	Plant C	Plant D
Coal Waste	Coal Waste A	Coal Waste B	Coal Waste C	Coal Waste D
SECOND STAGE FLOTATION				
Collector	PAX	PAX	SIBX	PAX
Frother	MIBC	MIBC	MIBC	MIBC
Depressant	Dextrin	Dextrin	Dextrin	Dextrin
collector (kg/t)	2.33	0.70	2.33	2.33
frother (kg/t)	0.11	0.22	0.11	0.11
depressant (kg/t)	0.93	0.93	0.93	0.93
Sulphur-Rich Concentrate Yield (% of feed)	25.00%	15.00%	2.20%	13.10%
Sulphur-Rich Concentrate Ash Content (%)	31.70%	na	60.00%	28.90%
Sulphur-Rich Concentrate Total Sulphur (%)	1.30%	1.01%	17.60%	2.68%
Benign Tailings (% of feed)	30.00%	19.00%	67.60%	67.20%
Benign Tailings Ash Content (%)	46.70%	na	62.70%	40.80%
Benign Tailings Total Sulphur (%)	0.50%	0.50%	0.65%	0.38%
References	Amaral Filho, et al. (2012)	Magabane and Naidoo (2011)	Iroala (2014)	Kazadi Mbamba (2011)

3.2 Flotation Plant Design and Capital Costs

The two-stage flotation process described for coal desulphurisation produces three products: a coal concentrate, a sulphide enriched concentrate and a sulphide lean tailings stream. It is envisaged that the process can be applied to coal waste from legacy or active coal fines discard dumps. It could also be incorporated into the coal processing circuit, receiving the coal waste direct from the thickener underflow or being incorporated into existing coal flotation plants by separation decisions being informed by both quality and recovery of product streams as well as waste streams.

Preliminary equipment sizing, pricing and indicative capacities have been presented for the economic assessment for the two-stage flotation plant for a single fine coal waste sample (Harrison, et al., 2013; Jera, 2013). Flotation, filtration, thickening and pumping were identified as the major unit operations. Several process design assumptions, including process design criteria, were made based on preliminary test work results and coal preparation industry standards. These formed the basis of the proposed flowsheet presented in Figure 3-1 and the techno-economic feasibility study that follows.

3.2.1 Desulphurisation process description

3.2.1.1 Preparation circuit

The preparation circuit was designed with a front-end loader that tips the coal waste feed into the receiving bin. The tailings from the receiving bin are conveyed to the plate feeder where water is added to form a slurry of the desired feed density, prior to entering the first stage conditioning tanks.

3.2.1.2 Flotation circuits

The flotation process incorporates the two-stage operation that separates a coal fraction from sulphide minerals through flotation, with the tailings for the first flotation circuit forming the feed of the second circuit.

Each flotation circuit consists of a conditioning tank which receives the slurry. Collector and frother reagents are added to each of the stages with a coal suppressant added in the second stage. The conditioned pulp enters the flotation bank. The number of cells in the flotation bank depends on the feed rate and retention time required. The froth from the first stage is collected in the concentrate tank as the clean coal concentrate recovered for sale. The tailings from the first stage report to the second stage conditioning tank before entering the second stage flotation circuit. The sulphides in this stream are floated and recovered as sulphide concentrate while a sulphide-lean stream is recovered as “benign” tailings. The three products from the flotation circuit then enter the dewatering circuits.

The design parameters for all plants are presented in Table 3-1 with bench-scale flotation reagents and stream properties for Plants A-D are recorded in Table 3-3.

3.2.1.3 Dewatering circuits

The clean coal concentrate, sulphide-rich concentrate and sulphide-lean tailings are subjected to a dewatering circuit to remove excess water. Coal and sulphide concentrates are passed directly through filter presses while sulphide-lean tailings, which have a much higher moisture content, are thickened prior to being filter pressed. The dewatered products are subsequently conveyed to stockpiles.

Water recovered from the dewatering circuit is recycled to supplement process water. Spillages in the plant are recovered by sump pumps and recycled back to the process. It is thus assumed that there are no water disposal requirements within this design.

3.2.2 Plant site requirements

As this study mostly considers existing coal mining activities, it was assumed that transport networks, water supply and labour are readily available. Electrical power for the flotation plants was estimated on current Eskom rates and it was assumed that electrical grid connections are in place. Provision was made for a generator in case of power cuts or power interruptions for essential equipment. Fire extinguishers and carbon dioxide protection for electrical substations were also assumed to be in place.

The dewatering circuit was designed to recover water from the products and eventually recycled back into the plant together with any spillages as process water. A water circuit for potential fire hazards in all areas was provided. Fresh water supply was assumed to be available from the local municipality. Allowances for the installation of any additional water infrastructure have been made.

Road and rail networks provide access to the plant. This is to allow the transportation of chemical reagents and other raw materials to the plant and clean coal product to be easily dispatched by railway. Process plant structures, workshop, laboratory, offices and stores were also incorporated into the plant site requirements when developing the techno-economics. Labour required for the construction of facilities and plant structures, as well as skilled labour to meet operational plant requirements, were assumed to be available in the surrounding areas.

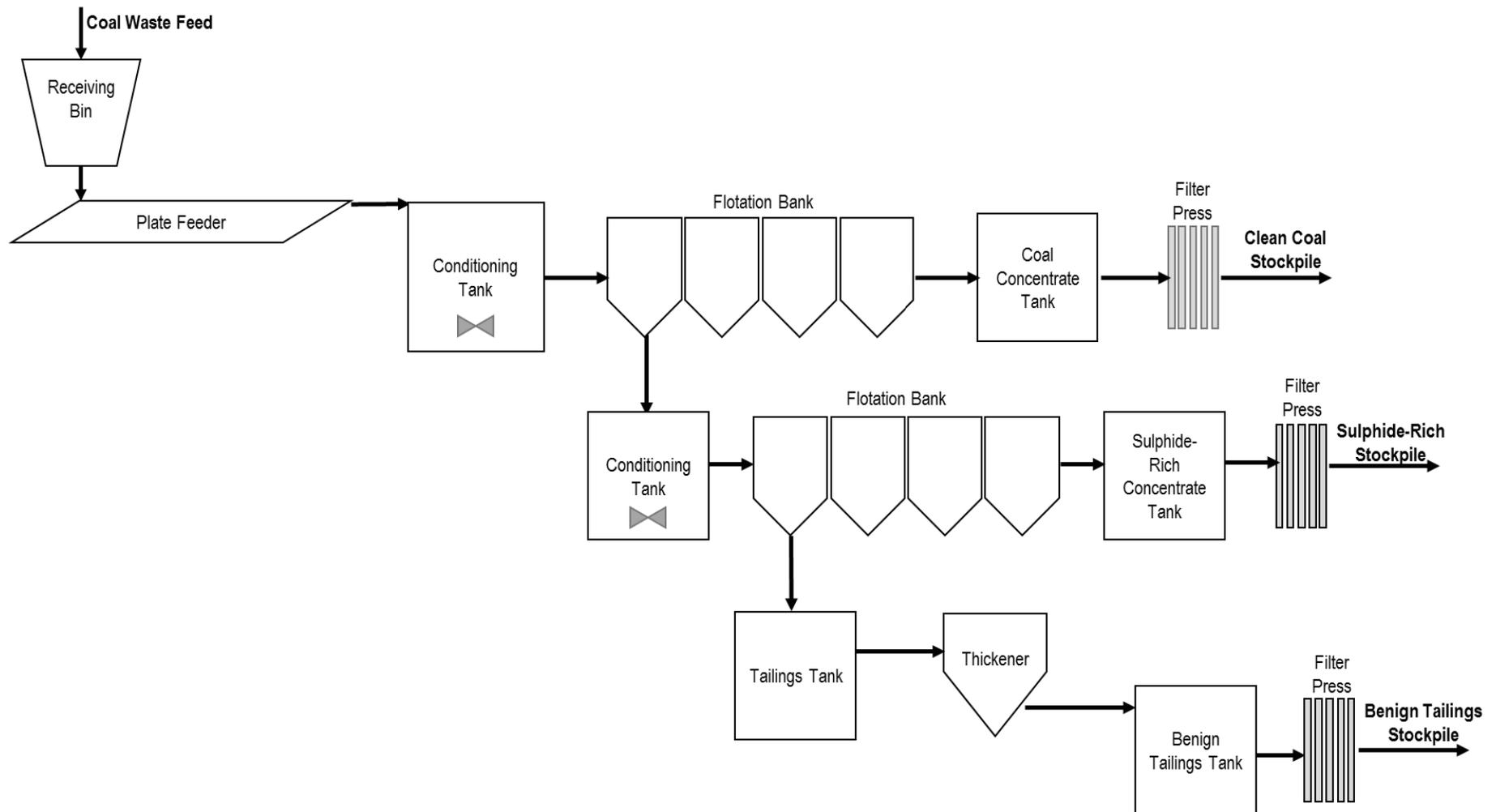


Figure 3-1 Two stage coal desulphurisation block flow diagram: based on Jera (2013)

3.2.3 Capital cost of plant

The project life and plant availability were set at 15 years and 82%, respectively. Order of magnitude estimates with accuracies within 30-50% were utilised and are reflected in the cost analyses.

Table 3-4 lists the estimated capital expenses for setting up the standard two stage flotation. Prices for major equipment units were sourced from vendors in 2013 and general cost estimates were determined using factor methods (Sinnott, 2005). The capital expenditure for all equipment was estimated at ZAR 53 million, with total capital expenditure for the installed plant estimated at ZAR 198 million, including a 20% contingency.

Table 3-4 Capital costs for coal desulphurisation Plants A-D (Jera, 2013)

Major equipment		ZAR (x1000)
Ball mill		13 600
Flotation cells		11 631
Thickener		3 912
Filters		17 494
Other equipment		6 363
<i>Total equipment</i>		<i>53 000</i>
Capital costs		
		ZAR (x1000)
Equipment	35%	53 000
Equipment delivery	7%	10 600
Equipment installation	10%	15 142
Instrumentation and controls	3%	4 542
Piping	7%	10 600
Electrical	8%	12 114
Buildings – structural steel	17%	25 742
Civil and earthworks	9%	13 628
Yard improvements	1%	1 514
Service facilities	3%	4 542
<i>Direct Costs ‡</i>	<i>100%</i>	<i>151 424</i>
Engineering and supervision*	5%	7 571
Construction*	5%	7 571
Start-up costs*	1%	1 514
Contingency*	20%	30 285
Total cost	131%	198 365

* these are based on the direct equipment costs (‡)
See Appendix A for source of factors

3.3 Operating parameters and costs

The cost of reagents is calculated using actual costs of reagents and usage projected from the experimental data used as a basis for each plant. The projected annual reagent costs are listed for each plant in Table 3-5. Annual operating costs are enumerated in Table 3-6, using the following estimated costs per unit. The plant water utilised is calculated at ZAR 7/m³ and electricity costs at ZAR 1.94/kWh. The labour, overheads, laboratory and transport cost estimations are taken as ZAR 20/tonne, ZAR 5/tonne, ZAR 2/tonne and ZAR 20/tonne, respectively (Jera, 2013). Reagent costs are calculated from the dosages listed in the original experimental data (Section 3.1). Disposal costs and contingency are based on the total direct operating costs, at 2% and 20% respectively. The total annual operating costs for Plants A-D range between ZAR 140 million and ZAR 170 million. The flotation reagent costs are the highest operating cost by far, contributing over 60% to the direct operating costs, with the collectors being the highest cost among the reagents used (Figure 3-2).

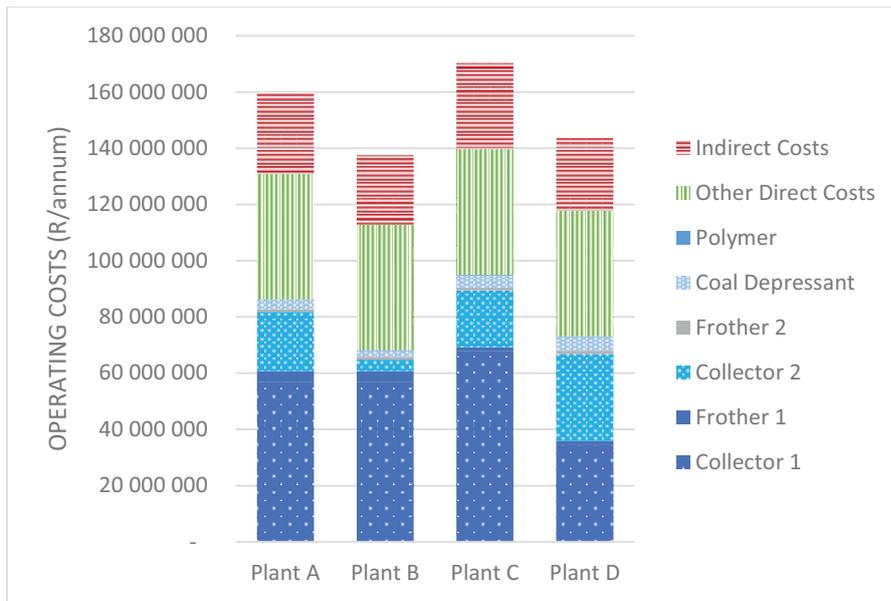


Figure 3-2 Proportions of annual Operating Costs for Plants A-D. Numerals 1 and 2 refer to the frother and collector reagents used in the corresponding stages of the flotation circuit.

Table 3-5 Annual reagent costs for Plants A-D

	Plant A					Plant B					Plant C					Plant D							
	Dos- age	Feed	Re- agent	Unit Price	Cost	Dos- age	Feed	Re- agent	Unit Price	Cost	Dos- age	Feed	Re- agent	Unit Price	Cost	Dos- age	Feed	Re- agent	Unit Price	Cost			
	[kg/ tonne]	[tpa]	[tpa]	[R/ tonne]	[R/annum]	[kg/ tonne]	[tpa]	[tpa]	[R/ tonne]	[R/annum]	[kg/ tonne]	[tpa]	[tpa]	[R/ tonne]	[R/annum]	[kg/ tonne]	[tpa]	[tpa]	[R/ tonne]	[R/annum]			
<i>Stage 1</i>																							
Collector	2.79	720 000	1 977	28 800	56 939 733	2.79	720 000	1 977	28 800	56 939 733	1.40	720 000	992	68 310	67 768 997	1.86	720 000	1 318	26 117	34 423 496			
Frother	0.28		198	19 000	3 769 905	0.28		198	19 000	3 769 905	0.11		78	19 000	1 481 034	0.11		78	19 000	1 481 034			
<i>Stage 2</i>																							
Collector	2.33		908	23 300	21 158 909	0.70		174	23 300	4 045 207	2.33		1 152	17 524	20 195 755	2.33		1 326	23 300	30 892 007			
Frother	0.11	396 000	43	19 000	814 569	0.22	252 000	55	19 000	1 036 724	0.11	502 560	54	19 000	1 033 762	0.11	578 160	63	19 000	1 189 270			
Depressant	0.93		362	9 250	3 352 788	0.93		231	9 250	2 133 592	0.93		460	9 250	4 254 993	0.93		529	9 250	4 895 071			
<i>Other</i>																							
Polymer	0.07	720 000	5.95	45 000	267 862	0.07	720 000	5.95		267 862	0.07	720 000	5.95	45 000	267 862	0.07	720 000	5.95	45 000	267 862			
Total Reagent Cost [ZAR/annum]					86 303 765						68 193 023						95 002 403						73 148 739

Table 3-6 Total annual operating costs for Plants A-D

	Cost/ Unit	Units	Plant A		Plant B		Plant C		Plant D	
			ZAR	Proportion	ZAR	Proportion	ZAR	Proportion	ZAR	Proportion
<i>Direct Operating Costs</i>										
Water	7.00	R/m3	907 200	1%	907 200	1%	907 200	1%	907 200	1%
Electricity	1.25	R/kWh	7 200 000	6%	7 200 000	6%	7 200 000	5%	7 200 000	6%
Labour	20.00	R/tonne	14 400 000	11%	14 400 000	13%	14 400 000	10%	14 400 000	12%
Maintenance	5%	equipment	2 650 000	2%	2 650 000	2%	2 650 000	2%	2 650 000	2%
Overhead	5.00	R/tonne	3 600 000	3%	3 600 000	3%	3 600 000	3%	3 600 000	3%
Laboratory	2.00	R/tonne	1 440 000	1%	1 440 000	1%	1 440 000	1%	1 440 000	1%
Transport	20.00	R/tonne	14 400 000	11%	14 400 000	13%	14 400 000	10%	14 400 000	12%
Reagents		Table 3-5	86 303 765	66%	68 193 023	60%	95 002 403	68%	73 148 739	62%
Total Direct Costs			130 900 965	100%	112 790 223	100%	139 599 603	100%	117 745 939	100%
<i>Indirect Operating Costs</i>										
Disposal Costs	2%	indirect op	2 618 019	2%	2 255 804	2%	2 791 992	2%	2 354 919	2%
Contingency	20%	Indirect op	26 180 193	20%	22 558 045	20%	27 919 921	20%	23 549 188	20%
Total Indirect Costs			28 798 212	22%	24 813 849	22%	30 711 913	22%	25 904 107	22%
Total Annual Operating Costs			159 699 178	122%	137 604 072	122%	170 311 515	122%	143 650 046	122%

3.4 Cash Flow Analysis

Flotation desulphurisation of fine coal waste or hard rock tailings is proposed to achieve both prevention of acid rock drainage and enhanced resource productivity. Product streams include a clean coal concentrate, a sulphide-rich concentrate and a sulphide-lean (benign) tailings. The fine coal product, with an approximate calorific value of 23 MJ/kg, has a potential domestic and export market, subject to product quality requirements. The sulphide rich-concentrate fraction is an acid generating stream that can either undergo further processing for value generation (Chapter 4) or be contained in an environmentally acceptable manner conforming to government legislation. The sulphide-lean tailings fraction is a non-acid generating stream with potential uses at the mine site (Chapter 5) or beyond which reduce waste to be disposed and enhance resource productivity.

3.4.1 Potential product revenue

While all three product streams have potential value, the clean coal concentrate has immediate and recognised product value. The potential revenue is easily estimated based on an existing market with which the owner-entity is already engaged and for which unit prices are readily available. The estimated annual revenue, based on a unit value of R 349/tonne for the coal product (2013 value, Jera (2013)), was calculated using production estimates based on experimental data. The projected revenue from the clean coal stream for Plants A-D is presented in Table 3-7.

Table 3-7 Estimated annual revenue from clean coal concentrate sales for Plants A-D

Plant	Annual Production (t/year)	Unit Price (ZAR/t)	Revenue (ZAR/year)
Plant A	324 000	349	113 million
Plant B	468 000	349	163 million
Plant C	217 440	349	76 million
Plant D	141 840	349	50 million

Both streams from the 2nd stage flotation, or physical separation, process have documented potential for valorisation. However, ongoing research is underway to assess options adequately and inform decision making on preferred options. Studies on further processing and uses of the sulphide-rich concentrate and benign sulphide-lean tailings are contained in Chapters 4 and 5 respectively. The expected annual rates of production for these two streams in each of the four plants A-D, are reported in Table 3-8.

Even in the event of no further value generation, there will be the reduced cost of containment and disposal compared to the costs associated with untreated coal waste. This is concomitant with the much-reduced volume of the acid-producing stream which requires containment and the increased ease of disposal associated with the sulphide-lean tailings as well as its reduced volume. Disposal costs are further reduced if the output streams from the 2nd stage flotation are re-purposed, even in the absence of value generation.

In this analysis, the potential positive value of these two streams is not included. Further, the cost of containment and disposal is also excluded on the basis that this will certainly be less than the current cost associated with the management of the coal waste. In short, this techno-economic analysis provides the worst-case scenario of no additional value generation beyond the additional coal recovered and no savings in disposal costs.

Table 3-8 Estimated annual production of sulphide-rich concentrate and benign tailings streams for Plants A-D

Plant	Sulphide-Rich Concentrate Annual Production (t/year)	Benign Tailings Annual Production (t/year)
Plant A	180 000	216 000
Plant B	108 000	136 800
Plant C	15 840	486 720
Plant D	94 320	483 840

3.4.2 Cash flow analysis

This economic model represents a framework to allow different process assumptions to be tested such that the resultant options can be evaluated. Table 3-9 provides a summary of the cash flow analysis. The order of magnitude estimates are based on earnings before interest, taxes, depreciation and amortisation.

Based on the estimates of capital and operating expenditure, with a discount rate of 14% and no escalation, the NPV and IRR for Plants A-D are calculated (Table 3-9) over 15 years. The NPVs are all negative but vary by up to an order of magnitude with changes in the coal concentrate yield with attendant change in the reagent costs for the second stage flotation. However, it is important to note that the NPV is based on revenue estimates for the saleable coal concentrates only (Table 3-7) and do not make allowances for potential revenue from sulphide-rich and benign tailings fractions (Section 3.4.1). Further, this analysis has not taken into account the reduced disposal costs that can be associated with desulphurisation and re-purposing of waste materials. Even in the absence of these, Plant B with its high coal concentrate yield has positive gains and therefore an IRR at 10%.

Table 3-9 Projected cash flow analysis for Plants A-D over 15 years with a discount rate of 14% and no escalation

	Plant A	Plant B	Plant C	Plant D
Project NPV (ZAR)	- 484.7 million	- 40.3 million	- 778.3 million	- 776.6 million
Internal Rate of Return (%)	-	10	-	-

The highest cash flow was observed for Plant B, with a NPV of ZAR 40 million (IRR of 10%). Compared to Plant A that used the same flotation reagents, the NPV was significantly lower for this operation (ZAR 480 million, Table 3-9). This is attributed to the markedly higher coal yields achieved in Plant B (65%) compared with 45% in Plant A (Table 3-3). Due to the higher coal yields, less is processed in the second stage in Plant B, which ultimately incurs lower reagent costs. Higher revenues with lower overall operating costs are therefore observed for Plant B (Table 3-7; Table 3-9 and Figure 3-2).

With comparatively lower coal yields, Plants C and D each have notably lower NPVs than Plants A and B at ca. ZAR 780 million. The coal waste feed (Table 3-2) to Plant C has a lower coal content (50% combustibles) than the other three wastes (>65% combustibles), therefore the lower yield (30%) is unavoidable. There is an even lower coal concentrate yield (20%) in Plant D; however, here the first stage collector (dodecane at 1.86 kg/tonne) has a considerably lower cost than the first stage collectors for the other plants, which keeps the NPV approximately the same as that of Plant C.

3.4.3 Sensitivity Analysis

A sensitivity analysis was conducted considering the impact of varying the project parameters on the corresponding plant NPVs. Changes in the coal price, clean coal concentrate yield, reagent costs, total operating costs and utilisation costs were investigated. As with the base case scenarios, variances in the NPV were calculated over 15 years with a discount rate of 14% (Figure 3-3).

The effect in changing the coal base price from ZAR 349 per tonne on the project NPV was initially investigated. Price increases up to 45% (ZAR 506 per tonne) and decreases up to 60% (ZAR 140 per tonne) are considered in Figure 3-3. With a base-case NPV close to breakeven point, favourable gains as little as 4.5% would bring Plant B's NPV within the positive range. However, a price increase of 45% is insufficient to yield a positive NPV for Plants A, C and D. The yield of the coal concentrate for these scenarios are too low for this increase in coal price to result in a breakeven for these processes where clean coal as the only revenue generating product. Even though Plant A has a NPV 60% greater than Plants C and D and a coal concentrate yield of 45%, a positive NPV would only be realised under these conditions if the base coal price were increased by about 65%. However, it is important to note that the reduced disposal costs following coal recovery and potential repurposing of the sulphur-lean fine coal waste material will influence the potential financial gain.

The influence of fluctuating coal concentrate yields on cash flow for Plants A-D are also investigated in Figure 3-3. Variances between -60% and +40% were explored. In Plant B, the yield cannot be significantly increased as approximately 85% of combustible coal is already recovered. As expected, increasing and decreasing trends in yields are close to those for observed for changes in the coal price. This is because both affect the total revenue in the same direct relationship. However, the rate of change in the coal concentrate yields is steeper as a change in the yield inversely affects the reagent costs of the second stage. Thus, an increase in the coal concentrate yield reduces the plant operating costs as the revenue increases and *vice versa* with decreasing yields. Further to this, and not considered here, is the reduction in amount of material to be disposed and associated reduction in disposal costs with increasing yields of coal. The breakeven point in this model is achievable should the coal concentrate yield for Plant A increase by 59% to obtain yields which approximate those observed in Plant B. This is unlikely to be practically possible for Plants C and D with coal yields around 30% and below (Table 3-3). With a single revenue generating source, significant increases in the yields, approaching 100%, would be required for these plants to be profitable.

An inverse relationship with NPV is noted with plant reagent costs (Figure 3-3). As noted with product costs and yields, marginal changes in the reagent cost would make Plant B profitable. For Plant A, a 75% decrease in operating costs at the same coal price and yield would be required to achieve a NPV of zero. In comparison, with lower revenues due to lower yields, Plants C and D cannot reach breakeven point even with zero reagent.

The total operating costs, like the reagent costs, are inversely proportional to the NPV (Figure 3-3). With the reagent costs contributing to just under 50% of the total operating costs, incremental changes in the operating costs are more significant when including other direct and indirect costs. Plant A therefore reaches breakeven point at about -50% of the base case operating costs, whereas a decrease in operating costs for Plants C and D of -75% and -89% respectively is required before they reach an NPV of zero.

The assumed base value for plant utilisation is 82%. Sensitivity to plant availability or utilisation was investigated across a change of -30% (57.4% utilisation) to +20% (98.4% utilisation) and is shown in Figure 3-3. For Plant B, with its positive annual cash flow, the NPV rises shallowly with increased utilisation and only reaches breakeven if the plant utilisation exceeds 100%. For Plants A, C and D a change in utilisation gives a (false) inverse relationship with NPV, since the negative cash flow disappears at zero utilisation. This is clearly a confirmation that none of these plants can be economically feasible without considering alternative applications for the sulphur rich concentrates and benign tailings streams and investigate their potential as either products or in terms of reduction in environmental damage.

The potential value of the other economically significant factors is evaluated in Section 3.5.

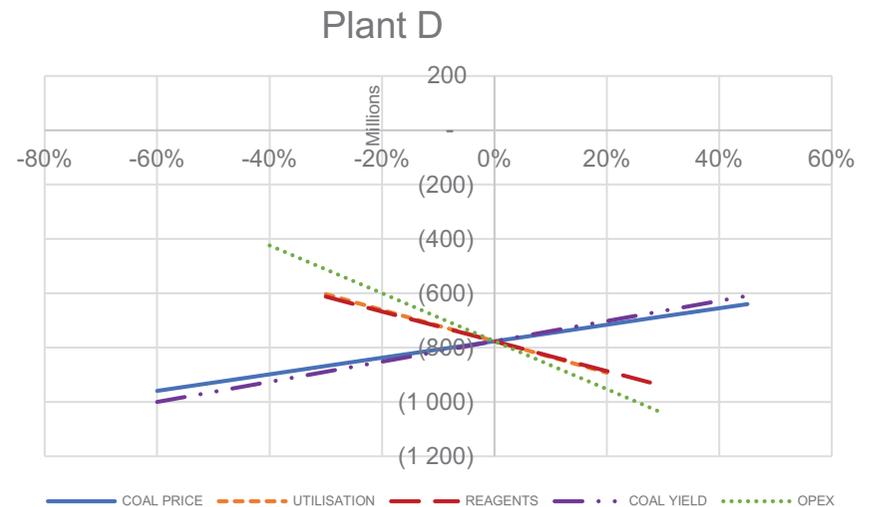
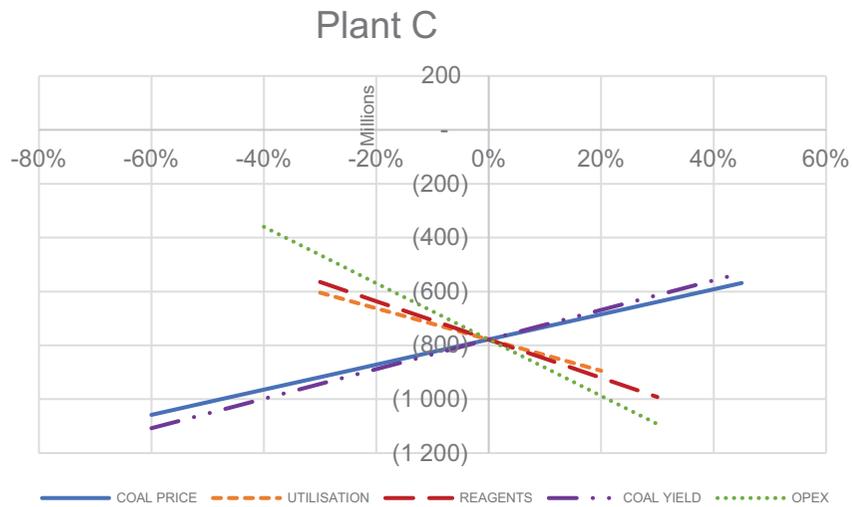
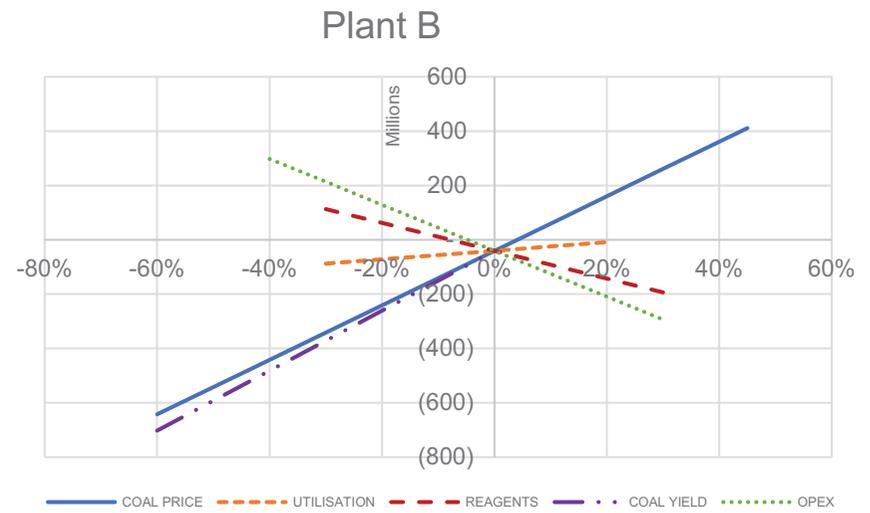
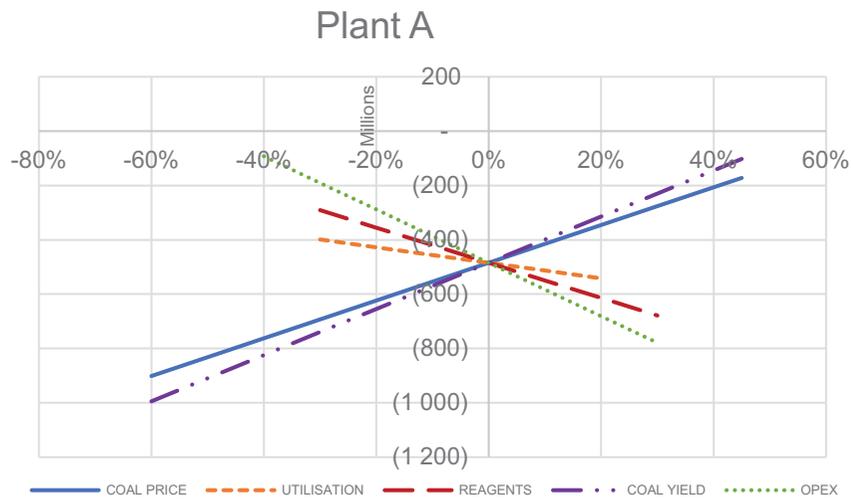


Figure 3-3 Project sensitivity analyses for Plants A-D: changes to process factors against NPV (without escalation) in ZAR millions over 15 years at 14% discount rate

3.5 The Valorisation Factors Other than the Clean Coal Product

3.5.1 Identifying additional valorisation factors

This two-stage desulphurisation operation has two key purposes: ARD prevention through desulphurisation and enhanced resource recovery. The cash flow analysis in Section 3.4 was restricted to account for the most measurable product of the two-stage flotation process only, namely the clean coal concentrate from the first stage. In order to gain a true sense of the techno-economic feasibility of the project, it is necessary to consider both the potential for value generation from the additional two byproducts, i.e. the sulphur-rich concentrate and the benign tailings from the 2nd stage flotation, as well as the reduction in material for disposal owing to its diversion to these products or to non-revenue-generating re-purposing.

The two-stage flotation process was developed to minimise mine wastes and to prevent the generation of ARD. Consequently, the primary “product” from this process is a suite of exit streams two of which are non-acid generating and one of which is an acid-generating stream which has a much-reduced volume compared to the original coal waste stream. It is intended that each of these streams be re-purposed to enhance resource efficiency. The sulphide rich concentrate will require further processing to form a third non-acid-generating product, as discussed in Chapter 4. To address the techno-economic feasibility of the desulphurisation process, it is thus necessary to consider both the associated savings with reduced quantities for disposal and the value generation of all by-product streams and to consider the cost of disposal. These are considered in Sections 3.5.2 and 3.5.3.

3.5.2 The cost of mitigation of ARD

ARD is recognised to be an issue in perpetuity for acid-generating wastes (Feris & Kotzé, 2014) and has potential to cause large scale environmental disruption in both rural and urban areas, including serious health issues for human habitats, agriculture and for wildlife areas. The monetary value of this disruption is beyond the scope of this economic assessment. However, the cost associated with methods currently in use for treatment of ARD and for reduction in volume of ARD water can be evaluated as a proxy for evaluation of the social costs of unmitigated ARD.

Jera (2013) presented the rough capital and operating costs of a number of existing processes for mitigation of ARD and tabulated these against the costs of the two-stage flotation process. This analysis is adapted to show costs for the equivalent of 720 000 tpa coal waste (Table 3-1) produced over 15 years, and with subsequent treatment of ARD (Table 3-10). In this assessment, these costs exclude the capital cost associated with building the containment structures necessary for capturing the ARD water in order to keep it out of the environment before treatment.

Three of the processes are ARD treatment processes, with varying levels of technological complexity. The fourth approach seeks to prevent ARD generation by sealing off the mine waste to prevent the ingress of water or oxygen. These engineered covers can range in complexity, but all of them require constant monitoring for deterioration with time. There is always a measure of water ingress with resultant ARD; however, this is much reduced compared with the open mine waste. The use of a capping system thus always includes some form of treatment system to deal with the ARD resulting from the imperfect seal.

Table 3-10 Comparison of ARD mitigation and prevention methods: Adapted from Jera (2013)

	Treating Water which has run through mine waste			Preventing water entering mine waste	Reducing volume of AF waste	Removing acid-forming capacity		
	Limestone	RO	KNEW	Capping	Desulphurisation			
	<i>Neutralisation using limestone</i>	<i>Reverse osmosis</i>	<i>Neutralise, filter, chemical precipitation, ion exchange</i>	<i>Engineered cover systems with limestone treatment for ingress</i>	<i>2 Stage Flotation with no further processing and capping of remaining AF waste with limestone treatment of ingress</i>		<i>2 Stage Flotation with further processing of 2 product streams</i>	
Capacity	720 000 tpa coal waste							
Comments on capacity	<i>Depends on rainfall^x and total area of dump^y</i> 35 000 m ³ acid water/year \equiv 100 m ³ acid water/day			\equiv 0.5 ha/year ^y with 20% ingress		+ < 0.125 ha/year ^z AF waste		+ cost of further processing
CAPEX: assume proportional to capacity	2 100 ZAR/m ³ /day	35 000 ZAR/m ³ /day	11 000 ZAR/m ³ /day	ZAR 4.5 million + ZAR 40 000 per year mine operation	ZAR 198 million ¹	ZAR 1.1 million + ZAR 10 000 per year mine operation	ZAR 198 million ¹	Unknown ^{2,3}
CAPEX: 15 over years mine of operation	ZAR 3.15 million	ZAR 52.5 million	ZAR 16.5 million	ZAR 67.5 million + ZAR 0.75 million		ZAR 16.5 million + ZAR 0.15 million		Unknown ^{2,3}
OPEX	ZAR 1.73/m ³	ZAR 6.00/m ³	ZAR 77.00/m ³	ZAR 1 200 pa ^w per year mine operation	ZAR 153 million ¹ pa	ZAR 300 pa ^w per year mine operation	ZAR 153 million ¹ pa	Unknown ^{2,3}
OPEX: after 15 years mine operation	ZAR 91 000 pa	ZAR 315 000 pa	ZAR 4 million pa	ZAR 18 000 pa ^w	<i>none post mine closure</i>	ZAR 4 500 pa	<i>none post mine closure</i>	
Time of costs	<i>Increasing annually during mining operation as the size of the dump increases. In perpetuity post mine closure.</i>				<i>Immediate reduction of dump size. In perpetuity for reduced dump.</i>		<i>Immediate total resolution.</i>	
Value-added Products	gypsum	gypsum potable water	various salts potable water		coal (ZAR 100 million pa) ¹ benign tailings ³		coal (ZAR 100 million pa) ¹	sulphur-based product ² benign tailings ³
Wastes	limestone sludge	brine	brine		sulphide-rich concentrate ²		none	none
All costs are indicative only. Mitigation costs extrapolated from Jera⁴				Assumptions w Limestone costs for 20% ingress, excluding inspection and "repair" costs associated with cover x Rainfall = 700 mm per year y Specific gravity of coal mine waste = 1.4 (Wills & Napier-Munn, 2006); height of dump 100 m (USGS, n.d.) z Sulphur-rich concentrate = 25% of feed (Table 3-3) Table 3-3				
1 Average for Plants A to D (Sections 3.2.3 and 3.3) 2 See Chapter 5 3 See Chapter 4 4 Jera (2013); Rykaart <i>et al.</i> , (2006); Waygood and Ferreira, (2009); The Council for Geoscience, (2010); Bewsey, (2013); GARD guide, (2013)								

The methods for ARD treatment take the acid water already formed and, at the most basic level, neutralise it, usually producing a gypsum precipitate. This can be followed by a combination of mechanical and chemical processes which further treat the water for removal of dissolved metals to produce potable water as a product. Some processes produce metal salts of value. All ARD treatment processes result in waste products, notably sludges and brines. Unless significant water storage is included in the design, temporary failure in the plant or natural flooding can cause untreated water to enter the environment. This is also possible as a result of failure in the containment structures.

It must be noted that, unlike the two-stage flotation process, while the capping system reduces ARD generation, it does not remove the risk for future ARD generation. Further, ARD treatment is required to extend over 10s to 100s of years, owing to the ongoing nature of ARD generation. None of these systems create benign conditions that are risk-free to the social and natural environment, without the removal of the sulphide fraction as proposed in the two-stage process. This means continued operating and monitoring expenditure is needed for decades (or longer) after mine closure.

3.5.3 Potential revenue from second-stage products

In the cash flow analysis (Section 3.4), the sulphide-rich concentrate and the benign tailings from the second stage flotation are excluded from contributing value and the disposal costs and ongoing monitoring and mitigation costs of ARD are also excluded. In Table 3-10 the potential cost of mitigating ARD from an unprocessed sulphide-rich concentrate is included. This cost continues for an unknown time after mine closure, typically 10s of years, or more. If desulphurisation of the waste rock and fine coal waste is completed to remove the risk and associated liability of ARD generation, then the plant may either be decommissioned risk-free on closure or used to treat previously unprocessed mine waste. The motivation for the two-stage desulphurisation process takes account of both the removal of ARD risk and the option of subsequent use of both the low-volume sulphide-rich stream and the high-volume sulphide-lean stream.

The potential for the low volume sulphide-rich stream, investigated in Chapter 4, includes both further processing of the sulphide-rich fraction and uses that tolerate the sulphides without further processing. In Chapter 4, options for use are presented and evaluated using a decision-making process. Detailed technical and economic exploration of three uses is incorporated. It is noted that this level of further use is acceptable on a breakeven basis only, owing to the advantage that ARD is fully eliminated. Some higher value products have been proposed, particularly those that remove the sulphides from the environment altogether. These high-value products may be of sufficient economic value to create sufficient revenue to offset the cost of removal of ARD risk. These options should therefore be investigated further.

The third product of the two-stage separation process is the usable benign tailings, i.e. large-volume, sulphide-lean stream. Owing to its processing, this product stream is both non-acid-generating after the removal of the major portion of the sulphides and has a reduced combustibles content (Table 3-2 and Table 3-3). Both these factors open up a number of options for use, some of which are replacements for products used on the mine site and normally sourced from elsewhere at a cost while some have potential for use beyond the mine site but within proximity to avoid large transport costs. These options are explored and evaluated in Chapter 5. Further research into these uses is underway to assess the economic benefit (WRC K5/2761). In addition to any value creation, re-purposing of the major volume fraction will influence the cash flow positively, owing to reduced disposal costs.

The cash flow analysis presented in Section 3.4 is strongly dependent on the base-case coal concentrate yield and its price. This is because, as the yield of this stream falls, there is a concomitant increase in feed rate to the second stage of the process resulting in increasing operating costs. If either of the products from the second stage can be valorised, this value will contribute to offsetting the dependence on coal concentrate yield. This is of importance for low-coal containing wastes, such as Coal Waste C in Table 3-3, where the coal yield cannot be increased enough to break even.

3.6 Techno-Economic Feasibility of Two-Stage Flotation of Coal Waste

The cash flow analysis (Section 3.4.2; Figure 3-2) and sensitivity analyses (Section 3.4.3; Figure 3-3) clearly demonstrate the potential for process optimisation. Considering revenue only from the coal concentrate recovered from the stage one flotation, the overall cost of the mitigation process is heavily dependent on the grade of waste being processed. In all cases, it is of primary importance to increase the coal recovery, while maintaining a reasonably low ash content in this product stream. Potential for reducing operating costs, especially through reduction in reagent costs, is promising. This will should include characterisation of the waste for each specific plant, with regulation of reagent dosage for optimal balance of product streams, as is currently being considered in WRC K5/2761.

Further research into re-purposing and potential valorisation of the stage two product streams, sulphide-rich concentrate and benign tailings, is essential. Revenue from these streams will alleviate the vulnerability to the volatility of the coal price (Figure 3-3). Further, where coal waste of low coal content is used (e.g. coal waste C), revenue from these two products are key to increase the economic feasibility (Table 3-2; Table 3-7), while also enhancing resource efficiency and reducing disposal costs. On further processing of the sulphide-rich stream, the full remediation value of this process will be realised (Table 3-10).

Where a two-stage desulphurisation plant is appended to an operating mine process, any loss incurred will be offset against the profits of the operating mine. This will reduce the impact of the cost of remediation. In addition, legislation is in force holding mine owners financially and practically liable for adequate remediation of all ARD in perpetuity, and for rehabilitation of land prior to mine closure (Feris & Kotzé, 2014). Hence financial advantage exists for removing the threat of ARD as part of the mining process, the in-house use of benign tailings in land rehabilitation and the enhanced potential of achieving full closure status.

Many mining countries carry the legacy of historical mine dumps, many of which were created without ARD mitigation structures. The processing of these historical mine dumps to mitigate liability is crucial to minimise damage to the natural environment and to the health of the human population living near these. In these cases, responsibility for remediation of legacy sites – in terms of the environment, society and its cost – can seldom be attributed to the polluter (Feris & Kotzé, 2014). In South Africa, the legislation derived from the constitution lays the ultimate responsibility on the South African government in cases where the polluter cannot be traced (Feris & Kotzé, 2014). This has resulted in public-private initiatives to deal with the challenges (DWA, 2010).

This early stage techno-economic analysis shows that future, more robust analyses should include an evaluation of the environmental and social cost of not processing the historical waste dumps to benign streams, with concomitant removal of pollution risk. In terms of existing mines, there may be additional costs related to non-compliance with legislation, and these should be determined.

4 PROCESSING THE SULPHIDE-RICH CONCENTRATE FROM FINE COAL WASTE FOR VALUE GENERATION

This chapter considers potential uses or processing options for the acid-generating element in the sulphide-rich concentrate generated in the 2nd stage flotation of fine coal waste. Since the most important sulphidic mineral in South African coal waste (and in general) is pyrite, the focus is on uses and processing options for pyrite. Other minerals and materials in this sulphide-rich stream include a residual coal fraction as well as quartz, gypsum, calcite and kaolinite with small quantities of dolomite, epsomite, siderite and jarosite (Kazadi Mbamba, 2011).

The sulphide-rich stream contains up to 13% sulphur (Howlett & Marsden, 2013). While this stream may need upgrading for some of the proposed processing options that follow, methods for its further upgrading are not considered in this study. Other uses do not require further processing. When ultra-pure pyrite is required for a certain processing option, a dry, ultra-pure stream was assumed as input material for the process, along with some suggestions on how poorer quality input material may affect the viability of the processing option. When a lower quality input is required, a quality appropriate for the processing option will be assumed.

In Section 4.1 of this chapter, we identify potential products that can be made from the sulphide-rich stream. Here, the quality of starting material required, the proposed processing route and nature of products is assessed for each option. Thereafter decision-making approaches to select the most appropriate alternatives are developed and suitable criteria to inform this selected. Further, decision-making is applied to select three uses of the sulphide-rich stream with potential in Section 4.2. These three processes are evaluated in detail in Sections 4.3 to 4.5.

4.1 Processes for Valorisation of Sulphide-Rich Concentrate

4.1.1 Sulphuric acid

Sulphuric acid is one of the most important chemicals in use today. It is used in processes ranging from paper making to metal etching. Traditionally, sulphuric acid is produced by roasting pyrite in the presence of air (Runkel & Strum, 2009), as shown in Figure 4-1.

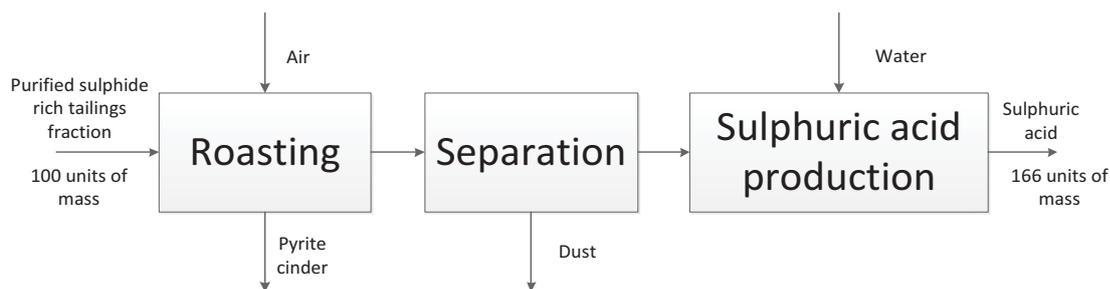


Figure 4-1 A simplified process flow diagram for sulphuric acid production (Runkel & Strum, 2009).

The process for sulphuric acid production starts with the roasting of pyrite at temperatures of between 600°C and 1000°C. In the roaster, the iron in pyrite is transformed to iron oxides, or pyrite cinder, while the sulphur is released as sulphur dioxide. The fluidised bed roaster allows intimate contact between the air and the pyrite and can accommodate small particle sizes (Runkel & Strum, 2009). When the sulphur dioxide-rich gas leaves the roaster, dust is separated using a cyclone and a hydrostatic precipitator, after which the gas is cooled and dried. The gas is then catalytically oxidised to sulphur trioxide, which is absorbed by dilute sulphuric acid, producing more concentrated sulphuric acid. Around

98.5% sulphuric acid is produced by this process and the offgas stream is practically free of sulphur dioxide, which is a good environmental feature of the process (Runkel & Strum, 2009).

A major drawback of this processing option is that one needs a pyrite stream containing 30 to 52 wt% sulphur (ESA/EFMA, 2000) in order to produce sulphuric acid cost-effectively. If the ash content of the pyrite is too high, the pyrite cannot sustain the roasting reaction with its own heat of oxidation. On the other hand, if the coal content of the stream is too high, separation of CO₂ from the SO₂ subsequent to roasting is required. Many of the modern technologies for separating CO₂ from SO₂ are quite energy intensive and may therefore not allow for cost effective acid production.

Sulphuric acid has a relatively large and established market with many potential customers already in existence. This is a major advantage. An additional advantage, using 2012 figures, is that the production of sulphuric acid is not expected to be in excess of demand from 2012 through 2015, as expressed at the 2012 Sulphur World Symposium (Feytis, 2012). This means that competition is not currently a global problem.

The heat contained in the SO₂ gas stream exiting the roaster can be used to preheat the sulphide-rich stream entering as well as to produce steam for electricity production. The iron oxides, or pyrite cinder, that are produced as by-products can be used as an iron source material in the production of cement (Runkel & Strum, 2009) or in the production of paint or concrete pigments, as mentioned in Section 4.1.2 (De Almeida Silva, et al., 2011).

4.1.2 Paint and cement pigment

In addition to using pyrite cinder as a pozzolanic material for cement production, it can also be used to manufacture the pigment, hematite. Pyrite cinders mainly consist of the strongly magnetic iron oxides maghemite ($\gamma\text{-Fe}_2\text{O}_3$) and magnetite (Fe_3O_4) (Shoumkova, 2003) and can be processed to produce hematite ($\alpha\text{-Fe}_2\text{O}_3$) pigment. Hematite, which colour ranges from metallic grey to red, is commonly used as a pigment in paint, plastic, rubber, paper and even cosmetic and the process for producing it from pyrite cinders is shown in Figure 4-2, yielding a rust-red pigment.

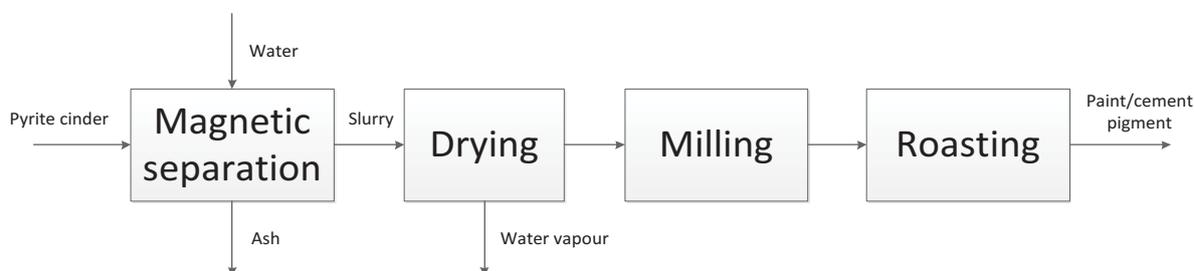


Figure 4-2 The process for producing paint and cement pigment.

The first step of the process involves purifying the material from any excess silicates and gangue materials by magnetic separation (Shoumkova, 2003). This is necessary to ensure that the hematite is as pure as possible, making the produced pigment brighter. The material is slurried and passed through a magnetic separator, concentrating the magnetic component of the cinders to 82% with a yield of 89% of the useful material (Shoumkova, 2003). After drying the concentrate, it is milled to 4-5 μm and roasted at temperatures around 870°C for about 30 min to produce a pigment of acceptable quality.

Hematite pigment is a well-known product with an established market and producers. A scenario study based on the South African market and its production from the sulphide-rich stream from processed coal waste will be necessary to determine whether this production process is a good outlet for pyrite-derived products.

This processing option does not require a high grade pyrite stream to produce a good quality pigment, because a simple and effective separation method can be used to purify the feedstock. The sulphide-rich concentrate could therefore be used for this purpose.

4.1.3 Glass pigment

Pyrite is commonly used to give glass an amber colour. The amber coloured glass is used for beer and medicine bottles. It absorbs ultra-violet radiation, protecting the contents from oxidation (Schellenbach & Krekeler, 2011). Preparing pyrite for pigments is a relatively simple process, as shown in Figure 4-3.

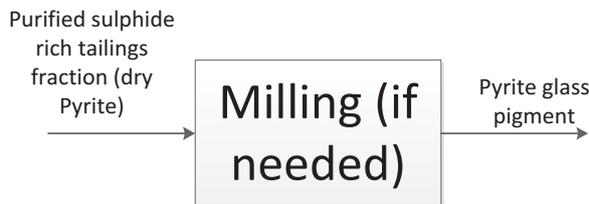


Figure 4-3 The process for producing glass pigment from pyrite.

The purified pyrite is dried using a counter current air drying system, ensuring that it is within the quality specification of 3% moisture (African Pegmatite, 2007). The pyrite is then milled to the appropriate size fraction (between 106 µm and 425 µm, or as specified by the client) and packaged before being shipped to customers.

This option is a well-known processing pathway for recovering value from pyrite. There are already established suppliers in the market, with established supply chains and customers. A scenario study based on the South African context is needed to ascertain the feasibility of adding additional pigment producing capacity to the South African market. As before, this processing option requires a very pure pyrite stream, which will require substantial upgrade of the sulphide-rich tailings stream.

4.1.4 Ferric sulphate as coagulant

Ferric sulphate is an important chemical for the treatment of industrial wastewater and sewage via the Fenton reaction. It is produced from pyrite-containing ores using a heap-leaching set-up (Colling, et al., 2011), as shown in Figure 4-4.

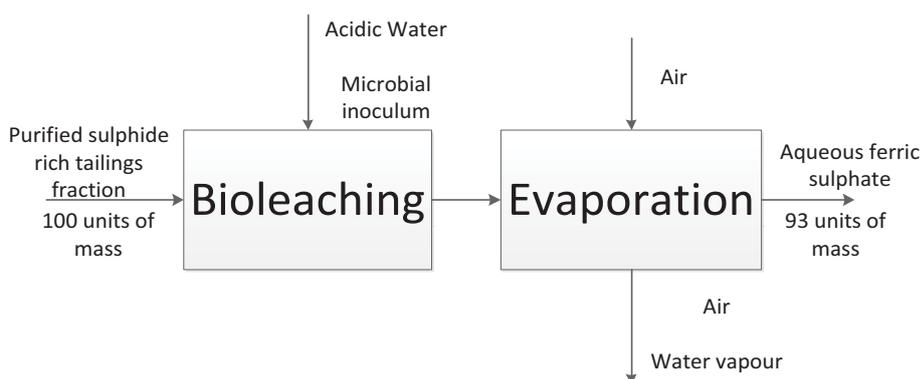


Figure 4-4 The process for the production of ferric sulphate from pyrite (Colling, et al., 2011).

As shown in Figure 4-4, the production of ferric sulphate for water treatment is relatively simple, with heap leaching being implemented, followed by a concentration step. The coal tailings used by Colling et al. (2011) had a pyrite content of 51% and a size fraction of between 2 mm and 6 mm. This size fraction is typically viewed too small for effective permeability of heap processes. Further, heap

leaching does not demand high grade pyrite. Production of ferric sulphate by heap leaching of coarse coal discards is currently underway in CeBER as part of WRC K5.2761.

Following construction of the heap and its inoculation with ferrous and sulphur oxidising microorganisms, acidic water is recirculated through the heap to facilitate leaching. The relative flowrates of the recycle and outflow from the overall process (corresponding to fresh feed flow rate) is controlled to achieve the desired concentration of ferrous sulphate. These processes may need to be preceded by a washing step to reduce the concentration of potassium amyl xanthate (PAX) to levels at which microbial activity is not inhibited within the heap. Alternatively, if the right consortia of microorganisms, tolerant of PAX, can be inoculated in the heap, a washing step may not be necessary (Okibe & Johnson, 2002).

The ferric sulphate produced using bioleaching has been shown to be as effective as a commercial product in waste water treatment (Colling, et al., 2011), demonstrating that this process can produce acceptable quality ferric sulphate coagulant. This may be a good process to implement at coal mines, since it has potential to reduce logistics expenses and make use of the mines' existing infrastructure and equipment. Implementing ferric sulphate formation at coal mines will also reduce the environmental footprint of the ferric sulphate production process. The economic viability of producing these waste water treatment chemicals from waste coal fractions will depend on the cost of the water treatment chemicals. Pyrite-rich concentrate is a plentiful and cheap input material.

4.1.5 Ferrous sulphate heptahydrate crystals

Ferrous sulphate heptahydrate, or melanterite, is used for water purification, as fertilizer and to control anaemia in humans and animals (Vigânico, et al., 2011). The production process of ferrous sulphate heptahydrate crystals is very similar to that of ferric sulphate, as shown in Figure 4-5.

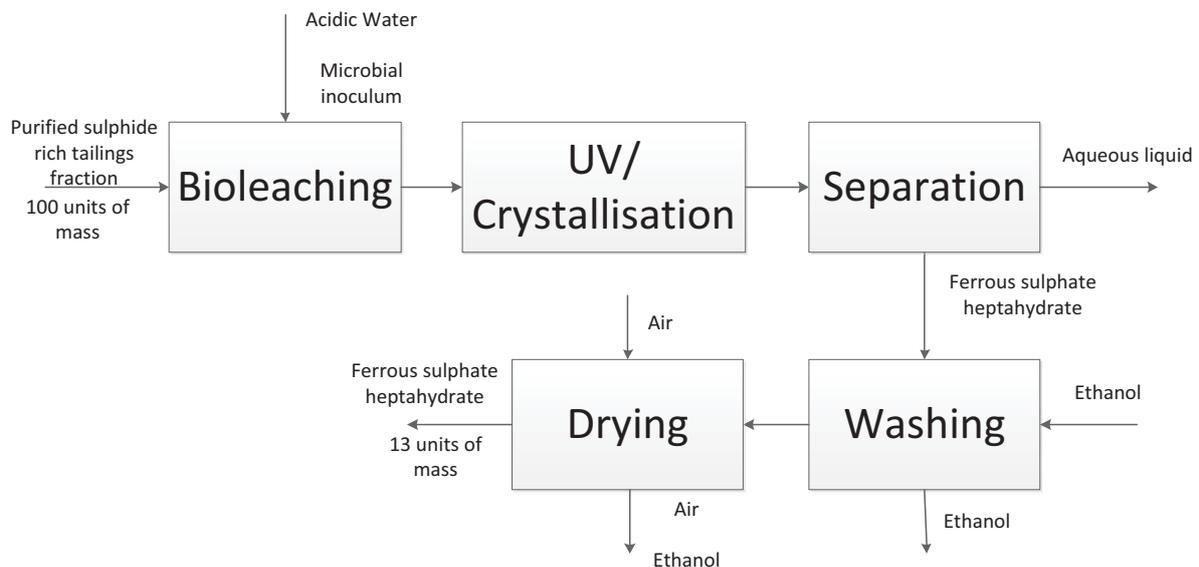


Figure 4-5 The production process for ferrous sulphate crystals from pyrite (Vigânico, et al., 2011).

As with the production of ferric sulphate, the production of ferrous sulphate from pyrite includes a heap leaching process. The leachate is recirculated until a sufficiently high concentration of ferric sulphate is achieved. This process will be accelerated by the addition of a microbial inoculum, containing pyrite oxidising bacteria. After bioleaching, the leachate will be processed by exposing it to between 77 mW/cm² and 260 mW/cm² of UV radiation (Vigânico, et al., 2011) to photo-reduce the ferric to ferrous iron and form ferrous sulphate heptahydrate crystals. The crystals are separated from the aqueous liquid using filtration, decantation or centrifugation, then washed with ethanol (Vigânico, et al., 2011) before being dried.

This process can also be implemented using UV radiation from natural light (Collienne, 1983). In this case, reduction will take place during daylight hours and re-oxidation during the evenings may occur if precipitated ferric hydroxide is present in the leachate (Collienne, 1983). Longer time periods would also be required for the photoreduction step to take place, due to the lower UV availability in natural light. It does, however, provide a cost-effective alternative to using UV lights.

This process shows promise, since the product has many potential outlets. The main challenge associated with this processing option is likely to be the electricity requirement for the UV photoreducers, which may be prohibitively expensive. If natural UV radiation is employed, however, this should reduce the production price significantly. The pyrite content of the heap may also need to be increased to increase heap leaching efficiency. Using the small particle size of the flotation product, care will need to be taken to maintain good permeability of the heap, for instance through coating the concentrate on gravel to alter packing size. An alternative is to consider tank or vat leaching.

4.1.6 Hard/secondary lead refining

In the purification of hard lead for the production of battery grids, removal of copper and tin are important for the lead-antimony product to conform to Deutsches Institut für Normung (DIN) standards (Rabah, et al., 1999). The process, shown in Figure 4-6, is a relatively typical process for the de-coppering of such alloys.

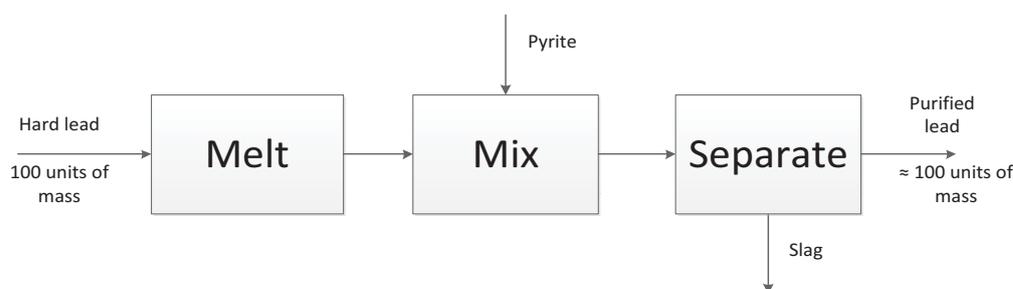


Figure 4-6 The process flow sheet for refining hard lead from pyrite (Rabah, et al., 1999).

In the refining of hard lead, the alloy is melted at temperatures of between 350°C and 550°C, followed by addition of pyrite and its mixing (Rabah, et al., 1999). A layer of copper- and tin-rich slag then forms on top of the lead, according to Equation 4-1 to Equation 4-4, and is separated from the lead. Rabah et al. (1999) has established that this method for lead refining is enhanced when sodium hydroxide is added in the stirring phase.



Addition of pyrite has also been successfully used to extract copper and other base metals from copper refining slag (Arslan, et al., 2002; Tumen & Bailey, 1990) through roasting and leaching. This process also involves the reaction of the sulphide with copper, etc. to form leachable base metal sulphides.

Once again, this processing option is within an established market and a market analysis is required to establish whether there is opportunity and potential of profit to enter this market. South Africa has

historically obtained much of its lead supply from secondary lead production within the country (Snodgrass, 1986), which is a positive indication for the viability of this processing option. The pyrite offered as a lead de-coppering agent by Ellen Chen Luoyang Dan Ding Import and Export Trading Co., Ltd. on Alibaba (2016) is a minimum of 40% sulphur and a maximum of 3% silicon oxide. Hence, considerable upgrading of the sulphide-rich stream is required for it to act as an effective refining agent without introducing additional impurities to the lead (Rabah, et al., 1999).

4.1.7 Copper smelting

Pyrite is added as auto-thermal feed for copper smelters, reducing the electricity demand of the smelter by releasing heat when oxidising (Crump, 1925). The oxidation heat of the pyrite almost exclusively heats the smelter contents to a point where the matte and slag form and are separated from each other (Peters, 1907), as shown in Figure 4-7. No mass balance was done, because the balance is dependent on the characteristics of the ore, which differ between regions.

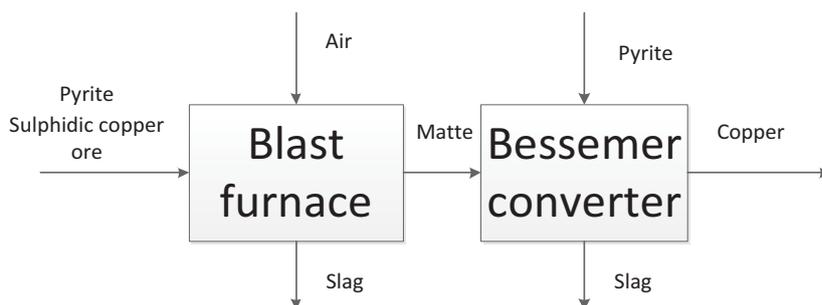


Figure 4-7 The copper purification process from pyrite (Peters, 1907)

Purified pyrite is used in smelting of low grade copper sulphide ores, increasing the energy value of ores by contributing the high heat of oxidation of iron and sulphur to the process (Peters, 1907). The pyrite is added to the blast furnace feed, along with sulphidic copper ore, silicate slagging materials (Peters, 1907) and fluxes. As air is blown into the furnace, the pyritic iron oxidises and releases energy and joins the slag material. The pyritic sulphur may either sublime or be oxidised and form SO_2 , releasing energy (Peters, 1907).

The concentrated copper (matte) is then separated from the waste material (slag) and is further purified in a Bessemer converter by oxidising the residual iron and sulphur in the matte (Rosenqvist, 2004) This is achieved by blowing air through the matte and adding silica as a slagging agent. Once again, the energy of oxidation of the iron and sulphur greatly contributes to the process heat. The copper that is produced by this process is normally 98% pure and can be further refined (Rosenqvist, 2004).

This technique is well known and is currently in use at Zambian copper smelters, including those owned by Konkola Copper Mines (Konkola Copper Mines, 2012). The pyrite added is purified by flotation, since it needs to be of a relatively high quality to avoid introducing impurities to the copper product. This method of copper smelting is especially useful in an environment where other fuel, in either the form of coke or electricity, is prohibitively expensive or difficult to obtain and where pyrite is relatively abundant. Many companies that use this method for refining copper have their own pyrite mines to supply the needed fuel for their processes. The upgrading and re-purposing of waste material has potential to avoid the need for the pyrite mining operation and associated environmental burden, provided the waste source is geographically close enough to avoid excessive transportation costs. The process also allows production of sulphuric acid, which can be used in the other processes related to the companies' activities. South African copper, however, is smelted using a reverberatory furnace by the single copper producer in the country. Pyrite smelting would therefore not be considered until the current equipment is replaced.

4.1.8 Chrome (VI) reduction

The chrome (VI) ion is a by-product of chromite refining, pulp and pigment production, and stainless steel production, amongst others (Kim, et al., 2002). It is problematic due to its high mobility and toxicity, in addition to being a Class A human carcinogen (Kim, et al., 2002). Treatment processes for water or other materials contaminated with chrome (VI) normally include the reduction of chrome (VI) to chrome (III) and the subsequent precipitation of the insoluble chrome (III). Various researchers have shown that pyrite may be a suitable reductant (Kim, et al., 2002; Lin & Huang, 2008). A suggested processing configuration is seen in Figure 4-8.

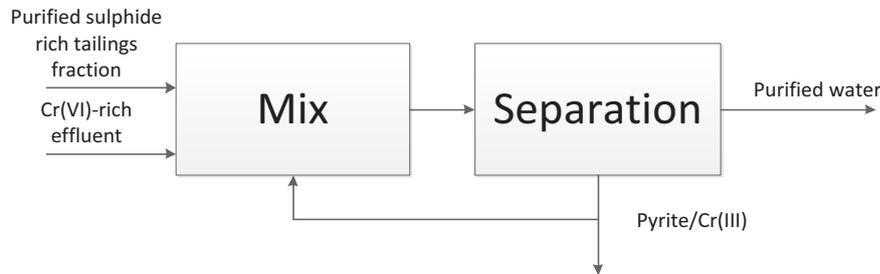


Figure 4-8 Process for reducing Cr(VI) to Cr(III) using liberated pyrite (Kim, et al., 2002; Lin & Huang, 2008)

Lin and Huang (2008) have shown that the process is highly dependent on pH, with a pH of 3 being ideal, and that the complete conversion of chrome (VI) requires excess pyrite, to the order of ten times the molar amount of chrome (VI). The residence time required for complete chrome (VI) reduction, assuming temperatures above 15°C, can be between two and eight hours during which the chrome (III) is precipitated. As such a mixing and settling set-up is recommended, with recirculation of pyrite.

This process may be a cost-effective way to treat chrome (VI) laden effluent from industries in Mpumalanga. Before it can be implemented, more research is needed around the mechanism of chrome (VI) reduction, ideal operating conditions for such a process and potential process inhibitors. It is noteworthy that this option does not seem to require a pyrite stream of high purity and therefore may be a good outlet for sulphide-rich streams that cannot be significantly upgraded.

4.1.9 Soil amelioration

One of the simplest options for extracting value from sulphide-rich waste is to use it as a soil ameliorant. Pyrite can be used to treat calcareous soil (Castelo-Branco, et al., 1999), or soil that suffers from a sulphide-deficiency (Shamim, et al., 2010). Both may lead to increased crop yields. In South Africa, the relevant soils can be found in the Northern Cape and North West Province, as well as northern Limpopo, south-western Mpumalanga and north-eastern Free State.

The sulphide-rich concentrate fraction can be used for this function as-is or can be washed to reduce the PAX content. A suggested processing route is therefore not included for this option.

Pyrite is useful for soil amelioration according to two mechanisms: it either adds a lacking nutrient to the soil in the form of sulphide (Shamim, et al., 2010), or it makes certain nutrients in the soil more accessible by reducing the pH of the soil slightly (Castelo-Branco, et al., 1999). It has also been shown that the moderate amounts of pyrite added to the soil does not negatively impact on plant growth, animal health or the toxicity of the soil. According to Shamim et al. (2010), 75 kg S/ha to 300 kg S/ha may be added to soil. Since this translates to the addition of between 2,800 kg and 11,100 kg of tailings per hectare of land, assuming a sulphur content of 2.7%. The effect of PAX on the environmental toxicity of the sulphide-rich tailings should be tested. Transport is likely to be a significant contributor to the price of the 'product', due to its relatively high volume and likely low price, hence proximity of application is highly desirable.

4.1.10 Cemented paste backfill

This option is especially relevant to underground mines that use waste backfill as aids for stabilising mine roofs. The cemented paste provides structural support for the mine while mining activities take place, but the backfill deteriorates with time (Benzaazoua, et al., 2008). This option allows the sulphide-rich fraction to be used beneficially and safely disposed of afterwards. The suggested process for producing cemented paste backfill is shown in Figure 4-9.

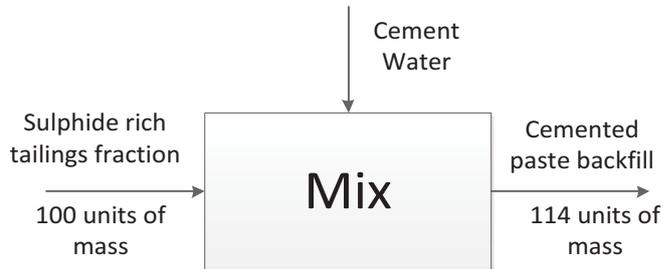


Figure 4-9 Production of cemented paste backfill (Kesimal, et al., 2005)

After being dewatered to 75%-85% solids by weight in a filter press, the concentrate is mixed with 3%-7% binding agent and water for correcting the cemented paste backfill consistency. Kesimal et al. (2005) narrowed the ideal binding agent to a mixture of Portland cement and pozzolanic materials, where the pozzolanic content is significant (Kesimal, et al., 2005). This gave the cemented paste backfill a higher resistance than other mixtures after 28 days' curing time. The cemented paste is then backfilled into the mine and left to dry.

The cemented paste backfill can provide structural support to operating mines and the cement and blast furnace slag can neutralise acid produced by the sulphide-rich tailings. Optimal backfill conditions and mixtures still need to be identified since only the mixture with a high pozzolanic content maintained its strength over 300 days. The extent to which the cement and pozzolan is able to neutralise the acid produced by the oxidation of sulphides has also not been quantified. This means that research is still needed before this option can be implemented. The advantage of this option is that it does not require a pure pyrite stream at all to be technically feasible.

4.1.11 Facilitating heap leaching

Pyrite can be used as a fuel for high temperature heap leaching of sulphidic minerals such as chalcopyrite (Kohr, et al., 2000). The oxidation of pyrite facilitates temperature increase and maintenance, because it is a highly exothermic reaction with a heat of reaction of 1 440 kJ/mol. It is also more readily oxidised than chalcopyrite, which makes it an ideal accompanying mineral for chalcopyrite heap leaching. More than 30 kg of liberated sulphidic minerals is needed per tonne of heap leaching material in order for a heap leaching process to be economical, but up to 90 kg/ton of liberated minerals can be used. Of this, around 25% should preferably be pyrite in a chalcopyrite leaching heap in order to maintain the temperature of the heap. This technique is also used for heap leaching of more readily oxidised copper ores in cooler climates (Schlesinger, et al., 2011).

Owing to the differing redox optima for leaching of pyrite and chalcopyrite, the main function of the pyrite is to heat up the heap to facilitate the onset of chalcopyrite leaching (Dew et al., 2011). The tailings would need to be of a high pyrite grade to prevent diluting the sulphide content of the heap and to avoid introducing any unwanted species into the heap.

4.1.12 Photovoltaics

Pyrite is a photosensitive semi-conductor with a direct optical band gap of 1.38 eV and an indirect optical band gap of 0.93 eV, which makes it a potentially suitable material for the manufacture of photovoltaic panels (Bi, et al., 2011). Since pyrite is normally considered to be a waste material it is

possible that pyrite solar-panels could be manufactured more cheaply than traditional polysilicon panels. Bi et al. (2011) produced a photovoltaic film composed of pyrite nanocrystals, stabilised using trioctylphosphine oxide. The film showed a stable photo-response, indicating that the film was of a high purity and had a good Hall effect (80 cm²/Vs). This technology is very promising. The process for production of photovoltaic panels using pyrite is very complex and a great deal of research and development still needs to be done; therefore, no proposed process flow sheet is provided.

4.1.13 Use in nano and micro linear actuators

The application of pyrite in nano and micro linear actuators is dependent on the slightly magnetic nature of pyrite which responds in the presence of a strong electric field (Cazacu, et al., 2010). For this application pyrite is incorporated into a heat-cured silicone rubber film, less than a millimetre thick. As before, the processing for this application is complex and requires considerable research, hence no processing flow sheet is provided.

Cazacu et al. (2010) showed that silicone rubber films incorporating pyrite exhibit good linear actuation behaviour. Reduced pyrite produced much better results than untreated pyrite. Pyrite itself may therefore only be used in special applications. Because of the research and development requirements and the small specialised market, this is not likely to provide an outlet for South African pyrite from waste mineral resources over the medium to short term.

4.2 Decision-Making Process for Selection of Valorisation Route

Decision-making methodologies are used to ensure that rational, defensible choices are made in complex decision-making contexts in which the most desirable alternatives are not readily apparent. The methodologies allow for the explicit consideration of trade-offs between alternatives, allowing decision-makers a deeper understanding of the decisions to be made. Early-stage process design decision-making is normally done under uncertainty with multiple criteria to consider (Cano-Ruiz & McRae, 1998). These decisions are also normally not considered in great detail, due to resource constraints (Sinnott, 2005).

In decision contexts where multiple criteria need to be considered, multiple criteria decision-making (MCDM) methodologies are used (Belton & Stewart, 2002). MCDM is an umbrella term classifying different types of methodologies (Belton & Stewart, 2002). Of these, a class called single index methodology was used in this project. Single index methodologies follow the five steps mentioned below (Von Winterfeldt & Edwards, 1986; Belton & Stewart, 2002). These five steps inform the structure of this section.

1. Identify alternatives and criteria
2. Evaluate each alternative with regards to each criterion
3. Ascribe weights to the criteria
4. Aggregate the evaluations of step 2 with the weights of step 3
5. Do sensitivity analyses

Before embarking on the detailed decision-making process, it was considered valuable to remove the non-starter alternatives. Since the stream to be processed is a waste stream that typically of low pyrite grade, the alternatives that could not potentially be implemented with a sulphur stream of 20% or less were removed from consideration in this evaluation. Upgrading methods would need to be established prior to their inclusion. The remaining alternatives were sulphuric acid production, ferric sulphate production, ferrous sulphate heptahydrate production, Cr(VI) reduction, soil amelioration, facilitating heap leaching and cemented paste backfill.

4.2.1 Identify alternatives and criteria

The identification and analysis of the alternatives is discussed in Section 4.1. The identification of criteria, however, is an important consideration that has thus far not received attention.

The identification of criteria is normally done with the decision-maker to establish their priorities. In lieu of using a focus group from the coal mining industry, industry sustainability reports (Anglo American, 2013; BHP Billiton, 2013) and sustainability indicator sets (Azapagic & Perdan, 2000; Global Reporting Initiative, 2013) were used to develop the criteria with the input of industry experts. The criteria were grouped into four categories: social, environmental, technical and financial, in line with triple bottom line reporting and techno-economic studies. The criteria are shown in Table 4-1.

Table 4-1 Criteria for evaluation of early-stage process design decision-making in the coal mining industry

Sustainability criteria for the South African coal mining context	Technical	Technical simplicity	Simplicity of chemistry and process control	
			System complexity	
		Technical maturity	Technical maturity	
		Conversion efficiency	Conversion efficiency	
	Social	Socio-economic development		Job creation
				Skills development potential
				Entrepreneurial activity development
		Health and safety	Operating environment health and safety	
			Community health and safety	
	Economic	Expected profitability	Expected profitability	
		Market risk	Product market	
			SA deficit	
		Scale of use		
	Environmental	Waste generation	Waste generation	
Resource efficiency		Mineral recovery		
		Energy consumption		
	Water consumption			

4.2.2 Evaluate each alternative with regards to each criterion

When multiple alternatives are evaluated with regards to multiple criteria, it is important to establish a standard to match performance to rating. To do this for an early-stage process design context, a constructed scale was developed that would allow for the rating of alternatives in an uncertain environment. The constructed scale is shown in Table 4-2.

Table 4-2 Constructed scales for evaluating the performance of early-stage design alternatives with regards to the identified criteria

Constructed scales	0	1	2	3	4
Technical total					
Simplicity of chemistry and process control	The process will be difficult to control and be sensitive to fluctuations in the operating environment and feed compositions, but relatively low-level staff will be able to make adjustments under supervision/guidance from experts.	The process will be relatively difficult to control and be sensitive to fluctuations in the operating environment and feed compositions, but relatively low-level staff will be able to make adjustments with guidance of experts available from time to time.	The process will require adjustments to be made to account for differing operating conditions or inputs, but relatively low-level staff will be able to make the adjustments.	The process will require some adjustments to be made periodically, but not constantly, and low-level staff will be able to make the process adjustments.	Process control will be simple and the process will be relatively unaffected by changes in operating conditions or feed compositions.
System complexity	The process will be complex, incorporating many recycle streams and interacting unit operations, as well as employing technically advanced unit operations.	The process will be complex, incorporating many recycle streams and interacting unit operations or employing technically advanced unit operations.	The process will be moderately complex, incorporating some recycle streams and incorporating the use of moderately advanced technology.	The process will be moderately complex to simple, incorporating few recycle streams and employing simple technology.	The process will be simple, requiring no recycle streams and have low-tech unit operations.
Technical maturity	The technology has never been implemented and is still being developed in the lab. Little development work has been done on this.	This technology has received a lot of RandD attention in the lab and has been implemented on a mini-plant scale.	The technology has been proven on a pilot or demonstration scale.	The technology has once or twice been successfully implemented on a commercial scale.	The technology is well-known and has been successfully implemented internationally for a number of years/commercially proven.
Conversion efficiency	<30%	30%-60%	60%-80%	80%-95%	>95%
Social total					
Direct job creation	5-10	11-20	21-45	46-60	61 or more
Operating environment health and safety	Workers will periodically be at risk of exposure to some hazardous (toxic, corrosive) and some moderately harmful materials, high temperature and pressures, as well as dangerous machinery and situations.	Workers will be at risk of exposure to moderately harmful materials with possible serious long-term side-effects, moderate temperatures and pressures and/or heavy duty machinery.	Workers are periodically at risk of exposure to moderately harmful material with possible serious long-term side-effects, heavy duty machinery or moderate temperatures and pressures.	Workers will be at risk of exposure to some corrosive or slightly toxic chemicals, machinery, etc.	Workers will be exposed to non-hazardous materials, ambient temperature and pressure.
Community health and safety	The community adjacent to the production area almost certainly experience negative health and life style effects, due to air, soil or water pollution, noise and dust creation. They may be influenced by physical dangers, such as nearby tailings dumps.	The community living near to the production area will face risk of local contamination of land, water or air in case of accidental leakages or spills, in addition to dust and/or noise (Tailings dumps?).	The community living near to the production area will face some negative consequences from living near to the plant, such as noise and dust, but not chemical pollution-related issues.	The community living near to the production area will be at low risk, but may experience some physical disturbances.	The community living near to the production area will in no way be adversely affected by the nearby production processes.

Constructed scales	0	1	2	3	4
Skills development potential	Majority of the jobs created will require unskilled labour (South African Gr. 9 or less).	Majority of the jobs created will require semi-skilled labour, with some unskilled.	The jobs created will require a mix of semi-skilled (majority), unskilled and highly skilled labour.	Majority of the jobs created will require highly skilled labour (e.g. graduate diploma holders), with some semi-skilled labour.	Vast majority of the jobs created will require highly skilled labour (e.g. graduate diploma holders).
Entrepreneurial activity development	The production process and products do not lend themselves to implementation by small/medium businesses or entrepreneurs i.t.o. support-industries or further product beneficiation, but only to large, established companies.	Some aspects of the products or process lend themselves to exploitation by small- or medium size business i.t.o. support industries and down-stream beneficiation or use, but only with the provision of significant support.	Some aspects of the products or process lend themselves to exploitation by small- or medium size business i.t.o. support industries and down-stream beneficiation or use.	The products and process lend themselves to exploitation by micro and small enterprises i.t.o. support industries, production and further beneficiation or use of the product. The enterprises will require support.	The products and process lend themselves to exploitation by micro and small enterprises i.t.o. support industries, production and further downward beneficiation or use of the product.
Economic/financial total					
Expected profitability	IRR <-10%	IRR -10%-0% (comparable to not implementing any process and discarding the waste in tailings impoundments)	IRR 0%-15%	IRR 15%-25%	IRR >25%
Is the product currently sold on the South African market?	No known local market: Potential customers will have to be introduced to the product and convinced of its efficacy.	Limited and sporadic local market: The product is sold on the market, but only a few companies buy it from time to time.	Limited but consistent local market: The product is sold on the market, but only a few companies regularly buy it.	Moderate local market: The product is sold to a moderate number of customers and a few uses for the product exists.	Extensive local market: The product is sold to multiple customers for multiple uses. It is bought and sold freely.
SA deficit	South African producers currently export product or experience oversupply in the country.	Imports or exports – supply and demand are relatively evenly matched.	Small volumes are imported (e.g. 1000-10,000 t). S.A. producers are mostly able to supply the demand.	Medium volumes of the product is imported (e.g. 10,000-100,000 t of imports).	The product is imported due to severely limited in-country supply (e.g. 10,000-2,000,000 t imports).
Scale of use	The product is typically bought in measures of kg.	The product is typically bought in measures of 10's of kg.	The product is typically bought in measures of 100's of kg.	The product is typically bought in measures of t.	The product is typically bought in 10's of t or more.
Environmental total					
Waste generation	The process will produce medium to large volumes hazardous waste.	The process will produce small volumes of hazardous waste and/or large volumes of moderately hazardous waste.	The process produces small volumes of moderately hazardous waste and large to moderate volumes of benign waste.	The process produces small volumes of moderately hazardous waste or moderate volumes of benign waste.	The process only produces small volumes of benign waste or no waste at all.
Mineral recovery	Less than 10% of the inherent mineral value is recovered.	10%-40%	40%-60%	60%-90%	90%-100%
Energy consumption	more than 100kWh/t processed	60-100 kWh/t processed	30-60 kWh/t processed	0-30 kWh/t processed	0 kWh/t processed – net energy producer
Water consumption	> 4 ℓ water/kg processed	2-4 ℓ water/kg processed	1-2 ℓ water/kg processed	0.5-1 ℓ water/kg processed	0-0.5 ℓ water/kg processed

The process alternatives were rated by design experts based on their design experience and judgement. Design experts were enlisted to ensure that the ratings of the alternatives with regards to the criteria was done as accurately as possible. The raters were also given the opportunity to identify the confidence with which they assigned a specific rating. This is to help with the inclusion of data in the aggregation step. The experts used the ranking sheet shown in Table 4-3 below, which also shows the preliminary results of the evaluation step.

Table 4-3 The preliminary aggregation of process ratings received from design professionals

Rating sheet	Theoretical max	Sulphuric acid	Sulphuric acid + paint pigment	Ferric sulphate	Ferrous sulphate	Cr(VI) reduction	Soil ameliorant	Facilitating heap leaching	Cemented paste backfill
Technical Total									
Simplicity of chemistry and process control	4	3.0	2.0	3.0	1.0	2.0	4.0	2.0	3.0
System complexity	4	2.0	1.5	3.0	1.0	2.5	4.0	3.0	3.5
Technical maturity	4	4.0	3.0	3.0	1.0	0.0	1.0	2.0	2.0
Conversion efficiency	4	3.0	3.0	2.0	1.0	1.0	2.0	2.0	3.5
Social total									
Job creation	4	1.0	2.0	1.0	1.0	1.0	0.0	1.0	1.0
Operating environment health and safety	4	1.0	1.00	3.0	3.0	1.0	3.5	3.0	3.0
Community health and safety	4	1.0	1.0	3.0	3.0	1.5	3.0	2.5	3.0
Skills development potential	4	2.0	2.0	2.0	2.0	1.0	0.0	1.5	1.0
Entrepreneurial activity development	4	1.0	1.0	1.5	2.0	2.0	3.0	1.0	2.0
Economic/financial total									
Expected profitability	4	1.0	2.0	1.0	1.0	1.0	1.0	1.5	1.0
Is the product currently sold on the market?	4	3.0	3.0	2.0	2.0	1.0	1.0	1.0	1.0
SA deficit	4	3.0	3.0	2.0	2.0	1.0	1.0	1.0	1.0
Scale of use	4	4.0	3.5	3.0	3.0	2.0	3.0	3.5	4.0
Environmental total									
Waste generation	4	2.0	2.0	2.0	2.0	1.00	3.0	2.0	3.0
Mineral recovery	4	4.0	4.0	2.0	2.0	2.0	3.0	2.5	2.0
Energy consumption	4	2.0	2.0	1.0	1.0	3.0	4.0	3.0	3.0
Water consumption	4	3.0	3.0	2.0	1.0	2.5	4.0	2.5	3.0

4.2.3 Ascribe weights to the criteria

The weighting step was done using three different weighting methodologies: SWING weighting, indifference weighting and a modified form of the analytical hierarchy process (AHP). This has been done with two sets of sustainability experts and further input from industrialists will be addressed over the coming months. The preliminary results are shown in Figure 4-10 and Figure 4-11 and indicate that the weights of the different experts differ markedly.

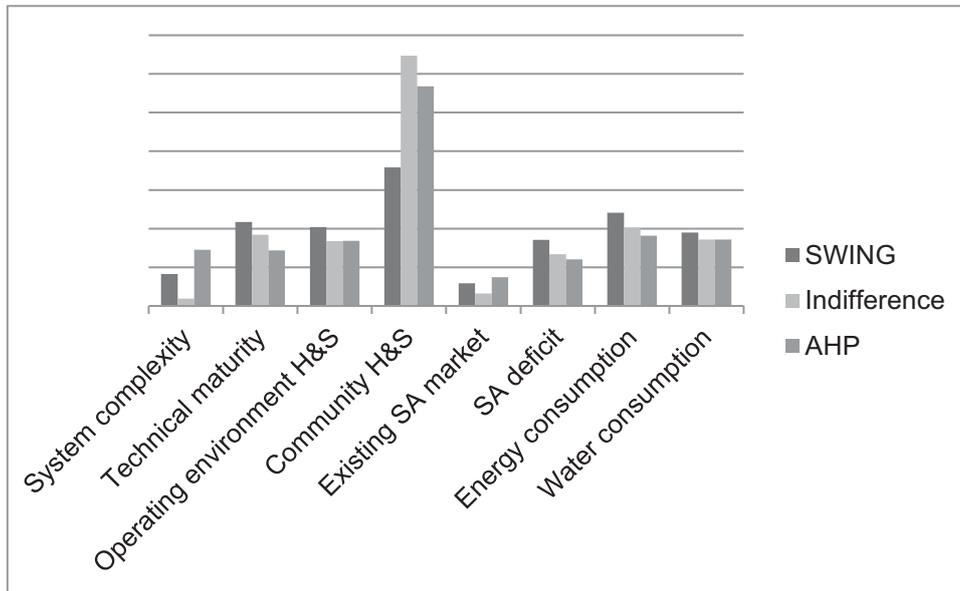


Figure 4-10 The weighting results from sustainability expert group 1

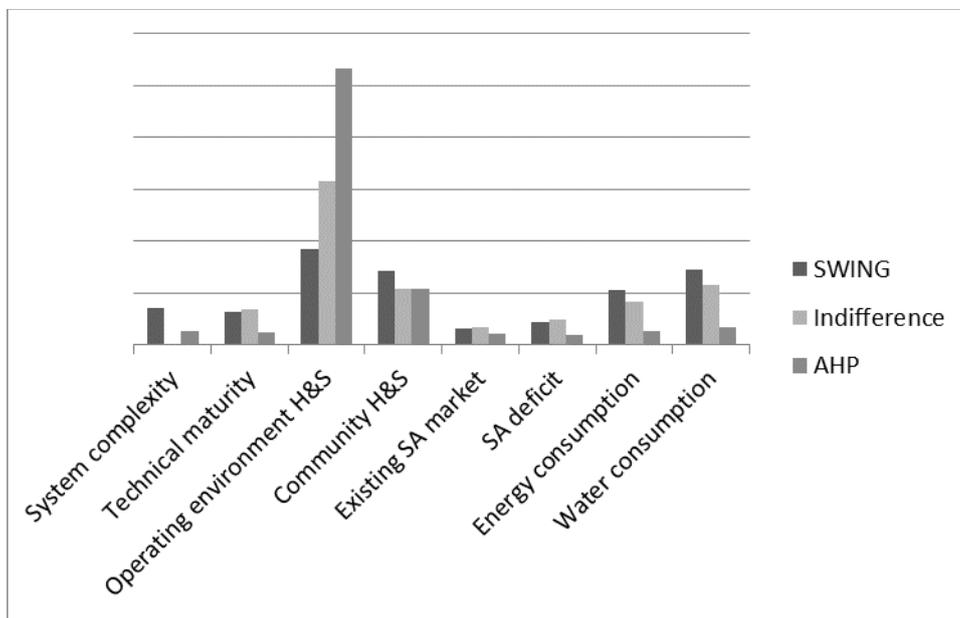


Figure 4-11 The weighting results from sustainability expert group 2

4.2.4 Aggregate the evaluations of step 2 with the weights of step 3

For this step the simplest aggregation method, the weighted additive model shown in Equation 4-5 was used. For this model, the weight of each criterion is multiplied with the rating of a process alternative for that same criterion and added to the product of the weight and ratings of all the other criteria. In this way a single score is calculated for each alternative. The alternative scores are then compared with

each other; the alternatives with higher scores are considered better performers than alternatives with lower scores. The results from the two groups of sustainability experts are shown in Figure 4-12 and Figure 4-13.

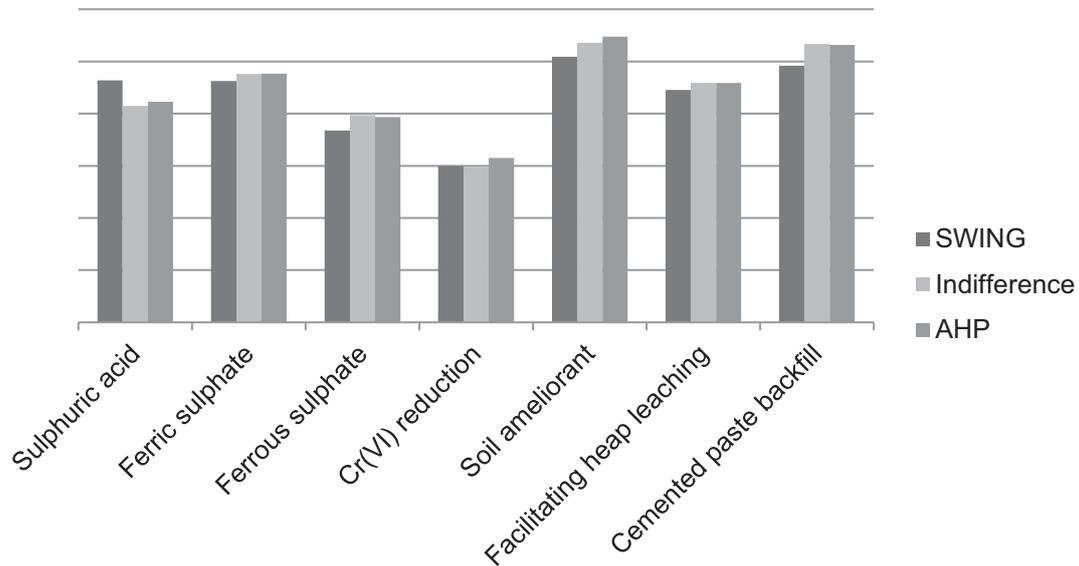


Figure 4-12 The aggregated results from sustainability expert group 1

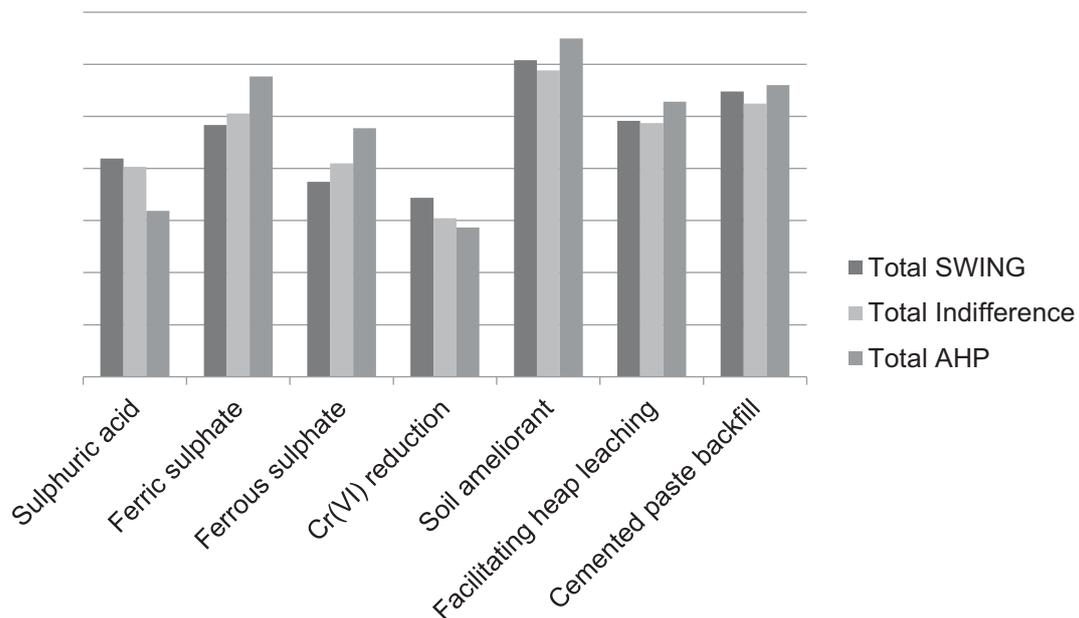


Figure 4-13 The aggregated results from sustainability expert group 2

4.2.5 Sensitivity analyses

As both the ratings and the weights are preliminary, it would be premature to do a full sensitivity analysis on the results. However, based on the outcomes shown in Figure 4-12 and Figure 4-13, it can be said with some confidence that the weighting method itself did not influence the outcomes of the decision-making process. In all cases, soil amelioration scored as the best performing option, while cemented paste backfill scored as the second best performer and ferric sulphate as third. This is also true for the scoring done by different sustainability expert groups. For both sets of rankings soil amelioration and

cemented paste backfill are the best performing options, with ferric sulphate production and facilitating heap leaching ranking third and fourth. Based on these results, soil amelioration (Section 4.3), cemented paste backfill (Section 4.4) and ferric sulphate production (Section 4.5) are considered in more detail.

4.3 Detailed Report on Soil Amelioration

In Sections 4.1.9 and 4.2.4, soil amelioration was identified as a potential option for downstream application of the sulphide-rich concentrate. In this section, the projected implications of this course of action are explored in depth and potential research and development requirements are highlighted.

The term “soil ameliorant” is generally used to refer to substances or processes that improve the physical properties of soil, such as structure, permeability and chemical properties such as, the latter including pH and metal toxicity (Bradshaw, 1997; Castelo-Branco, et al., 1999; Wong, 2003; Liebenberg-Weyers, 2010). Ameliorants often also include nutrients, as in the case of organic matter, which improves soil structure as well as nitrogen, phosphorus and iron availability (Foth & Ellis, 1996). The narrower term “fertiliser” tends to refer to concentrated substances that are added to soils to address specific nutrient deficiencies (Abadía, et al., 2011; Banath & Holland, 1976; Edis & Norton, 2012; Edmeades, et al., 2005; Mortvedt, 1991; Sharma & Swarup, 1997; Shenker & Chen, 2005). Sulphide-rich concentrate from South African coal mines would contain sulphur predominantly in the form of pyrite along with a significant coal component and some other gangue minerals (Iroala, 2014; Kazadi Mbamba, et al., 2012). The active ingredient in terms of both acid rock drainage (ARD) generation and soil amelioration is pyrite. The possibility of using pyrite as a soil ameliorant is therefore examined, with the rest of the stream initially assumed unimportant from the point of view of soil characteristics and nutrients.

Pyrite can be used to treat soil that is calcareous (Castelo-Branco, et al., 1999), alkaline or that suffers from iron or sulphur deficiency (Shamim, et al., 2010), thereby leading to increased crop yields. Pyrite is useful for soil amelioration according to three mechanisms: it supplies a lacking nutrient to the soil in the form of sulphide (Shamim, et al., 2010); it improves the soil structure by precipitating sodium ions (Somani, 1986; Vlek & Lindsay, 1978); or it makes certain nutrients in the soil more accessible by reducing the pH of the soil (Castelo-Branco, et al., 1999). This section deals with the theoretical and practical aspects of applying sulphide-rich concentrate to soils.

4.3.1 Soil characteristics

The soil required for healthy plant growth is different for different plants. Some plants prefer more acidic soil, while others prefer more neutral or alkaline soil, for instance. Plants are show varying hardiness to different conditions, such as drought or waterlogged soil. Despite this, there are some general soil characteristics that tend to be better for growing a variety of food crops, including good soil structure, cation exchange capacity, neutral pH and high organic matter content. There are also many nutrients that are important for crop growth in general. However, here only the characteristics and nutrients which are relevant to the sulphide-rich concentrate stream as a soil ameliorant are examined. Of the physical soil characteristics, soil structure and pH are discussed. With respect to soil nutrients, sulphur and iron are considered.

4.3.1.1 Soil structure

A good soil structure, called ‘crumb structure’, is important for proper drainage of soil as well as ensuring that sufficient water is retained in the soil for plant use (Malherbe, 1948). An unstructured sandy soil allows all precipitation to drain away, while an unstructured clay allows almost no water ingress and is too dense for plant roots to permeate effectively (Malherbe, 1948). Unstructured clays also tend to expand upon wetting and crack when dry. A crumb soil structure ensures that enough air is present in the soil for the respiration of plant roots, seeds and soil microbes (Malherbe, 1948).

Soil takes on a crumb structure when clays are flocculated and the soil is rich in colloidal organic matter in the gel state. Soil needs to be treated correctly to prevent the soil structure from deteriorating. Tilling the soil when too wet or dry, for instance, reduces soil structure (Malherbe, 1948). Also, a saline soil with a high exchangeable sodium content runs the risk of deflocculating clay particles, leading to an unstructured soil (Malherbe, 1948). Lastly, stockpiling soil also leads to compacted, structureless soil (Gouws, 2014).

Gypsum is normally added to improve soil structure, especially when the poor structure is due to an excess of sodium in soil. The gypsum precipitates some of the excess sodium in the form of Na_2SO_4 , while replacing the adsorbed sodium with calcium (Foth & Ellis, 1996; Malherbe, 1948).

4.3.1.2 Soil pH

Foth and Ellis (1996) have reported extensively on soil pH. Soil pH has a significant impact on nutrient availability and biological activity in soils and is therefore in important consideration when assessing a soil's fertility. The pH, called 'activity' in soil science, is affected by the mineralogy of the soil, the biological environment and the chemical environment in the soil. Soil pH is measured by mixing equal masses of soil and distilled water for an hour and then taking a reading using a glass-calomel pH probe. Neutral soils (pH 6.6-7.3), unless saline, are generally good for growing a variety of food crops, while acidic soils or alkaline soils may need pH adjustment in order to sustain agriculture.

Alkaline soils tend to be characterised by low nutrient availability, both due to the low solubility of many compounds at a high pH and to calcium dominating the dissolution equilibrium in the case of calcite-induced alkalinity. Nutrients such as nitrogen, phosphorus, iron and manganese are deficient or reduced in effectiveness at high pH. In addition, boron tends to reach toxic levels at high pH. Alkaline soils generally derive their alkalinity from dissolvable carbonates, such as calcite, leaving calcium and magnesium as the dominant cations. When sodium carbonate dominates, however, the soil tends to be extremely alkaline and the sodium ions cause soil colloids to de-flocculate and disperse, negatively affecting soil structure.

Sulphur and sulphur compounds are the most used remediation media for alkaline soils. These can be applied in 'bands', limiting sulphur contact with soil, or as a blanket application to the soil. Band application means that bands of the soil have the ameliorant lightly worked into it, while blanket application means that the ameliorant is lightly worked into the whole surface of the soil. Such applications typically only affect the top layers of the soil. Treating only 'bands' of soil requires much less sulphur to achieve an improved outcome, due to the spot-reduction of pH, enabling plants to extract nutrients from those parts of the soil in the critical period of early growth (Barber, 1984). Soil texture also plays a role in determining the amount of sulphur needed to ameliorate a piece of land. Clayey soils, for instance, need up to twice the amount of sulphur needed by sandy soils.

Acid soils may suffer from metal toxicity, nutrient unavailability and low cation-exchange capacity and care should be taken to avoid acidifying soils. Treating acid soils with acidic fertilisers without adding neutralising agents such as lime, for instance, could render the soils infertile (Malherbe, 1948). Other sources of soil acidity are acid rain, carbonic acid formed by the dissolution of carbon dioxide respiration products of roots, microbes and animals, and the removal of soil neutralising capacity by high rainfall. For this reason, high rainfall areas are more prone to acidic soils. Lime is the most popular agent for neutralising acid soils.

4.3.1.3 Sulphur

Plants need large amounts of sulphur, a component of protein. Protein-rich plants therefore tend to require more sulphur for healthy growth (Malherbe, 1948). Sulphur is absorbed at the root surface of young roots and root hairs (Barber, 1984) in the form of sulphate ions dissolved in water (Edis & Norton, 2012). Around 3 to 5 mg/l sulphate is normally sufficient for plant growth (Foth & Ellis, 1996). The presence of ammonium ions in the soil solution along with higher temperatures, up to about 35°C, and

lower pH values, down to about pH 4 seem to improve the uptake of sulphates (Barber, 1984; Foth & Ellis, 1996).

Despite the significant need for sulphur, soils tend not to be sulphur deficient. There are many reasons for this. Fertilisers were historically supplied in a sulphate form, such as superphosphate and ammonium sulphate. Sulphur from the air is deposited onto soils, especially during rain-events where sulphur species in the air dissolve in the raindrops (Barber, 1984; Foth & Ellis, 1996; Malherbe, 1948). Sulphur levels in the air tended to be particularly high near industries and fossil-fuel burning power stations, which released sulphurous compounds into the air through their flue gasses (Barber, 1984; Foth & Ellis, 1996). However, increasing restrictions on air emissions can be expected to reduce this form of sulphur entering the soil with time. Stover tends to have a significant sulphur content and is often incorporated back into soil after harvesting (Foth & Ellis, 1996). Sulphate ions become available to plants by desorption of ions from iron or aluminium minerals and dissolution of sulphate minerals in the soil (Barber, 1984; Foth & Ellis, 1996). Sulphur is added to soils in the form of elemental sulphur and other sulphur-bearing fungicides (Tweedy, 1981) and gypsum for soil amelioration (Malherbe, 1948). Plants can also absorb sulphur dioxide from the air (Barber, 1984).

Sulphur deficiency is, however, becoming increasingly common due to specialised fertilisers that do not contain sulphur, such as diammonium phosphate and potassium chloride. Sulphur air pollution is also reducing due to stricter air quality standards and sulphates are easily leached out of the soil in high rainfall environments (Barber, 1984; Foth & Ellis, 1996; Malherbe, 1948). For these reasons sulphur fertilisers are increasingly added to soils. Normal application rates range from 22 kg-S/ha to 44 kg-S/ha (Foth & Ellis, 1996).

Sulphur fertiliser can be applied in solid or liquid form (Banath & Holland, 1976; Edis & Norton, 2012; Edmeades, et al., 2005; Sharma & Swarup, 1997). Examples of solid fertilisers include gypsum, superphosphate, ammonium sulphate, magnesium sulphate, potassium sulphate, sodium sulphate and elemental sulphur (Banath & Holland, 1976; Edmeades, et al., 2005; Foth & Ellis, 1996; Sharma & Swarup, 1997). Gypsum, superphosphate and ammonium sulphate are commonly used, since they have other advantages as well, such as correcting soil structure and other nutrient deficiencies. Liquid fertilisers include water dissolved sulphates and thiosulphates, while bisulphites and polysulphides are also used (Edis & Norton, 2012). Ammonium thiosulphate and potassium thiosulphate are common liquid fertiliser options and contain a higher proportion of sulphur, compared with sulphate products (Edis & Norton, 2012). These liquid products are phyto-toxic, however, and should not come into contact with plants or seeds. Further, they are incompatible with certain other fertilisers, such as acidic compounds (Edis & Norton, 2012). Slow and fast acting sulphur fertilisers are available that perform other functions as well.

4.3.1.4 Iron

The formation of chlorophyll requires iron, despite the molecule itself not containing iron, making it an essential plant nutrient (Malherbe, 1948). Soils tend to have high iron mineral content, but due to the relative insolubility of many of those minerals, especially at alkaline pH, it may not be available to plants (Abadía, et al., 2011; Barber, 1984; Foth & Ellis, 1996; Mortvedt, 1991; Shenker & Chen, 2005; Malherbe, 1948). Insufficient iron is associated with soils with high pH, low moisture content and low organic matter content (Foth & Ellis, 1996; Mortvedt, 1991; Shenker & Chen, 2005). Organic material in soils increases iron availability by forming complexes with iron in solution, which plants can then access by reducing it at the root surface (Foth & Ellis, 1996).

Due to the insolubility of iron compounds at high pH, adding iron-containing compounds as fertiliser does not address the problem unless it is added as an organic complex. The aim of iron fertilisers is consequently either to put iron into the soil (or even directly into the plant) in a form that is plant-available or to reduce the pH of the soil to render the iron compounds inherent in the soil soluble. Substances that are used for this purpose include chelators, natural iron-containing materials, sulphurous materials and iron salts (Abadía, et al., 2011; Shenker & Chen, 2005). Synthetic chelators, for instance, are water-

soluble substances that form complexes with transition metal cations, which keeps the nutrients plant-available by preventing them from precipitating. They are effective in increasing the plant-available iron in soil but are expensive and only routinely applied to high-value crops (Abadía, et al., 2011; Shenker & Chen, 2005). Since the reason for most of the plant unavailability of iron in soils is due to an alkaline pH, soil acidification is another option for increasing iron availability (Shenker & Chen, 2005; Vlek & Lindsay, 1978; Wallace, et al., 1976). This can be done by adding acidic or acidifying compounds to the soil (Mortvedt, 1991). Only sections of the soil are typically acidified, due to the cost involved in acidifying an entire soil body, especially if it has a significant buffer capacity (Mortvedt, 1991; Shenker & Chen, 2005).

4.3.2 Application of pyrite to soils

The application of pyrite as a soil ameliorant has been studied using pyrite from widely different sources, such as mine wastes (Banath & Holland, 1976; Castelo-Branco, et al., 1999; Wallace, et al., 1976) and acidic soils (Barrau & Berg, 1977; Castelo-Branco, et al., 1999; Shamim, et al., 2010). Research has been done on both lab and pilot scale and it has been shown to be effective in improving plant growth, as measured by an increase in dry matter yield, for a variety of species.

Pyrite-rich materials have been shown to be effective in improving the plant growth on sulphur- and iron-deficient soils (Shamim, et al., 2010; Tiwari, et al., 1985; Wallace, et al., 1976). If the particle size is small enough, it is comparable with conventional sulphur fertilisers, such as gypsum, sublimed sulphur and sodium sulphate (Banath & Holland, 1976; Shamim, et al., 2010). Pyrite also tends to slow-release the sulphur and iron nutrients to the soil (Banath & Holland, 1976), due to the time involved in oxidation. This is advantageous, since it reduces the need for repeated application.

Pyrite oxidation reduces the pH of alkaline soil, which further increases iron availability. Pyrite addition to alkaline soil has been shown to improve plant nutrient uptake (Banath & Holland, 1976; Castelo-Branco, et al., 1999; Shamim, et al., 2010; Tiwari, et al., 1985; Wallace, et al., 1976), as measured by the nutrient content of the different plant organs. This is despite the fact that in very alkaline soil, the soil pH can stay stable despite the addition of pyrite (Castelo-Branco, et al., 1999; Somani, 1986; Wallace, et al., 1976). It is probable that soil micronutrients become available upon pyrite addition by pyrite particles acidifying the microenvironment within which they are located, without changing the bulk soil characteristics appreciably (Banath & Holland, 1976; Somani, 1986). As expected acidic soil was further acidified through addition of pyrite (Wallace, et al., 1976), necessitating lime addition.

Application of pyrite to soil has experimentally improved soil structure in clays (Somani, 1986; Vlek & Lindsay, 1978). The mechanism for this is probably similar to that of gypsum application to sodic soils in that the sodium precipitates as sodium sulphate. Pyrite application therefore has multiple advantages for alkaline and sodic soils.

For the pyrite to be effective as a soil ameliorant, it needs to oxidise at an appreciable rate. It has been shown that pyrite in soil oxidises much faster than would be expected if chemical oxidation was the mechanism (Vlek & Lindsay, 1978). Therefore, it is postulated that pyrite oxidation in soils is microbially mediated (Banath & Holland, 1976; Somani, 1986; Vlek & Lindsay, 1978), despite alkaline pH being unfavourable for most known sulphur oxidisers (Banath & Holland, 1976). Organisms that mediate the oxidation of pyrite under well-studied environments such as heap leaching and acid rock drainage (ARD) generation are normally most active in acidic environments (Somani, 1986). It is suggested that the soil microenvironments within which the microorganisms operate are sufficiently acidic for them to function or that some *Acidithiobacillus sp* are tolerant of alkaline environments (Banath & Holland, 1976; Somani, 1986). An increasing number of Fe and S-oxidising microbial species are being characterised (Smart et al. 2017) and evidence of micro-environments in the presence of the pyrite grains is being put forward (Govender et al., 2013, 2015)

Pyrite oxidation requires water and oxygen to take place, both as raw materials for the oxidation reaction and for microbial respiration (Schippers, et al., 1996). Application of pyrite should be close to the

surface of the soil, since soil aeration reduces with increased depth (Somani, 1986). The soil should also be damp to supply the water needed for the reaction. The rate of oxidation and dissolution will increase with increased particle surface area (Banath & Holland, 1976; Vlek & Lindsay, 1978), so when faster nutrient release is required, finer material should be applied. Banath and Holland (1976), for instance, found that material with a D_{80} of 20 μm released the iron and sulphur appreciably quicker than material with a D_{80} of 53 μm . When the pyritic material particle size is too large, for instance bigger than 3 mm, the oxidation rate is too slow to impact the soil's fertility appreciably (Sharma & Swarup, 1997).

In addition to sulphur and iron, pyrite streams from mining sources may also contain impurities. These may improve plant growth if they act as micronutrients but could be toxic to plants if excessive. Potential soil ameliorating pyrite from mining sources must be tested before application. In one pilot study in Portugal, application of pyrite to the land did not lead to toxic runoff, despite elevated levels of Cu, Pb and Zn in the waste (Castelo-Branco, et al., 1999). It also did not have any negative health effects on grazing animals (Castelo-Branco, et al., 1999). This is very material-specific and will need to be assessed for every pyrite stream.

4.3.3 Soil and agriculture in South Africa

4.3.3.1 Soil types

The soils considered in this section are either alkaline or typically have poor structure and are therefore likely to benefit from pyrite application. These soils are silicic, calcic and vertic soils. Duplex soils could be added as some duplex soils are alkaline. However, since it is not possible to pinpoint the locations of alkaline variations of the soil group, it is not considered here. Melanic soils could also be added, but they are scattered and cover only small areas. In addition to this, they can be saline and the location of the saline soils cannot be deduced given the data currently available. This section draws heavily on *Soils of South Africa* by Fey (2010) and information is from that work, unless otherwise referenced.

Silicic soils

Silicic soils, shown in Figure 4-14, are particular to dry areas, since they require arid environments to form. The subsurface layer is either hard or very hard and in South Africa it is called dorbank. Silicic soils typically have a pH between 7.5 and 9, although values between 5 and 10 are not unknown. They tend to have low electrical conductivity and high amounts of exchangeable sodium and, often, calcium. Due to the elevated pH, plant-available iron tends to be low, while the available boron tends to be so high "that it is toxic to agricultural crops" (Fey, 2010). Silicic soils are located mainly in the central and western areas of the Northern Cape Province. There are also small pockets of silicic soils in the Eastern Cape and Western Cape.

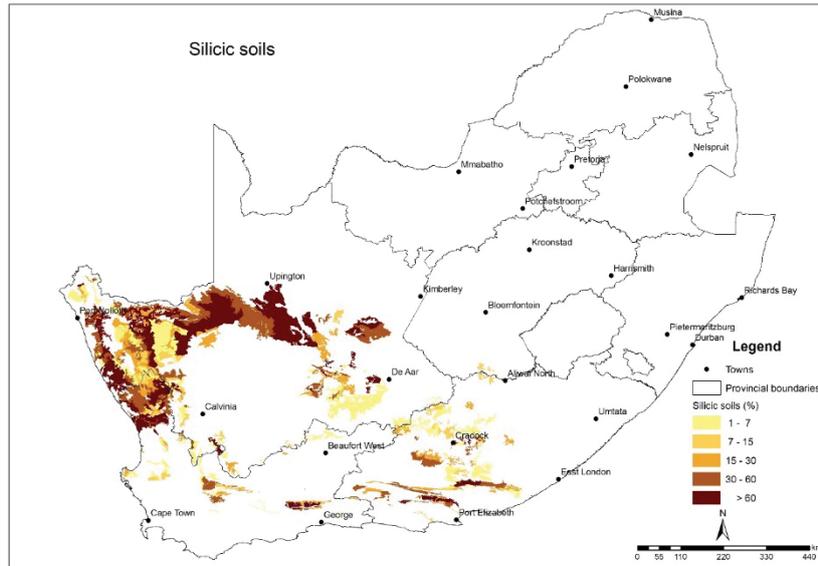


Figure 4-14 The location of silicic soils in South Africa (ARC-ISCW, n.d.)

Calcic soils

Calcic soils, shown in Figure 4-15, are mostly soils that are described as calcareous, or rich in calcium carbonate. This grouping also includes soils with a magnesium carbonate element or are formed by gypsum, rather than lime. The topsoil layers tend to have a crumb or granular structure and medium to fine texture. The subsurface layers are cemented by calcium carbonate rather than silica, as is the case with silicic soils. The soil has a neutral pH in the topsoil. In the lower layers the pH increases to between 8 and 8.5. The soil is base-rich, which means that there is ample capacity for neutralisation. As such the soils have low nutrient availability, especially with respect to iron and phosphate. The soils also suffer from plant-toxic boron levels, high salinity and stoniness. The highest density of calcic soils is located in the central and north-eastern region of the Northern Cape Province and into the south-western region of the North-West Province.

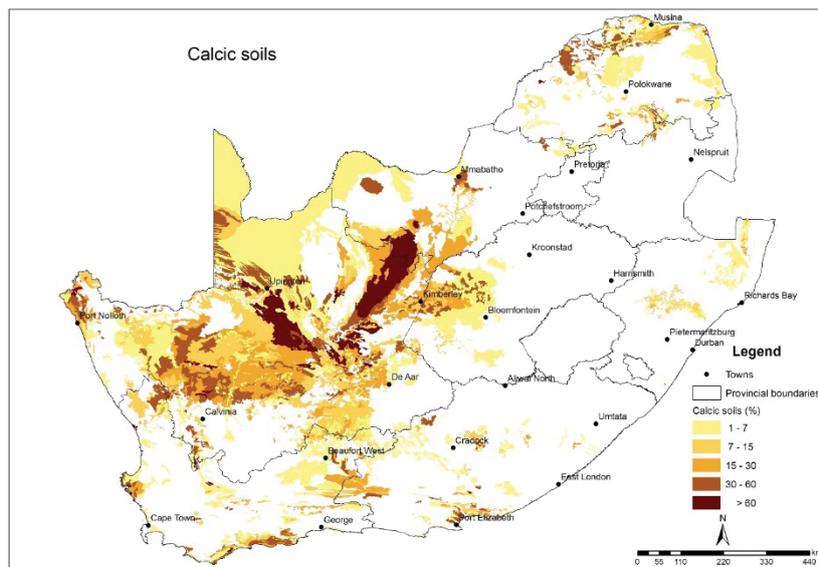


Figure 4-15 The location of calcic soils in South Africa (ARC-ISCW, n.d.).

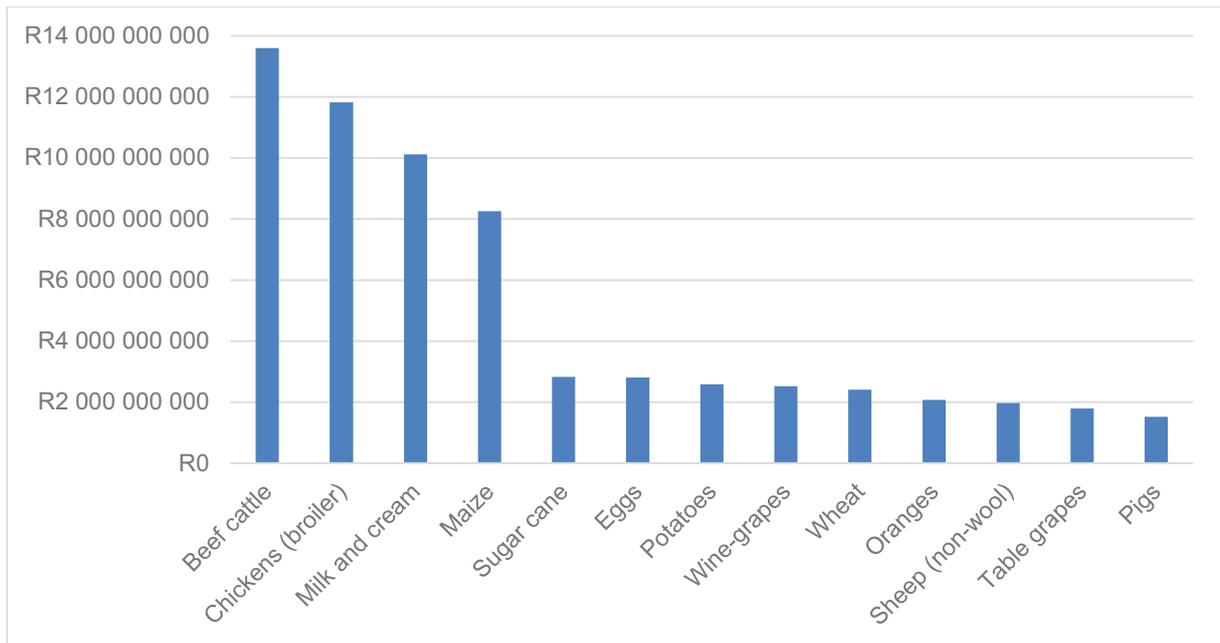


Figure 4-17 Income from agricultural commodities in South Africa in 2007 (Statistics South Africa, 2007). Only commodities with an income above R 1 500 000 000 are shown. This information, though dated, is the most complete South African data set available. The placements of different commodities should have stayed similar in this period.

Maize, grapes and different types of grasses prefer a slightly acidic soil and good soil structure helps with healthy root development (Chvyl, et al., 2010; Heiniger, 2001; Islam, et al., 2016; Malherbe, 1948). Figure 4-18 shows that grain production is mainly concentrated in the North West, Free State, Mpumalanga and Western Cape provinces. This is corroborated by the data shown in Figure 4-19 which shows that the Free State, Mpumalanga and North-West provinces are the greatest summer cereal producers. The Northern Cape, not indicated as a grain producer in Figure 4-18, does produce some grains. The data of the significant horticultural products of table grapes and wine grapes are shown in Figure 4-19 with the Western Cape and Northern Cape being the most significant producers of grapes. Figure 4-19 shows that the largest number of cattle are found in the Free State and North West provinces. The small land area shown in Figure 4-18 for cattle in these provinces does not give an indication of this.

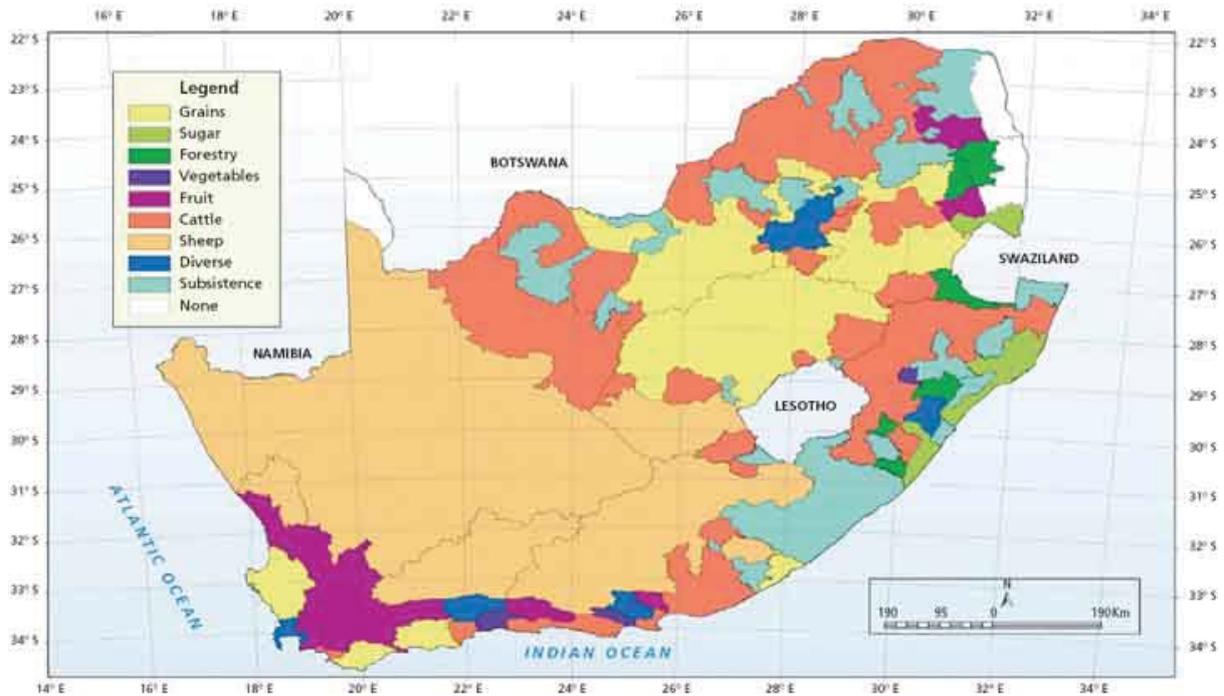


Figure 4-18 Agricultural zones in South Africa (Food and Agriculture Organisation of the United Nations, 2005)

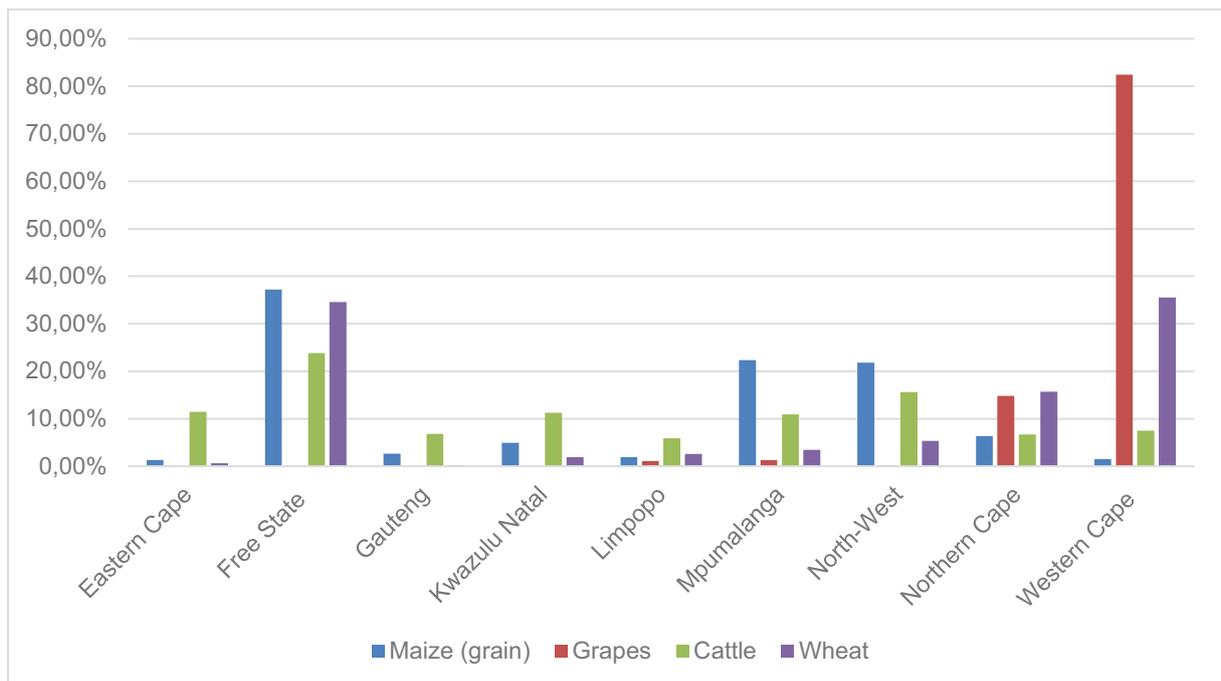


Figure 4-19 Percentage of total South African production (Statistics South Africa, 2007). This information, though dated, is the most complete South African data set available. The placements of different commodities should have stayed similar in this period, though.

4.3.3.3 Locations of South African coal mines

South Africa's coal seams are located in Mpumalanga where the historically richest seams are found, the northern Free State, Limpopo where new mines are being commissioned, and KwaZulu-Natal where coal fields have largely been depleted and active coal mining ceased. Since the mines located near Witbank, Ermelo and Middleburg in Mpumalanga are the largest operating mines, sourcing the sulphide-

rich concentrate from these areas is considered, because the processing plant(s) could be larger and achieve economies of scale.

Not all coal mines beneficiate the coal they produce. Coal mines that produce coal for export or for metallurgical use are more likely to wash the coal produced and therefore produce fine coal waste. Only beneficiating mines that produce more than 3 Mt/yr of coal are considered in analyses, because they are more likely to produce enough fine coal waste to beneficiate economically. The locations of these mines are shown in Figure 4-20. Most of the mines are located in the Witbank-Middelburg area, except for the large Grootegeluk mine in Limpopo and the small Savmore mine near Piet Retief. This will be taken into consideration when transport costs are calculated for specific locations.



Figure 4-20 The locations of mines that produce and beneficiate more than 3 Mt/yr of coal (Google, AfriGIS (Pty) Ltd, 2016; Prévost, 2011).

It is clear, when comparing locations of the soils that may benefit from application of sulphidic minerals to the location of coal mines, that there are only vertic soils in some proximity to coal mines. Silicic and calcic soils are found in areas far from the South African coal fields, in the North West, Northern Cape, Western Cape and Eastern Cape provinces. Transport cost is therefore likely to be a significant consideration for the application of sulfidic minerals to these soils.

4.3.4 Potential of sulphide-rich concentrate in soil amelioration

4.3.4.1 Characterisation of the sulphide-rich stream

Due to the variable nature of the coals used, leading to variability of minerals between mines and even within the same mine, it is necessary to use an approximation of the composition of the sulphide-rich stream for this analysis. The fine and ultra-fine coal streams in South Africa typically contain significant amounts of coal, quartz, kaolinite, pyrite, calcite, siderite, dolomite and sulphate (Kotelo, 2013; Kazadi Mbamba, 2011). Therefore, some or all of these minerals will be present in the sulphide-rich stream. Throughout experimental work on remediation and beneficiation of coal waste streams at UCT, the sulphide-rich stream has consistently been shown to be acid forming using Net Acid Generation and Acid Based Accounting static tests as well as UCT's Biokinetic test (Iroala, 2014; Kazadi Mbamba, 2011).

Table 4-4 shows results obtained for three sulphide-rich concentrates from fine coal waste thus far. From these, it is clear that the sulphide-rich concentrate is much reduced in volume compared with the original fine and ultra-fine coal stream. Coal is, however, still a significant component in the sulphide-rich stream. This stream is also likely to contain small or trace amounts of potentially hazardous elements, the mobility of which can be established using sequential chemical extraction (Broadhurst & Petrie, 2010) and must be taken into consideration when examining possible applications for this stream.

Table 4-4 Characteristics of sulphide-rich concentrate (Iroala, 2014; Kazadi Mbamba, 2011)

	Yield (%)	Ash (%)	Sulphur (%)
Stream 1	13.1	28.9	2.68
Stream 2	11.0	36.0	10.7
Stream 3	2.2	60.0	17.6

Despite the relatively low volume of the sulphide-rich concentrate compared with the fine and ultra-fine coal waste, a significant volume is generated. In 2001 the Department of Minerals and Energy indicated that the mass of tailings produced was 21% of the total coal waste produced (DME, 2001). Production statistics for 2006 placed the mass of the coal waste produced at 67.7 Mt. Hence, around 14.2 Mt of slurry waste was produced in 2006, the last year that waste production was reported (Chamber of Mines of South Africa, 2007). Coal production figures have not changed significantly since 2006, so remain relevant. These tailings were typically dewatered to 40% solids in a thickener (Jera, 2013). Modern practice is to dewater waste to around 85% solids (Fundikwa, et al., 2016). Assuming that the results achieved for Stream 3 in Table 4-4 are achievable for most fine and ultra-fine coal waste streams and modern practice, total sulphide-rich concentrate production of 0.15 Mt/y is expected. Should the yield match stream 2, then the production would be 0.75 Mt/y.

The particle size distributions of sulfidic minerals can be expected vary widely between coal mines. Researchers at UCT have encountered particle size distributions with D_{80} of between 220 μm and 68 μm on material that has not yet been processed using flotation (Iroala, 2014; Opitz, et al., 2015). The particle size distribution of the sulphide-rich stream is likely to be towards smaller particles, but has not been measured. The measured particle sizes are larger than most used in literature for soil amelioration. Castelo-Branco et al (1999), for instance, used material with a D_{80} of 30 μm and Banath and Holland (1976) found that material with a D_{80} of 20 μm was more suited for use as soil ameliorant than material with a D_{80} of 53 μm . Unless milled further, South African sulphide-rich concentrate will therefore oxidise slowly and require large application rates compared with literature in order to achieve a measurable improvement in soil condition. The exact rates will be dependent on the available surface area and will need to be assessed for individual sulfidic mineral streams. The larger particle size may also facilitate less frequent application to the soil.

Additionally, flotation will most likely remove the bulk of the sulphate present in the sulfidic minerals, since these are soluble. Many of the pyrite streams reported in literature contain some sulphate due to oxidation (Banath & Holland, 1976; Tiwari, et al., 1985). This sulphate is then quickly released into the soil and can provide short-term impact while the pyrite oxidises, thereby producing positive short and long term results. This will be a particularly important consideration in applications where the soil structure needs to be improved. The sulphidic mineral streams can be stored and incubated to increase the sulphate content.

There are other differences between a sulphidic mineral stream obtained from a beneficiating South African coal mine and the pyrite streams considered in literature. The coal component in this stream is one example. The effect that coal might have on the growth and development of plants is unclear and this would need to be tested. Transformation of coal into humic substances by microbial action has been shown (Vengadajellum, 2008); this would improve the desirability of the stream greatly through contributing to soil structure. Xanthates, which are toxic substances (Okibe & Johnson, 2002), are used

in the second stage of the two-stage flotation process as a sulphide collector, concentrating into the sulphidic mineral stream (Fundikwa, et al., 2016), hence may remain present. The natural degradation of xanthates is reviewed by Kunene (2014). Although other pyrite streams in literature are also produced using flotation (Banath & Holland, 1976; Metson, 1972), the use of xanthates is not explicitly mentioned; the effect of flotation chemicals on pyrite oxidation and plant growth needs to be investigated before application.

4.3.4.2 Costing for application in soil amelioration

Road transport costs in South Africa

Due to the long distances between economic centres in South Africa, transport cost is an important consideration when evaluating the economics of a potential solution. Freight transport within South Africa is limited to road and rail transport with pipelines being used for large volume oil transport (Simpson, 2013; Viljoen, et al., 2015). Rail transport tends to be more cost effective and more energy effective, but due to the limited capacity of the rail network, road transport is extensively used in South Africa (Simpson, 2013; Viljoen, et al., 2015). The cost of road transport is therefore considered.

The cost for road transport of a product depends on its mass as well as the travel distance and is therefore normally quoted in R/t-km (Braun & Van Zyl, 2012; Simpson, 2013), assuming the transport vehicle is filled to capacity. Both loading with a lower mass than its design and whether vehicles are full on the away and return legs of a trip influences transport cost. The cost of a vehicle to carry load one-way is roughly double use on both legs (Braun & Van Zyl, 2012), depending on the truck contractor.

In 2012, Braun and Van Zyl found that transport using a truck filled in both directions costs between R 0.27/t-km and R 0.44/t-km, depending on the truck type and the number of days used per year. In 2013, Simpson found that the cost for major transport routes was around R 0.47/t-km and in 2015 transport costs for a large truck were around R 0.39/t-km according to Viljoen et al. (2015). These figures compare well with Braun and Van Zyl's (2012) upper number. Simpson (2013) added around R 0.16 to that cost to account for externalities such as accidents, CO₂ emissions and traffic. In 2015 Viljoen et al. found that transport costs per tonne-km had stayed relatively stable between 2012 and 2015, with similar costs in December 2012 and April 2015 being applicable. This is mostly attributed to the drop in fuel price in 2015. Based on this information a conservative estimate of the 2016 road freight cost was R 0.94/t-km, since not all transport destinations allow for a full return trip.

Application rates of pyrite vs gypsum

In many applications, pyrite is likely to act as a replacement for gypsum for improving soil structure in addition to improving the pH of the soil. It is therefore useful to compare the mass of sulphidic mineral stream needed to replace the gypsum currently applied to soil.

The amounts of sulphur in dry sulphidic concentrate and gypsum are around the same. Sulphidic concentrate has two mol of sulphur per 364 g (17.6% S), assuming a sulphur content of 17.6% from Table 4-6, while gypsum has one mol of sulphur per 172 g (18.6% S), i.e. per mass the sulphidic concentrate and gypsum provide a similar sulphate load, assuming that the gypsum also carries some impurities. Since pyrite is more slow-acting than gypsum, since it needs to oxidise before the sulphate becomes available, it is assumed that double the amount of sulphidic concentrate will be used at first compared to gypsum. Frequent re-application of sulphidic concentrate should not be necessary, however, since sulphate is less likely to leach out with big rains and the large once-off application continues to be effective for longer than the small particle size, quickly depleted gypsum application. This provides advantages in terms of labour efficiency over several growing seasons.

The sulphidic mineral stream is most likely not dry. The flotation concentrate stream is in slurry form which is filtered to improve water efficiency (and waste disposal), as discussed in Sections 4.3.4.1 and 3.2.1 (Fundikwa, et al., 2016; Jera, 2013). Based on the recent trend of dewatering to 85% solids, it is assumed that the sulphidic mineral stream contains 15% water. Hence, the initial application rates of

the sulphide stream need to be at least 2.3 times that of gypsum for the same short term effect, but with longer duration.

4.3.4.3 Potential for application in specific agricultural sectors

In this section, the potential for ameliorating farming land used for producing the most important agricultural commodities is considered. These are livestock, grain (maize and wheat) and grape production. These are all important commodities in the Northern Cape, where the majority of the soils benefitting from amelioration by sulphide-rich minerals are located (Statistics South Africa, 2007). Other areas of importance are the Eastern Cape, Western Cape, Free State, Mpumalanga and the North West provinces.

Livestock farms (Northern Cape, Eastern Cape, Mpumalanga, North West)

Cattle farming is practiced in all the provinces in South Africa. In the Northern Cape and Eastern Cape large tracts of land are used for extensive grazing (Bapela & Mariba, 2002). For this reason, there is some likelihood of cattle farming being practiced in all the areas with alkaline or poorly structured soils. Although soil used for extensive grazing tends not to be fertilised or otherwise cultivated, sulphidic minerals are likely to improve the carrying capacity of the land by reducing the pH and improving soil structure. Therefore, the locations of silicic and vertic soils were compared with the locations of cattle in the country to understand where the potential market may be.

The locations for calcic soils were not used for this purpose, since calcic soils have neutral top layers, with alkalinity accumulating in the lower soil layers (Fey, 2010). Only when such soils are ripped for agriculture will the top layers mix with the lower layers and become alkaline. This is not the case with grazing and the neutral top layers should be adequate for sustaining plant growth.

Cattle farmers in the areas listed in Table 4-5 could find application of sulphidic minerals useful. It is important to remember that not all the soils around the following towns are alkaline, but that some farmers or farms are affected. Game farms located in these areas are also likely to benefit from distribution of sulphidic minerals. In the Eastern Cape relatively small areas of soil are silicic (Fey, 2010). Some farmers in the municipalities shown in Table 4-5 may therefore benefit from application of sulphidic minerals, but it is likely to be a small number in each case.

Table 4-5 The livestock-producing areas where application of sulphidic minerals may be useful and lowest transport cost from nearest coal mine area (in all cases Witbank area) (ARC-ISCW, n.d.; Google, AfriGIS (Pty) Ltd, 2016; Statistics South Africa, 2007; Statistics South Africa, 2007; Statistics South Africa, 2007; Statistics South Africa, 2007)

Province	Towns	Distance from nearest coal mine	Transport cost
Northern Cape	Britstown, Carnarvon, De Aar, Douglas, Griekwastad, Hopetown, Kenhardt, Namaqualand, Prieska, Upington	Between 719 km and 1246 km	Between R 676/t and R 1171/t
Eastern Cape	Aberdeen, Bedford, Graaff-Reinet, Kirkwood, Pearston, Somerset East, Steytlerville, Tarka, Willowmore	Between 941 km and 1163 km	Between R 884/t and R 1093/t
Mpumalanga	Bethal, Hoëveldrif, Standerton	Between 79 km and 145 km	Between R 74/t and R 136/t
North West	Brits, Rustenburg	Between 166 km and 234 km	Between R 156/t and R 220/t

Grain Producers (Free State, North West, Mpumalanga, Western Cape and Northern Cape)

Maize and wheat are the two most important grains produced in South Africa and are grown mainly in the Free State, North West, Western Cape and Mpumalanga, with some being grown in the Northern Cape, as shown in Figure 4-19.

The North West province has some calcic soils in the south-west and centre as well as some vertic soils in the north-east, which may be alkaline or have poor structure and therefore benefit from application of sulphidic minerals. The important maize-producing area is towards the centre and south-eastern border of the province (Statistics South Africa, 2007), but some farms in the areas shown in Table 4-6 are probably affected by calcic soils.

In Mpumalanga there are vertic soils in the south-west, while the Free State has some vertic soils in the north-east. Vertic soils may benefit from application of sulphidic minerals since these soils are characterised by a clay-rich A-horizon which can be relatively structureless and compact. The high base status of the soil would also guard against soil acidification. Farms in the areas mentioned in Table 4-6 may benefit from application of sulphide-rich concentrate. (ARC-ISCW, n.d.; Statistics South Africa, 2007; Statistics South Africa, 2007).

The Northern Cape has large land areas covered in silicic and calcic soils. Grain production in the Northern Cape is, however, practiced towards the very east of the province, along the Free State border and near water sources. Nevertheless, a proportion of grain producers in the areas shown in Table 4-6 are likely to benefit from application of sulphidic minerals. The areas where sulphidic mineral application to soil is useful to grain farmers are therefore small and a targeted approach to these farmers would be recommended. The exact amounts of sulphide-rich concentrate needed will vary on a farm-by-farm basis and depend on other factors such as weathering of the pyrite material and particle size, as discussed in Sections 4.3.4.1 and 4.3.4.2. It is also important to note that rainfall rates have been declining since the 2007 census and that some marginal areas may no longer produce grain, as in the case of Kenhardt (Agrimark Kenhardt, 2016).

Table 4-6 The grain-producing areas where application of sulphidic minerals may be useful and transport cost from nearest coal mine (in all cases Witbank area)
(ARC-ISCW, n.d.; Google, AfriGIS (Pty) Ltd, 2016; Statistics South Africa, 2007; Statistics South Africa, 2007).

Province	Town	Distance to nearest coal mine	Transport cost
Free State	Heilbron, Frankfort	Between 213 km and 236 km	Between R 200/t and R 222/t
North West	Vryburg	521 km	R 490/t
Northern Cape	Barkley-West, Hartswater, Hopetown, Kakamas, Kimberley, Prieska, Warrenton	Between 550 km and 1013 km	Between R 517/t and R 952/t
Mpumalanga	Balfour, Bethal, Hoëveldrif, Standerton	Between 79 km and 145 km	Between R 74/t and R 136/t
Western Cape	Oudtshoorn	1256 km	R 1181/t

The attractiveness of the sulphide-rich concentrate solution depends on the distance between the source of sulphidic minerals and the farm in question as well as the price of gypsum and distance to nearest gypsum quarry. In Hartswater, the closest agricultural region in the Northern Cape to the Witbank coal field, for instance, the transport cost for sulphidic minerals will be R 1192/t-(gypsum equivalent) and the gypsum, sourced from Daniëlskuil, will cost R 620/t with transport costs included (Kalkor (Pty) Ltd, 2016). In Kakamas, the furthest agricultural area in the Northern Cape

from the Witbank coal field, the transport of sulphidic minerals alone will cost around R 2180/t_{gypsum equivalent} as compared with the R 550/t for gypsum. Neither option leaves room for profit for the producers. In practice sulphidic concentrate probably has to cost significantly less than the price of gypsum, because farmers will require substantial cost benefits to switch from a known product to a relatively untested option.

The price dynamics are similar for farmers potentially interested in sulphidic minerals in the North West, since the distances from the Witbank coal field and Daniëlskuil gypsum quarry are comparable relatively (Google, AfriGIS (Pty) Ltd, 2016). They are also likely to be the same in the Western Cape, since Oudtshoorn is very far from the Witbank coal field. In Mpumalanga and the Free State, however, sulphide-rich concentrate should have a significant advantage over gypsum, since the transport costs from the Witbank/Middelburg coal fields are significantly lower and gypsum quarrying is unlikely to be able to compete.

Grape producers in the Northern Cape

The grape industry in the Northern Cape has grown tremendously between 2001 and 2010. It is now the second largest producer in South Africa after the Western Cape (Department of Agriculture Forestry and Fisheries, 2011). Only the grape industry in the Northern Cape is considered here, because the vast majority of grapes in the Western Cape are not grown on soil that would benefit from application of sulphide-rich concentrate.

Grapes in the Northern Cape are mainly planted in the Siyanda district, near Upington and Kenhardt (Department of Agriculture Forestry and Fisheries, 2011; Statistics South Africa, 2007). Planting is done on the banks of the Orange River with irrigation water being drawn from the river. The soil is silicic or calcic, suffering from alkalinity and the associated nutrient deficiencies (Fey, 2010; Foth & Ellis, 1996; Malherbe, 1948).

To improve soil fertility, farmers in the area use acidic nitrogen, potassium and phosphorus fertilisers (Burger, 2016; Slabbert, 2016). This, coupled with higher soil temperatures, has historically been sufficient for preventing iron deficiencies in vines (Slabbert, 2016). A large agricultural business in the area routinely uses only the traditional NPK fertilisers along with any micro-elements that may be deficient (Slabbert, 2016). Gypsum is added to soil as an ameliorant to improve soil structure and not as a sulphur fertiliser (Burger, 2016). Sulphur is used on leaves and stems as an anti-fungal agent (Burger, 2016) and additional sulphur may also reach the soil in that way.

The main reason for adding pyrite to soils is likely to be for its ameliorating effect, as a gypsum substitute. Pyrite may also be useful to prevent early-season iron-deficiency, when soil temperatures are still low. Reducing the pH of the soil may improve nitrogen and micro-element availability in soils, reducing the need for other fertilisers (Foth & Ellis, 1996). As calculated in the section on Maize farming, the initial pyrite application is proposed to be at least double that of gypsum, due to the slow oxidation of pyrite. However, a longer-term impact is expected.

The transport distance of sulphidic mine waste is significant, even if material is sourced from the coal mine nearest to grape production. The cost of transport of sulphidic materials from the Witbank coal fields to Upington will be around R 2010/t_{gypsum equivalent} and to Kenhardt around R 2192/t_{gypsum equivalent}. The cost of gypsum in the area varies depending on the source. In Kakamas gypsum sells for R 550/t excluding VAT (CA Bruwer Konstruksie, 2016) and in Daniëlskuil it costs R 440/t excluding VAT (Kalkor (Pty) Ltd, 2016). When taking into consideration transport costs to Upington the gypsum from Daniëlskuil is cheaper and costs around R 700/t excluding VAT. In these areas, the sulphide-rich concentrate costs significantly more to transport than the cost of gypsum. In practice, as stated above, for farmers to move from a known product to an untried one would almost certainly require substantial potential financial benefit. Transport cost to the grape producing regions will therefore have to be subsidised for this to be a possible area for use.

4.3.4.4 Risks in application of sulphide-rich concentrate to land

One risk in the agricultural application of sulphide-rich concentrate is the significant coal content of the stream (Iroala, 2014), which could negatively impact farms' risk profile with regards to runaway fires. This is a real consideration in the more arid regions where the concentrate would potentially be applied. Another risk is the possibility that the sulphide-rich concentrate could contain hazardous compounds, which could leach into the underground water supply and reduce yield or build up in the plants and affect the consuming organisms. Examples of such materials are xanthates used during flotation and heavy metals that may be present in the stream. Prior testing for heavy metals and their mobility is essential in evaluating suitability of the particular sulphide-rich stream for application in agriculture.

4.3.5 Conclusions with respect to soil amelioration

As expected, sulphur deficiency was not considered a problem by farmers asked about their fertilisation practices. This is probably due to the large amounts of gypsum already added as soil ameliorant. Unexpectedly, however, iron deficiency was also not considered a problem, except early in the season. This is most likely due to the acidity of other fertilisers used.

Based on the low prices for the competitor product, gypsum, and the long distances that the sulphide-rich concentrate stream would need to travel it is unlikely that application as soil ameliorant will be profitable to mines, at least in the short run. For the option to be viable it may have to be considered as a cost-effective and safe form of waste disposal. The safety of applying sulphide-rich concentrate to farmland would, however, have to be more clearly established first.

Assessment of potential use of sulphide-rich concentrate therefore needs to extend beyond the economic assessment to consider both additional advantages in soil amelioration (such as the application of double the sulphur load and the introduction of companion minerals) and to the mine in terms of removing long-term disposal.

At application rates of 6 t/ha sulphide-rich concentrate needs to be applied to around 52 000 ha of farmland annually to absorb the concentrate produced. At an average maize farm size of 2 300 ha (Steyn, 2013), this equates to 23 farms, which seems like a reachable target.

4.4 Detailed Report on Cemented Paste Backfill

In Section 4.1.10 cemented paste backfill (CPB) was highlighted as a potential use for the sulphide-rich concentrate stream. This section considers CPB as an application for sulphide-rich concentrate produced by the UCT two stage flotation process. The importance of the different components of the CPB is considered, along with potential challenges associated with the application. The potential environmental impact is also explored, and a preliminary study of the South African context is conducted. The section is then concluded with a summary of the findings and the research gaps that still need to be addressed.

4.4.1 Cemented Paste Backfill (CPB)

Backfilling of mines has been practiced since the late 1800's (Jung & Biswas, 2002). Filling of mine stopes is done for a variety of purposes: waste disposal, reducing the surface footprint of waste disposal facilities; support of underground structures like roofs, walls and pillars to prevent caving in of the mine void and attendant ground subsidence; providing extra roof support to facilitate extraction of support pillars; and providing work surface for miners underground (Belem & Benzaazoua, 2004; Jung & Biswas, 2002; Brackebusch, 1994; Kesimal, et al., 2005; Tariq & Nehdi, 2007; Rankine & Sivakugan, 2007). Mine stopes can be backfilled with waste rock, slurry or cemented paste (Landriault, 1992; Sivakugan, et al., 2006). Rock fill requires placing by mechanical means and slurry fill lacks mechanical strength. Cemented paste backfill can both provide mechanical strength and be placed using piping systems.

Cemented paste backfill is the tailings from mining and ore processing operations mixed with binder and water and used to fill open voids, called stopes, in underground mining operations (Belem & Benzaazoua, 2004; Benzaazoua, et al., 2008; Brackebusch, 1994; Jung & Biswas, 2002; Kesimal, et al., 2005). CPB contains tailings, at about 70-85 wt% solids. To this around 2-7 wt% binders, normally a mixture of Portland cement and pozzolanic materials, and mixing water are added (Belem & Benzaazoua, 2004; Benzaazoua, et al., 2008; Rankine & Sivakugan, 2007). The exact composition depends on the function of the CPB and the transport requirements of the paste.

Different CPB functions have different strength requirements. The measure of strength that is commonly used for CPB is uniaxial compressive strength or unconfined compressive strength (UCS), which is measured by compressing a sample along its length until it breaks. For example, an UCS of 0.3 MPa is sufficient for underground disposal (Fall & Samb, 2009; Tariq & Nehdi, 2007; Jung & Biswas, 2002) with 1 MPa being required if the CPB has a free-standing wall face or acts as a roof (Jung & Biswas, 2002). Similarly, the recommended strength for roof support ranges between at least 4 MPa and 5 MPa (Belem & Benzaazoua, 2004; Tariq & Nehdi, 2007; Fall & Samb, 2009). There are detailed models available for calculating the required strength of CPB for certain applications. Belem and Benzaazoua (2004) can be consulted for more information.

When more binder is added, the UCS of CPB increases (Brackebusch, 1994; Ercikdi, et al., 2009). The binder represents a large portion of the cost of the CPB. To illustrate, the cost of implementing CPB on a mine has historically been around 10% to 20% of the operating cost and the cost of cement contributes about 75% of the cost of implementing CPB (Belem & Benzaazoua, 2004; Jung & Biswas, 2002). Due to the cost of binder, the strength of CPB needs to be optimised with regard to both its purpose in order to minimise cost (Fall, et al., 2005).

Another important factor to consider in the preparation of CPB is the transportability of fresh paste (Jung & Biswas, 2002; Landriault, 1992). A characteristic of paste backfill that makes it easier to transport via pipelines is that it does not segregate, or segregates very slowly, upon being stagnant (Landriault, 1992). To achieve this characteristic, a minimum of 15 wt% of the tailings solids must be a particle size of 20 μm or smaller (Landriault, 1992). The physical properties of tailings therefore strongly influence the rheology of CPB (Jung & Biswas, 2002).

4.4.2 Water: CPB preparation and transport

The water content of CPB governs the consistency of the paste which, in turn, impacts its transport characteristics, hence it is carefully adjusted when preparing the mixture (Brackebusch, 1994). The water content required to achieve a specified consistency differs from paste to paste. CPB with a high fines content and porosity will require more water to achieve a certain consistency, since the higher surface area requires more water to coat the particles (Fall, et al., 2005; Jung & Biswas, 2002; Landriault, 1992) and particle size influences slurry viscosity. Lower water content CPB tends to have superior strength characteristics (Benzaazoua, et al., 2002; Jennings & Thomas, 2009), hence the trade-off between transportability and strength of the paste also needs to be optimised.

CPB is often transported vertically and horizontally under gravity in reticulation pipelines (Belem & Benzaazoua, 2004; Jung & Biswas, 2002; Landriault, 1992), although it is normally possible to pump CPB horizontally for up to a kilometre (Belem & Benzaazoua, 2004; Brackebusch, 1994). The pipelines generally have an inside diameter of between 10 cm and 20 cm (Belem & Benzaazoua, 2004; Brackebusch, 1994; Jung & Biswas, 2002). Transporting the paste with a method other than pipelines adds to the cost of the operation and is normally not considered.

CPB is a non-Newtonian fluid, presenting as a Bingham plastic fluid, a yield pseudoplastic fluid or a yield dilatant fluid, depending on the CPB (Belem & Benzaazoua, 2004; Jung & Biswas, 2002; Landriault, 1992; Sivakugan, et al., 2006). The fluid characteristics of CPB are therefore complex and difficult to measure since the viscosity changes with shear rate (Jung & Biswas, 2002). In practice, the

consistency and flowability of the CPB is measured using simplified approaches such as the slump test and power draw from mixing equipment.

The slump test is a standard concrete test which measures the slump of wet cemented material in a slip cone when the cone is removed (Belem & Benzaazoua, 2004; Jung & Biswas, 2002). The consistency of the paste and the slump values are correlated for specific cements. To make CPB transport via pipelines possible, paste with a slump of between 15 and 25 cm is required (Belem & Benzaazoua, 2004). An alternative measurement is the power draw of mixing equipment used in producing CPB (Belem & Benzaazoua, 2004). This is related to the consistency of the paste being mixed; when it reaches a pre-determined level, the paste is known to be the right consistency for CPB. In practice, power draw is used as a target to achieve the right consistency for transport. To understand this better, an understanding of the production process is useful.

CPB is produced by mixing dewatered tailings, binder and mixing water. CPB production requires careful control of the moisture content, since small variations in moisture can alter the rheological properties of the CPB (Brackebusch, 1994; Jung & Biswas, 2002). The tailings are often dewatered more than necessary for the CPB mixture, so that a precise amount of water can be added to achieve the desired consistency. In other cases, the tailings are left slightly wetter, but sand is added to increase the solids content of the final mixture (Landriault, 1992). CPB is also often batch-mixed, because it is easier to control than a continuous process (Brackebusch, 1994; Jung & Biswas, 2002).

The solids content is an important factor in the strength of CPB. The most important component in the gaining and sustaining of strength, however, is the binder used in producing CPB. Section 4.4.3 considers the mechanism by which binders work those used in CPB.

4.4.3 Binders: CPB cost and strength

CPB binders are often used in combination. The binder used ubiquitously for production of CPB is ordinary Portland cement hereafter referred to as cement (Tariq & Nehdi, 2007). Other binders, called pozzolans, can be added to the binder mix to improve the paste characteristics or reduce cost of CPB (Walker & Pavía, 2011). Here the functioning of cement and additional binding agents in CPB are considered, using Jennings and Thomas's (2009) work *Materials of Cement Science Primer: The Science of Concrete* are the primary reference source, unless otherwise referenced.

The most common use for cement is in the production of concrete, typically a mixture of water, cement and aggregate. Since the same properties of cement lead to stabilisation whether in concrete or CPB the term 'cemented materials' is used when referring to the finished product. Cement consists predominantly of alite (Ca_3SiO_5), belite (Ca_2SiO_4), aluminite ($\text{Ca}_3\text{Al}_2\text{O}_6$) and ferrite ($\text{Ca}_2\text{AlFeO}_5$) with other compounds present in small quantities which play a minor role in strength of the cemented material. The key components for strength of the cemented material are Ca_3SiO_5 and Ca_2SiO_4 since these hydrate (dissolve, react with water and are deposited) to form the component responsible for the majority of the properties of the hardened material: calcium-silica-water gel, typically called C-S-H gel. C-S-H gel covers the particles in the cemented material and binds them together to form a unified whole. The gel formed takes up a larger volume than the original cement particles, reducing the pore size or voidage in the final product and thereby strengthening it further, since a smaller pore size translates to a stronger cemented material (Fall, et al., 2005).

Another hydration product of Ca_3SiO_5 and Ca_2SiO_4 is portlandite ($\text{Ca}(\text{OH})_2$) which forms crystals within the pores in the material. The $\text{Ca}(\text{OH})_2$ is rigid and prevents the mixture from shrinking appreciably during curing (i.e. hardening). $\text{Ca}(\text{OH})_2$ is very soluble, however, and can dissolve when the material comes into contact with water from the environment. This increases the porosity of the cemented material, which in turn reduces its strength and allows the ingress of sulphate ions. The increased exposure to sulphate ions leads to a phenomenon known as sulphate attack which weakens the material (Section 4.4.5).

Often other materials without cementing properties of their own are added to cemented mixtures, to extend and strengthen them. These materials are called pozzolans and have a relatively high silica (SiO_2) and alumina (Al_2O_3) content (Walker & Pavía, 2011). These react with $\text{Ca}(\text{OH})_2$ cement hydration product to form C-S-H gels. More C-S-H gel is therefore formed per mass of cement added and the function of the cement is extended. Adding pozzolanic materials therefore reduces the amount of cement required and, since pozzolans are usually less expensive than cement, this reduces the cost of the total binder mixture (Jung & Biswas, 2002). By reacting with $\text{Ca}(\text{OH})_2$, pozzolanic materials reduce the presence of soluble components in cemented materials, thereby improving the durability of cement mixtures.

Pozzolans are often industrial by-products or wastes, less expensive than cement. In some cases, the positive effect and quick reaction time of a pozzolan on cemented materials have increased their demand and thus the price of the pozzolan to exceed that of the cement it replaces. Silica fume is an example of such a material.

Examples of common cheaper pozzolans include ground blast furnace slag, cement kiln dust and fly ash. The degree of amorphousness and particle surface area influence the reactivity of pozzolanic materials (Walker & Pavía, 2011) and contribute to their suitability for a specific purpose. For example, even highly amorphous pozzolans are slower to react than cement which slows the curing rate. The type of binder mixture thus impacts the strength and curing rate of cemented materials (Benzaazoua, et al., 2002; Ercikdi, et al., 2009; Kesimal, et al., 2005; Tariq & Nehdi, 2007; Coussy, et al., 2011).

The amount of water added to CPB affects not only the rheological properties of CPB (Section 4.4.2), but also the strength. A lower water to cement (w/c) ratio when mixing the paste reduces pore sizes, leading to increased strength. The water content of cemented materials is therefore important, since enough needs to be added to effect cement hydration and improve handling, but little enough needs to be added to keep the pore sizes of the cemented matrix at a minimum. Any excess water stays present in the cemented materials' pore structure.

4.4.4 Aggregate: CPB properties

The curing process and cement hydration products for CPB are significantly more complex than that of concrete (Mozaffaridana, 2011). In concrete, the aggregate is mostly inert and the hardening process is a reaction between the cement, water and any secondary chemical added to give specific properties. In CPB, the tailings used as aggregate may react with the cement and cement hydration products to produce different hydration products or to resist hydration. It is therefore crucially important to understand the interaction of specific tailings with possible binders through well-designed experiments before implementing these in practice.

It is known that the particle size distribution of tailings is important in the handling of CPB (Landriault, 1992). It is also well-known, from geotechnical engineering and soil mechanics, that material with an even particle size distribution, called well-graded, has higher compressive strength and lower porosity than material with a narrow range of particle sizes (Farokhzad, et al., 2016). It can therefore be assumed that the particle size distribution of tailings impacts structural properties of CPB and its handling. Some experimental data supports this (Fall, et al., 2005; Kesimal, et al., 2005). At some operating mines, gravel and sand are added to CPB to improve the particle size distribution (Khaldoun, et al., 2016). There is also evidence that finer material requires more binder to achieve strength, possibly due to the larger surface area needing to be covered (Farokhzad, et al., 2016; Kesimal, et al., 2005). More work is needed to establish an ideal particle size distribution for the tailings component of CPB.

4.4.5 Potential issues in CPB

4.4.5.1 Curing

An important strength-related consideration is the time required for curing of CPB. It needs to cure fast, so that the mining operations depending on the physical strength of the CPB do not need to pause

unnecessarily (Wu, et al., 2015; Nasir & Fall, 2010). When CPB is intended for use as roof support, for instance, it is required that the mixture reaches an acceptable strength soon after being placed, to facilitate the continuation of mining (Jung & Biswas, 2002). It also needs to cure slowly enough to be transported into the mine and placed without hardening. The former issue tends to be more important than the latter, due to the nature of the hydration process of Ca_3SiO_5 and Ca_2SiO_4 , together with the addition of gypsum to prevent the other important cement components from curing too fast (Jennings & Thomas, 2009).

Concrete sets and reaches a high percentage of its maximum strength after a few days, depending on the mixtures and additives, but it continues to cure for long time afterwards. For instance, Jennings and Thomas (2009) give an example in which concrete reaches 90% of its strength at 90 days curing, after which it continues to increase in strength to 100% at around 180 days. This indicates that the curing process continued for months after the cement has been placed. Ercikdi et al. (2009) have also shown that CPB can continue to gain strength even after 270 days curing. Kesimal et al. (2005), however, have illustrated that for CPB with a high sulphate content the fill strength can peak at around 180 days, after which it can decline. This highlights the importance of long-term studies for assessing the curing properties of CPB using sulphide-rich tailings before embarking on this application.

4.4.5.2 Sulphate attack

Sulphate attack refers to the loss of strength in cemented mixtures associated with exposure to sulphate. It is of specific import to CPB produced using sulphidic mine tailings, since sulphide inside the cement matrix can oxidise and lead to deterioration of the strength of the material.

To understand sulphate attack, it is important to first understand the hydration of $\text{Ca}_3\text{Al}_2\text{O}_6$ and $\text{Ca}_2\text{AlFeO}_5$. The hydration products of $\text{Ca}_3\text{Al}_2\text{O}_6$ form quickly, giving off so much energy that, if allowed, it could set the cement in less than an hour (Jennings & Thomas, 2009). This is undesirable, since cemented materials need to be transported and placed before setting. To prevent the rapid hydration of $\text{Ca}_3\text{Al}_2\text{O}_6$, gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$) is added, which leads to a different hydration reaction and the formation of $\text{Ca}_6\text{Al}_2(\text{SO}_4)_3(\text{OH})_{12} \cdot 26\text{H}_2\text{O}$ (the mineral ettringite) (Jennings & Thomas, 2009). Gypsum is normally depleted before all the original cement material can react to form ettringite. Ettringite then becomes unstable and reacts with the remaining $\text{Ca}_3\text{Al}_2\text{O}_6$ to form monosulphates (Jennings & Thomas, 2009). $\text{Ca}_2\text{AlFeO}_5$, undergoes similar reactions and the result is that most of the final hydration products are in the form of monosulphates (Jennings & Thomas, 2009).

Sulphate attack occurs when the monosulphates react with sulphates in the pore water, leading to the deposition of ettringite which takes up a larger volume than the original monosulphates (Jennings & Thomas, 2009; Kesimal, et al., 2005; Setina, et al., 2013). The resulting internal stresses in the material cause cracks and consequently reduced strength and increased ingress of substances from the environment (Jennings & Thomas, 2009). Increased exposure of the cementing materials to sulphate results, which, in turn, leads to other secondary detrimental reactions taking place (Jennings & Thomas, 2009). Sulphate attack has led to large cracks or the deterioration of CPB in some Canadian mines (Fall & Samb, 2009). This exposed the sulphidic minerals to air and water and led to spontaneous self-heating, which could lead to fire risk, especially in coal mines (Fall & Samb, 2009).

Sulphates can come from outside the CPB body, diffusing in through the pore water and weakening the cement. In the case of sulphide-rich tailings, however, sulphate is present in the residual water left within in waste body after dewatering by filtration or high-density thickening (Ercikdi, et al., 2009). This can prevent CPB from gaining strength and lead to reduced long-term strength (Benzaazoua, et al., 2002; Pokharel & Fall, 2013). Also, the sulphide minerals in the tailings body can potentially oxidise, further reducing the CPB strength from inside.

There is some indication that sulphate in the mixing water is more problematic than sulphide in the tailings. Experimental results where sulphide-rich tailings were constructed and water was sulphate-free were very promising from a longevity and an environmental point of view (Aldhafeeri, et al., 2016; Pokharel & Fall, 2013), while experimental results were less promising when mine tailings were used

that contained water containing sulphates from the mine (Benzaazoua, et al., 2002; Kesimal, et al., 2005). This suggests that tailings with high sulphate or sulphide content need to be washed before being applied as CPB. More test work needs to be done to confirm this.

To prevent or minimize sulphate attack, sulphate resistant cement (called ordinary Portland cement type V) can be used. It contains significantly less $\text{Ca}_3\text{Al}_2\text{O}_6$ than the standard Portland cement type I, (Jennings & Thomas, 2009). Also, cement mixtures which are more dense tend to provide some protection, since this restricts ingress of foreign compounds, so lower water to cement (w/c) ratios and higher cement addition rates are favourable for protection against sulphate attack (Ercikdi, et al., 2009; Jennings & Thomas, 2009).

The addition of pozzolans to the CPB mixture has also been shown to be protective against sulphate attack (Kesimal, et al., 2005; Setina, et al., 2013; Tariq & Nehdi, 2007), probably because the pozzolans consume the $\text{Ca}(\text{OH})_2$ phases, leading to denser paste microstructure and because pozzolans themselves can react and neutralise acidity and sulphates. Binders with high pozzolanic content tend to cure more slowly than pure cement binders (Jennings & Thomas, 2009; Kesimal, et al., 2005).

Despite taking precautions, acidic conditions and sulphidic conditions still seem to affect the long-term behaviour of CPB. An example of this is shown by Tariq and Nedi (2007), who used tailings containing 22% sulphur and found that after 180 days curing their backfill samples became acidic, despite using sulphide resistant cement and pozzolanic materials. These findings contradict Ercikdi et al. (2009) who found that CPB mixtures made with tailings containing 26% S and sulphate resistant cement gained strength even after 270 days. Long-term studies are needed, since no studies were found that considered the strength and sulphate-resistance of CPB over more than 360 days and only a few studies of 180 days are reported. This is important since Kesimal et al. (2005) illustrated that deterioration of the strength of samples is sometimes only seen after 180 days of curing.

Additionally, since gypsum is always added to cement (Jennings & Thomas, 2009), it would be useful to consider tests done on cement to which no gypsum has been added to assess whether the sulphate present in the tailings is consumed in the formation of ettringite and monosulphates, simultaneously, preventing fast curing and sulphate attack.

4.4.5.3 *Underground fires*

In coal mines, there is a risk of underground fires (Sahay, et al., 2014), especially when self-heating elements such as pyrite are present in the mineral. In active mines, underground fires are normally identified and stopped before they become uncontrollable (Sahay, et al., 2014), but fires in abandoned mines can burn for a long time before being discovered. For instance, the legacy underground coal mines at Kleinkopje and Landau mine have experienced underground fires for a number of years now and efforts to stop them have thus far been unsuccessful (Prévost, 2011; Van Wyk, 2014). Extremely high temperatures are possible even when fires are timeously identified and fought (Sahay, et al., 2014).

Sahay et al. (2014) have reported temperatures of up to 1058°C in experimental simulations of underground coal fires. Temperatures above 600°C have been shown to lead to the decomposition of the C-S-H gels that provide strength to cemented materials (Fall & Samb, 2009). The differences in thermal expansion between the tailings materials and cement hydrates as well as the gasification of pore water may also lead to cracking of the cement matrix, while the dehydration of C-S-H gels leads to increased pore sizes (Fall & Samb, 2009). This is expected to be aggravated when CPB is made with sulphidic tailings, since pyrite, for instance, starts decomposing and burning at 550°C, releasing more energy and increasing the temperature further.

While the effects of underground fires are serious and deleterious to CPB, fires do not occur routinely in operating mines. Since CPB is normally placed during the course of a mining operation rather than in abandoned mines, the underground conditions in operating mines must be considered for the curing and initial functioning of CPB. Conditions in filled closed mines are important in terms of longevity of CPB as a solution for long term issues.

4.4.6 Environmental impact of CPB

4.4.6.1 Environmental benefits

The environmental benefits associated with CPB are reduced or eliminated ground subsidence, a reduced surface footprint of waste disposal facilities and, possibly, reduced oxygen availability in mine workings (IIED, 2002; Lu & Cai, 2012; Tariq & Nehdi, 2007).

The subsidence of ground or creation of sinkholes near old or active mine workings is a serious problem, limiting the usefulness of land after mining has stopped (Wagner & Schuemann, 1991). This subsidence is somewhat unpredictable and depends on the method of mining employed and underground conditions, such as mine fires (Bell, et al., 2001). It is more common in mines shallower than 200 m (McCarthy & Pretorius, 2009) and where insufficient roof-support is left in the mines (Bell, et al., 2001). It is therefore specifically an issue with longwall mining (McCarthy & Pretorius, 2009) and in board-and-pillar mining with an extraction ratio above 70% (Bell, et al., 2001). Mine fires can also lead to ground subsidence and crown holes, due to consumption of the pillars (Bell, et al., 2001). This, in turn, fuels the fire by increasing air ingress into the mining cavity through cracks or holes in the ground (Bell, et al., 2001). In South Africa, underground coal mines tend not to be situated near densely populated areas, unless people have moved there after mining has stopped (Van Wyk, 2014; Wagner & Schuemann, 1991), so it is difficult to know exactly how common subsidence is or where the majority of the sink holes are located. Not knowing the locations or extent of underground fires in disused mines further complicates this. Due to the difference between South African and European rock masses and coal seam depth, European models for prediction are not suitable for use in South Africa (Wagner & Schuemann, 1991).

Ground subsidence can affect surface infrastructure (Skarzynska, 1995; Haibin & Zhenling, 2010). For instance, roads that cross old underground mines may need to have further support structures put in place to ensure structural integrity and safety of road users (Bell, et al., 2001). It can also affect the agricultural potential of land by changing the topography, making it unsuitable or dangerous for livestock or people and affecting the water supply (Van Wyk, 2014). Ground subsidence changes the flow-path of surface water (Skarzynska, 1995; Van Wyk, 2014). For example, it can create local dips in the ground surface which divert local streams or water courses with pooling of water in the subsided area (Skarzynska, 1995; Van Wyk, 2014; Haibin & Zhenling, 2010). The water in the mine void can become polluted and acidified through contact and reaction with the mine walls and materials left in the mine voids (Van Wyk, 2014). Avoiding surface subsidence is thus an environmental imperative and CPB can be used to effect that (Belem & Benzaazoua, 2004; Bell, et al., 2001).

Another potential positive impact of filling underground mine stopes with CPB is reducing the amount of air in the stopes, thereby reducing the risk of underground fires and ARD generation in coal mines. This will be especially applicable in coal mines that are closed or nearing end of life, since these are more likely to have uncontrolled underground fires due to lower levels of supervision. A mine would need to be completely filled, however, to remove the risk of underground fires. The lower levels of air and gas mobility in underground stopes of a completely filled mine should also reduce the ARD potential of sulphidic tailings, due to low oxygen availability. The efficacy of this approach will depend in large part on the long-term stability of the CPB in a completely filled mine and the movement of water in the underground environment, since leaching away of components could potentially open up air pockets inside the abandoned mine workings.

4.4.6.2 Environmental risks

Old mine workings have been known to flood and contaminate ground water resources (Van Wyk, 2014; Haibin & Zhenling, 2010). It is therefore important to understand how CPB chemical and physical stability in water is different from that of raw tailings. It is thought that CPB reduces the reactivity and leachability of tailings by encasing it in a dense cement matrix (Ercikdi, et al., 2009; Pokharel & Fall, 2013). There is some indication from practice that CPB may sufficiently isolate hazardous components

in tailings to prevent negative environmental impacts (Jiao, et al., 2011). The resistance of CPB to leaching therefore relies on the density and durability of the cement matrix.

Experimental studies on tailings containing arsenic, however, have shown that arsenic leached and continued to leach from CPB samples that were cured for a month and then immersed in deionized water for two months (Coussy, et al., 2011). Different binders did give different results, however (Coussy, et al., 2011), indicating that the cement fraction in CPB may not be an effective barrier in preventing water access to hazardous components, depending on the binders used. This depends on the complexes of the hazardous components with the cement hydrates, the pore sizes of the cement matrix, hydraulic conductivity and the long-term stability of CPB (Pokharel & Fall, 2013). Increased pore sizes and hydraulic conductivity lead to increased infiltration of water and compounds from the environment outside the CPB and also to increased diffusion of hazardous components from the CPB to the environment (Coussy, et al., 2011; Pokharel & Fall, 2013). Similar effects arise due to reduced stability of CPB and cracking.

An important determining factor of CPB stability is the sulphate content of the tailings and mixing water used to prepare CPB, since, as was mentioned in Section 4.4.5, sulphate attack reduces the strength and durability of cemented materials significantly. In addition, sulphides inherent in tailings may oxidise and produce even more sulphate ions and ARD. Proper design and long-term testing of the CPB in cases where high sulphide or sulphate content is present is therefore imperative, especially in cases where sulphate is present in the tailings water.

Lastly, the loss of material that could potentially have been reworked extracting additional value is a possible impact of CPB. This scenario depends on future metal prices and technology options, however, and is an environmental risk with any permanent waste disposal method considered.

4.4.7 Considerations for implementation on SA underground coal mines

4.4.7.1 Underground conditions in SA coal mines

South African underground coal mines are not particularly deep, with most of the mines under 200 m in depth (Vermeulen & Usher, 2006; Wagner & Schuemann, 1991) and the deepest being just shallower than 500 m (Mey, 2016). At these depths, excessive ambient heat is not a problem as it is with South African deep-level gold mines. The average underground temperature is around 20 °C (Hand, 2016), partly also due to the good ventilation systems in underground coal mines. This underground temperature compares well with international figures mentioned in CPB literature (Benzaazoua, et al., 2008; Coussy, et al., 2011; Fall, et al., 2005; Tariq & Nehdi, 2007) and means that experimental results should be more easily extrapolated to South African coal mine environments.

Low curing temperatures negatively affect cement hydration rate (Jennings & Thomas, 2009) and temperatures below 20°C have been shown to lead to increased hydraulic conductivity and pore size, which impact environmental characteristics of CPB as discussed in Section 4.4.6 (Pokharel & Fall, 2013). The increased pore size of cement cured at low temperatures will also probably mean that the strength of the CPB will be less than when cured at higher temperatures. This is corroborated by similar effects being witnessed in concrete (Jennings & Thomas, 2009). Lower underground temperatures do, however, lead to lower reactivity in the sulphidic phases which reduces the rate of sulphide oxidation (Aldhafeeri, et al., 2016). Lower sulphide oxidation rates lead to less sulphate being produced, which improves the longevity of the CPB, as discussed in section 4.4.5 and reduces the likelihood of ARD production from CPB becoming a problem in the medium to long term.

Due to the good ventilation, humidity in underground coal mines is also less than in many other underground mines. High humidity is good for cement curing, because higher humidity means that the CPB will cure evenly, with minimal shrinkage of the outside layers due to drying (Jennings & Thomas, 2009). However, with humidity reported at around 70% (Mey, 2016) it compares well with curing humidity values used in literature (Benzaazoua, et al., 2002; Fall, et al., 2005; Kesimal, et al., 2005). The humidity in underground coal mines should therefore be sufficient for even curing of the CPB.

Underground coal mines can be wet environments, due to the seepage of water into mine stopes (Vermeulen & Usher, 2006). The water ingress could be associated with precipitation percolation, underground water courses intersecting the mine void and leakage from underground aquifers into the mine voids (Vermeulen & Usher, 2006). Many mines will consequently be completely or partially flooded if not dewatered (Van Wyk, 2014). Decommissioned mines tend not to be dewatered and CPB will therefore have to be able to withstand periodic wetting or long-term immersion in water, without leading to the leakage of hazardous components inherent in the mine waste to the surrounding environment.

4.4.7.2 Applicability of literature results to the South African case

The sulphide-rich concentrate can still contain a significant coal component, up to 40% (Kazadi Mbamba, et al., 2012; Iroala, 2014). This leads to uncertainty as to how the results achieved in literature with tailings from hard rock ores will translate to that of coal tailings. Coal, for instance, has a much lower UCS than rocks (Wagner & Schuemann, 1991) and the interactions of organics with cement hydration products is unclear. Further testing is therefore imperative to establish the appropriateness of using CPB with tailings from coal mines.

Another potential issue with implementing CPB in South African coal mines is the presence of sulphate in the concentrate stream water due to the oxidation of sulphides, as discussed in Section 4.4.5. The sulphate in the water of sulphide-rich concentrate will be diluted during flotation, but the extent of this depends on the number of flotation steps and specifics such as pulp density used in the flotation rig. The sulphate content of the water in sulphide-rich stream will have to be tested and the stream assessed for appropriateness of use as CPB.

Lastly, the pozzolans available to South African mines as additives are different from those which have been used in literature. The differences in processes and virgin ores will affect the characteristics of cement kiln dust, blast furnace slag and fly ash. These differences will be present even between different plants in South Africa. The applicability of literature results with regards to these materials will therefore need to be confirmed on a case-by-case basis.

4.4.8 South African coal mine scenario

In order to make calculations for the South African scenario many assumptions need to be made, since there are a lot of unknowns at this early stage of technology development. In a previous project at UCT the two-stage flotation process achieved a sulphide-rich concentrate stream with a sulphide content of 17.6% (Iroala, 2014), which is about 2.2% of the mass of the original coal waste stream. This sample's characteristics were used in the calculations below. As shown in Section 4.1, the total sulphide rich stream in South Africa is assumed to be around 0.15 Mt/yr. This figure assumes that the original figures for the coal waste stream included 60% water (Jera, 2013), since it was assumed that conventionally most tailings are disposed of after thickening. The 0.15 Mt/yr figure also assumes that the concentrate has been dewatered to 85% solids using a filter press, as per modern standard practice (Jera, 2013; Fundikwa, et al., 2016). Where daily volumes were used, a capacity factor of 83% was assumed.

4.4.8.1 Volume of CPB produced and used

The volume of coal waste produced at different locations is important to know in order to estimate the potential volume of CPB which would be available. The mass of coal waste produced in South Africa can be calculated from available data, but the volume depends on the density of the tailings which varies greatly, based on the minerals present (Fall, et al., 2005; Rankine & Sivakugan, 2007). Pyrite, for instance, has a density of 5 t/m³, while the density of South African coals varies between 1.3 t/m³ and 2 t/m³ (Kazadi Mbamba, 2011; Iroala, 2014). Other important gangue minerals are kaolinite and quartz with densities of 2.6 t/m³ and 2.65 t/m³ respectively. A theoretical coal waste density, based on the assumptions mentioned above, is therefore taken as 2.6 t/m³.

The density of CPB cannot, however, be calculated based on a weighted average of the different CPB components. Mozaffaridana's (2011) experimental results, for instance, show that the density of CPB

was lower than the weighted average of the different CPB components. This indicates that other factors, such as air entrainment, probably influence the density of CPB.

The amount of sulphidic minerals in the CPB will also dictate the choice of binders that will be used, which will likely differ from mine to mine depending on the sulphidic mineral stream characteristics as well as its intended purpose. The density of CPB is therefore difficult to predict and experimental results are needed to have any certainty.

In order to attempt an estimation of the volumes of CPB likely to be produced in South Africa annually, a scenario is postulated where all the sulphide-rich concentrate potentially produced is used in a single application. Two different potential binder mixes, used in section 4.4.3, are then considered and the corresponding CPB volume calculated (Table 4-7). Two CPB densities from literature, 1.86 t/m³ and 2.15 t/m³ (Rankine & Sivakugan, 2007; Mozaffaridana, 2011), were used in the calculations. This gives the range within which the volume of CPB potentially produced in South Africa is likely to fall.

Table 4-7 The total volume of CPB to be produced in South Africa if all the possible sulphide-rich streams are used as CPB

	Binder %	Tot mass of CPB produced t	Tot volume of CPB m ³ /yr (density1*)	Tot volume of CPB m ³ /yr (density2*)	Volume of CPB m ³ /day (density1*)	Volume of CPB m ³ /day (density2*)
CPB mix 1	4	153 186	82 358	71 249	276	238
CPB mix 2	23	181 172	97 404	84 266	326	282

*density1 = 1.86 t/m³; density2 = 2.15 t/m³

This means that the total volume of CPB produced using sulphide-rich concentrate would be 200 to 350 m³/day, assuming all the coal waste was processed at a single location. Australian CPB operations are reported to backfill at a rate of between 20 and 200 m³/h, which is between 480 and 4 800 m³/day at 24 hour operation (Rankine & Sivakugan, 2007). However, even assuming that the extracted coal only is replaced, the largest underground coal mine in South Africa, Matla colliery (Prévost, 2011), will require between 16 000 m³ and 19 000 m³ of CPB to fill the void of the coal mined per day. Similarly, a mid-sized underground coal mine, Greenside colliery (Prévost, 2011), will require between 5 000 m³/d and 6 000 m³/d and a small underground coal mine producing around 0.6 Mt per year of coal will require around 1 000 m³/d of CPB. These figures indicate that CPB using sulphide-rich concentrate from coal waste streams would be substantially insufficient in terms of supplying the potential need for backfilling any South African coal mine. In the light of this, no costing of the process is included here.

4.4.9 Conclusions with respect to cemented paste backfill

The application of CPB has potential benefit for South African underground coal mines, both in terms of allowing safe extraction of support pillars and with respect to mitigating some of the long term environmental issues arising from the mining operation. CPB has potential to neutralise the harmful potential of the sulphide-rich concentrate produced when fine coal waste is processed through a two-stage flotation, but there are still several research issues outstanding before this can be stated with certainty. In particular, the long term stability of the cured sulphide-containing CPB has not yet been demonstrated, the effect of coal particles in the aggregate has not been sufficiently quantified, and the possibility of using Portland cement containing no gypsum has yet to be investigated at all. However, it is clear that this stream is far too small to play a role in beneficial CPB and it is suggested that any future research be directed to CPB using of the benign tailings or, possibly more economically attractive, the bottoms stream from the single flotation of the coal waste stream. The potential role of the sulphide in the fine coal waste as a source of low release sulphate to contribute to slow curing of cement has still to be investigated.

4.5 Detailed Report on Manufacture of Ferric Sulphate

The multiple criteria decision-making (MCDM) methodologies used in Section 4.2.4 prioritised the study of the two options for further use of the sulphide-rich concentrate which do not require further processing of the stream prior to use, namely as a soil ameliorant (Section 4.3) and as the agglomerate in cemented paste backfill (Section 4.4). Here the manufacture of ferric sulphate from the pyrite in this stream, selected as third of seven options (Figure 4-12 and Figure 4-13), is investigated further. The disadvantage of this use compared to the previous two is that it requires further processing with concomitant costs and management demands. The advantage is that the pyrite is completely removed from the environment, with a secondary advantage in the higher value of ferric sulphate as a product which may lead to value recovery.

4.5.1 Uses of ferric sulphate in South Africa

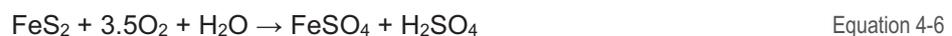
Ferric sulphate, $\text{Fe}_2(\text{SO}_4)_3$, has uses in a number of industries, including use as a mordant in dyeing (Pisitsak, et al., 2016; Mirjalili & Karimi, 2013), in the manufacture of pigments (Armanini, et al., 1979; Penniman, et al., 1921), and in the pickling of metals (Percival, et al., 1941). It is also used as an additive to animal feeds (Fritz, et al., 1970; Kim, 1998; Harrison, et al., 1992) and in purer forms in medical and dental applications as an autohaemorrhaging agent (Stringhini Junior, et al., 2015; Nouri & Sharif, 2015). However, the most likely application in the case of production on site from mine waste is as a coagulant in waste water treatment (Aguilar, et al., 2003; Tetteh, et al., 2017) since this would allow it to be used in house for existing waste water treatment.

In WRC report TT 405/09 (2009), Leopold and Freese include ferric sulphate in their report on chemicals for wastewater treatment. It is used extensively in South Africa, but most commonly in the Cape Town area since that is where the majority is produced (Protea Chemicals, 2017), while in the Gauteng area ferric chloride is more commonly used, being produced in that area. They report that, for water treatment purposes, ferric sulphate is usually sold as a solution with the price related to the strength of the solution in terms of iron. The iron content is usually between 8% and 14%. The ferric sulphate in South Africa is usually produced from iron oxide, specifically in the form of hematite (rust), which is available as a waste material from most steel works. The process is problematic since it requires the highly corrosive sulphuric acid raw material, operates at high temperature and is usually pressurised.

In their discussion of ferric sulphate, Leopold and Freese (2009) note that the iron for all processes is a by-product of other processes and the main cost factor is therefore the sulphuric acid required. The Molbase platform for chemical e-commerce lists 29 manufacturers (Molbase, 2017), though this does not include South Africa, and many more suppliers. The reference price given is US\$ 57/kg, with the individual pricing varying from US\$ 250/t for 11% purity to US\$ 350/kg for 99% purity. Alibaba.com list only suppliers, without a manufacturer subdivision; the prices for ferric sulphate (Alibaba, 2017) per tonne vary from US\$ 220 to US\$ 400.

4.5.2 Ferric sulphate from pyrite

Pyrite is one of the main minerals in the sulphide-rich concentrate and has the formula FeS_2 with the iron existing as ferrous ions (Fe^{2+}). In the right environment, the following reactions of the pyrite take place (Salomon-De-Friedberg, n.d.; Rawlings, et al., 1999):



The conditions to which the pyrite is subjected determine which of these reactions dominates. In order to produce predominantly ferric sulphate from pyrite, an appropriate process must be chosen. Two

options are presented: bioleaching and pressure leaching, which are both adaptations of processes used for metal recovery from pyrite-rich ores.

4.5.3 Ferric sulphate from pyrite by bioleaching

One of the available processes for production of ferric sulphate from pyrite is bioleaching. In this process, in addition to imposing physical conditions which promote this conversion, iron-oxidising bacteria are used to drive the process towards producing ferric ions

At Universidade Federal do Rio Grande do Sul, a team demonstrated the technical feasibility of producing ferric sulphate suitable for use as a coagulant from a pyrite fraction of coal tailings using the microorganism *Acidithiobacillus ferrooxidans* (Menezes, et al., 2016; Menezes, et al., 2017; Colling, et al., 2011). The bench-scale experiments processed coal tailings samples (51% pyrite) inoculated in packed bed column reactors to reach a maximum extraction rate after 8 weeks with recirculated water (Colling, et al., 2011). The solubilised iron (9.6%) was thereafter filtered and evaporated to increase the concentration to match trade product concentrations at just over 12% (Colling, et al., 2011). Consequently, only 4% of the available pyrite in the coal tailings sample was necessary to achieve the desired ferric sulphate grade for commercial purposes. Demonstrating its efficacy as a treatment agent, the experimentally produced ferric sulphate performed comparably to a commercially available product in water purification tests, treating water to well within the Brazilian drinking water standards (Colling, et al., 2011).

In the techno-economic feasibility study performed in Chapter 3, only one set of experimental data (Table 3-3) has sulphur concentrations in the sulphide-rich concentrate of the same order of magnitude as the pyrite-rich tailings used in the Brazilian studies (Menezes, et al., 2016). It is important to recognise that target ferric sulphate concentrations exiting the bioreactor must be within acceptable ranges to make this a competitive product and an economically viable option. Optimisation at bench-scale to obtain effluent liquors that are suitable as feedstocks to this proven technology would therefore be needed.

4.5.4 Ferric sulphate from pyrite by pressure leaching

An alternative process for the conversion of pyrite to ferric sulphate is pressure leaching. This process is primarily applied in the recovery of valuable metals in which ferric sulphate is a by-product of the leaching process (Salomon-De-Friedberg, n.d.; Papangelakis & Demopoulos, 1991; Long & Dixon, 2004).

The combination of temperature and pressure applied in these systems determine which iron and sulphur reactions predominate (Section 4.5.2). Ferric ions are favoured at high pressures with oxygen partial pressures up to 1035 kPa O₂, small particle sizes ranging from -53 +44 µm and at high temperatures up to 230°C (Long & Dixon, 2004). Under these conditions, dissolution reactions are accelerated. Compared with low temperature, atmospheric bioleaching systems, steady state in pressure leaching operations is reached several orders faster but requires more rigorous process conditions and control.

An investigation into the optimisation of the process for the purpose of producing ferric sulphate in an adequate solution for use in the water treatment industry is still needed. However, the technology is all in existence and in use in other parts of the mining industry (Salomon-De-Friedberg, n.d.; RPM, 2015).

4.5.5 Available resource in sulphide-rich stream

This method of dealing with the sulphide-rich stream could potentially be integrated into the two-stage flotation plant to convert the acid forming stream directly into the usable product, thus not posing any ARD risk.

In Chapter 3, the four coal waste plants were presented with an average sulphur content of 0.91% amounting to about 6 552 tpa sulphur for a plant that processes 100 tph coal waste. If it is assumed that all sulphur in these feed streams are in the form of pyrite, then about 12 260 tpa is available for

conversion. On average 33% of the sulphur is recovered to the sulphide-rich stream therefore nearly 6 700 tpa ferric sulphate, or 10 000 m³/annum of a 12% solution, could be produced if complete conversion was achieved.

For standard dosages of 300 mg/L of ferric sulphate (Camacho & Huerta, 2013), approximately 60 ML/day of wastewater could be treated using observed 2-stage flotation process capacities. For the integration of this process to be a successful ARD mitigation strategy, safe disposal or reuse of the residual fractions will only be achieved if the sulphur content in the waste stream is below 0.5%. For more dilute streams approximately 50% of the sulphide-rich streams would need to be converted to meet these requirements while conversions up to 96% of the more concentrated streams presented in Chapter 3 would be needed to minimise ARD risk.

4.5.6 Conclusions with respect to manufacture of ferric sulphate

The production of ferric sulphate from the sulphide-rich concentrates is an attractive option to valorise waste while simultaneously eliminating associated waste management costs. Used either within the coal mining sector or in partnership with water authorities, ferric sulphate may be used as a coagulant in wastewater remediation. Implementation of this process presents real potential for this product to be revenue generating should production outstrip internal use.

However, there are several factors which need further investigation before this option could be exploited. Technologically, both of these production routes must be further researched at bench scale. Bioleaching has been demonstrated as a viable option for the production of ferric sulphate for water treatment, while the feasibility of pressure leaching still needs to be established for the production of this product. In either case, optimisation studies are needed before either of these technologies are implemented. Design factors cannot be determined without this additional information. In addition, as part of a techno-economic study, potential usage needs to be scoped and the market properly evaluated.

4.6 Reviewing Options for Re-purposing and Potential Valorisation of Sulphide-Rich Concentrate

One of the keys to making the two-stage flotation process a truly revolutionary method of dealing with sulphide-containing mine wastes is the exploitation of the sulphide-rich concentrate. The importance is two-fold. Firstly, the industrial ecological imperative of removing the ARD potential of all streams exiting the process can only be realised through application of this stream in a manner which makes use of the pyrite in it. Secondly, the economic burden of the ARD mitigation process, while alleviated by the sale of the clean coal concentrate (Section 3.4.1), could be further eased or even removed by the valorisation of the sulphide-bearing stream.

Investigation of the plethora of potential industrial uses for pyrite yielded a list of thirteen active options to pursue. Multicriteria decision-making process theory was applied to these, identifying a number of criteria and having interested stakeholders rank them. Aggregation methods resulted in seven top-choice processes in ranked order. The top three options included its use as a soil ameliorant, aggregate in cemented paste backfill and manufacture of ferric sulphate. These were further researched.

With further investigation, these options could be practically pursued. As a soil ameliorant, the application of the sulphide-rich concentrate would need to be demonstrated to be beneficial and environmentally safe in each soil where application is considered within the different geographical situations. The long-term efficacy of the use of sulphide-rich aggregates in cemented paste backfill has yet to be adequately demonstrated. The repurposing of existing mining-industry technology for production of ferric sulphate still needs to be validated.

In terms of quantities, for use as a soil ameliorant there should be no shortage of suitable land near the coal mining regions if farmers can be persuaded of its benefits. Depending on the overall conversion

of pyrite to ferric sulphate, product outputs may exceed on-site utilisation. Assessment of the market would therefore be needed to investigate the potential of this product as a saleable commodity. The contribution which this stream could make to cemented paste backfill is insignificant in terms of total possible need for this application.

These two factors together suggest that cemented paste backfill should not be pursued as a use for the sulphide-rich concentrate, but rather as an option for the bulk benign tailings stream.

Economically, application as a soil ameliorant is not a direct valorisation of this stream. It may, however, be a worthwhile method of safe disposal, either breaking even or using the cost of transport in exchange for ARD mitigation. The production of ferric sulphate would be a definite valorisation of the sulphide-rich stream if the process is adequately optimised and appropriate trade markets are identified and targeted.

5 APPLICATIONS OF SULPHUR-LEAN TAILINGS: AN INDUSTRIAL ECOLOGY APPROACH

In this project, we seek not only to reduce the environmental burden of fine coal waste, mostly associated with generation of acid rock drainage, but also to enhance resource efficiency and reduce long-term land use and liabilities by re-purposing the 'waste' products to materials of value. Ideally this is achieved with valorisation; however, the re-purposing of these materials to replace the utilisation of other natural resources is also beneficial, particularly with the removal of the need for land for disposal. Owing to the low sulphur tailings stream being the largest by volume, its re-purposing is an important consideration. Thus, uses for the benign tailings stream are explored to move toward a zero waste system. In this chapter, we explore options for re-purposing the low-sulphur stream produced from the two-stage separation process described in Figure 3-1, after thickening.

It should be noted, that not all sulphur-lean tailings are benign, since the non-acid forming ore may contain other potentially toxic components such as some metals. This requires characterisation prior to implementation of the re-purposing approaches explored below.

5.1 Re-purposing of Sulphur-Lean Tailings

In the WRC K5/2215 report evaluating the benefits of this approach of coal waste desulphurisation by two-stage separation, Harrison et al (2013) reviewed potential applications for the low sulphur tailings fraction obtained from the sulphide flotation stage of the two-stage separation process under investigation. These included its application in metals recovery, cement additives and construction materials for roads, environmental remediation systems and in the construction industry as bricks and blocks. This has been extended to include potential to fabricate soils and for use in geopolymers (Table 5-1). Two options have been selected for further consideration in this report, with further studies undertaken in the current WRC project, WRC K5/2761.

Owing to the need to also consider sulphide-bearing waste rock, the co-disposal of this sulphide-lean fraction carrying neutralisation potential with acid generating waste rock is considered. Although fine mine waste has been used in covers for waste rock dumps previously, the use of de-sulphurised tailings has not been considered, nor has rigorous analysis of the stacking of the waste rock dump, means of application of the tailings and waste rock dump integrity been considered. This concept has been furthered as part of this project, with laboratory studies complementing the literature review (Section 5.2).

The second option considered involves application of the separated waste fractions in soils as a main substratum for fabricated soils (Firpo, et al., 2014; IUSS, 2014; González Cañibano, 1995; Li, et al., 2014; Huang, et al., 2015). These fabricated soils are then available for rehabilitation of the mine site, in preference to the burrowing of topsoils. Here a comprehensive literature review of relatively new concept of soil fabrication from waste materials as well as the first study on soil fabrication from desulphurised coal wastes from South African coal mines is reported.

The third major application of potential, brick manufacture, is studied in the project WRC K5/2761, undertaken in the same laboratory.

The advantage of the two applications presented in this chapter is that the tailings can be used within the mine environment for responsible mine management and closure. The co-disposal forms part of the broader approach to minimisation of ARD. The fabricated soil is used for the rehabilitation of the mine land. While important at all mine sites, it is particularly challenging in open cast mining environments which are common in South African coal mining. The on-site use of the mine 'waste'

material overcomes a number of legal complexities in South African law on the re-purposing of waste materials, particularly when crossing site boundaries.

In addition, the application of cemented paste backfills utilising sulphide mineral concentrates or benign desulphurised tailings as aggregates, have successfully been demonstrated to both support mine voids and limit the diffusion of oxidants to the active sulphide mineral sites (Food and Agriculture Organisation of the United Nations, 2005; Belem, et al., 2000; Broadhurst & Petrie, 2010; Coussy, et al., 2012). CPB has been thoroughly reviewed as an option for use of the sulphide-rich concentrate in Section 4.4. Owing to the smaller volumes of sulphide rich material being insufficient for CPB in South Africa and the lack of rigorous data on the fate of the sulphide under these conditions with potential for sulphate attack of the backfill, it is preferable to use the sulphide-lean tailings stream for CPB.

Table 5-1 Potential uses of the benign tailings fraction. Extended from Harrison, et al. (2013)

Potential Use of Benign Tailings	Comments	Reference
Co-disposal with acid producing rock	The aim is to prevent ARD. May be through covers or by layering.	Section 5.2
Fabricated soils	The aim is to provide soil suitable for revegetation or agriculture, with possible use in rehabilitation of mine sites	Section 0
Cemented paste backfill (CPB)	This option is considered in depth as an option for use of the sulphide-rich concentrate.	Section 4.4
<i>Construction</i> Constructed wetlands	For silicate-rich ores with no metal deportment.	Schellenbach and Krekeler (2011)
<i>Construction</i> Road construction	For silicate-rich ores with no metal deportment. Can be used for unpaved roads, as road substrate or as filler for embankments.	Agrawal, et al (2004) Pappu, et al (2007)
<i>Construction</i> Manufacture of composites, including bricks, blocks and tiles	Depending on mineralogy can be used to manufacture bricks, blocks or tiles	Taha, et al (2017) Roy, et al (2007) Celik (2015) Castro-Gomes, et al (2012)
Cement clinker	For ores with limestone and alumino-silicate content with no toxins or metal deportment. Clinker is a sintered material occurring as nodules in the Portland cement.	Celik, et al (2006) Sobolev and Arikian (2002)
Glass wool and ceramics	For silicate-rich ores. Glass wool is used for thermal and acoustic insulation. Produced by melting and then spinning (for wool) or pouring (for ceramic).	Marabini, et al (1998)
Recovery of secondary metals	Depends on gangue mineralogy, but especially focused on rare earth elements.	Binnemans, et al (2015) Michalková, et al (2013)
Manufacture of geopolymers	Depends on gangue mineralogy, but manufactured from silica-rich ores s amorphous inorganic polymers. Used in waste encapsulation, as new cements and as heat-resistant materials	
Soil fabrication for rehabilitation and further use	Creation of top soil using mine waste as the base material	Firpo, et al. (2014)

5.2 Sulphide-lean Tailings Used in Co-disposal with Acid Producing Rock

One of the reported means of ARD prevention is by co-disposal of waste rock with materials that impose diffusional limitations by either covers or barriers. These types of barriers may include simple applications such as water or oxygen consuming organic compounds. Increasingly, co-disposal of acid-producing mine waste with low-sulphur, non-acid-generating fractions produced on mine sites are investigated in ARD prevention strategies (Bussiere, et al., 2004; Demers, et al., 2008; Maddocks, et al., 2009; Gautama, et al., 2010). As these fractions are low in sulphur and are objectively benign, they act as permeable reactive barriers limiting the infiltration of oxidants to the sulphide mineral surface and further restrict transport through porous ore channels.

Frequently, ARD prevention studies have been performed at near neutral conditions utilising mostly air and water intermittently introduced to the test system. The batch-like experiments have seldom been performed over long periods and often ignored the influence of extended flow conditions on ARD generation. Further, accelerated leach conditions in acidified environments or in the presence of more aggressive oxidants, such as, aqueous ferric ions or in the presence of microorganisms that catalyse these reactions, have not been considered when determining the long-term efficacy of the mitigating procedure, despite the fact that it is well-accepted that these do become prevalent with time in the presence of sulphidic minerals. While studies relating to influence of increased bed surface area on leaching reactions have been reported, very little has been demonstrated in ARD applications (Erguler, et al., 2014).

In this study, the efficacy for the prevention of ARD, of co-disposing desulphurised tailings obtained from the flotation of ultrafine coal waste, was investigated. Experiments were designed to enhance the exclusion of increasingly aggressive oxidants to acid generating waste rock surfaces by constructing benign permeable reactive barriers in both covered and blended configurations. Tests were performed under continuous flow conditions to assess neutralising efficiency over prolonged periods of time. The study was further extended to determine whether the ore bed surface area has an effect on the mitigating procedure by performing tests in columns with varying aspect ratios. The suitability of the proposed method is assessed by analysing effluent solution concentrations over the extended timeframe.

5.2.1 Co-disposal of benign tailings: materials and methods

5.2.1.1 Samples

Interburden coal, with an average pore volume of 0.035 cm³/g (BET analysis), was obtained from the Waterberg Coalfield in South Africa. X-Ray Diffraction (XRD) analysis showed that the interburden was mostly composed of gangue materials quartz (41.0 wt%) and kaolinite (38.4 wt%) with the only sulphide mineral present being pyrite (1.7 wt%). The most abundant carbonate materials present in the interburden sample were 4.2 wt% calcite, 0.4 wt% siderite and 7.9 wt% dolomite (Figure 4-1a).

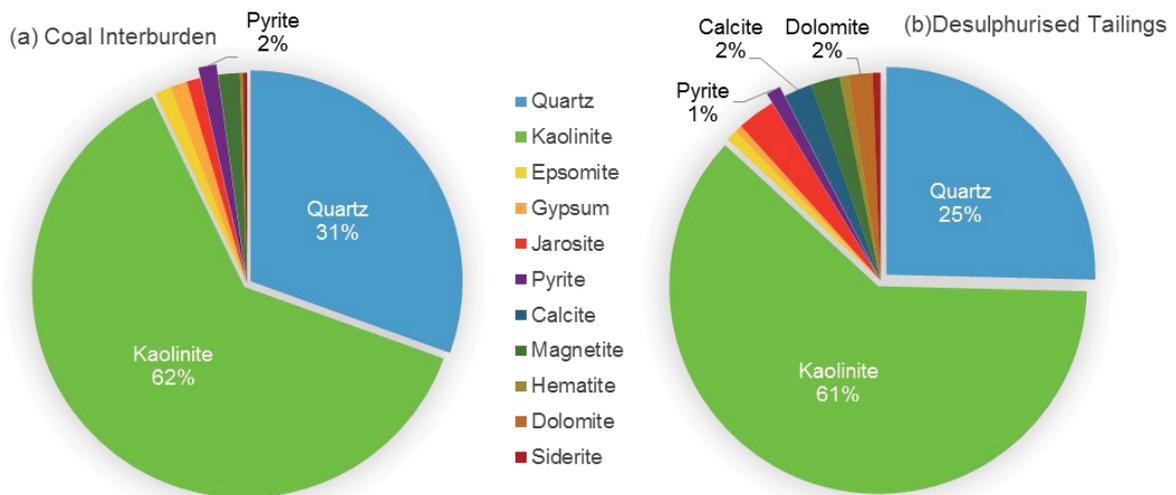


Figure 5-1 XRD analysis of (a) interburden coal obtained from the Waterberg Coalfield in South Africa and (b) desulphurised tailings generated from waste coal from the Middelburg region in South Africa (Kotsiopoulos & Harrison, 2018)

Ultrafine bituminous coal waste from the Middelburg region in South Africa was milled to reach approximately 75% passing 150 μm and floated, via a two-stage flotation procedure, to obtain benign desulphurised tailings with total sulphur content of less than 0.78%. The tailings containing 61% kaolinite, 25% quartz and 1% pyrite (Figure 4-1b), was generated according to the sulphide flotation procedure as described by Kazadi Mbamba, et al. (2013) to obtain a fine particle size distribution with 90% of the particles passing 104 μm and 50% passing 11.4 μm . At the second stage, this entailed floating the ultrafine coal waste at room temperature in a Leeds batch flotation cell agitated at 1200 rpm using potassium amyl xanthate (PAX, Senmin®) as collector for the sulphide minerals, Methyl iso-butyl carbinol (MIBC, Sigma-Aldrich) as the frothing agent and yellow dextrin (Africa Products (Pty) Ltd.) as coal depressant.

The acid generating and neutralising capacities of the Waterberg interburden and Middelburg desulphurised tailings were evaluated using the acid neutralising capacity (ANC), net acid generation (NAG) and net acid producing potential (NAPP) static tests applying procedures outlined by Skousen, et al. (1977) and refined by Stewart, et al. (2006) (NAG, NAPP). With an ANC and NAPP of 20.46 $\text{kg}_{\text{H}_2\text{SO}_4}/\text{t}$ and 27.99 $\text{kg}_{\text{H}_2\text{SO}_4}/\text{t}$, respectively, the interburden was characterized as potentially acid forming while the desulphurised tailings were found to be non-acid forming with an ANC of 57.72 $\text{kg}_{\text{H}_2\text{SO}_4}/\text{t}$ exceeding the maximum potential acidity (MPA 23.89 $\text{kg}_{\text{H}_2\text{SO}_4}/\text{t}$) to give a NAPP of -33.82 $\text{kg}_{\text{H}_2\text{SO}_4}/\text{t}$ (Figure 5-2).

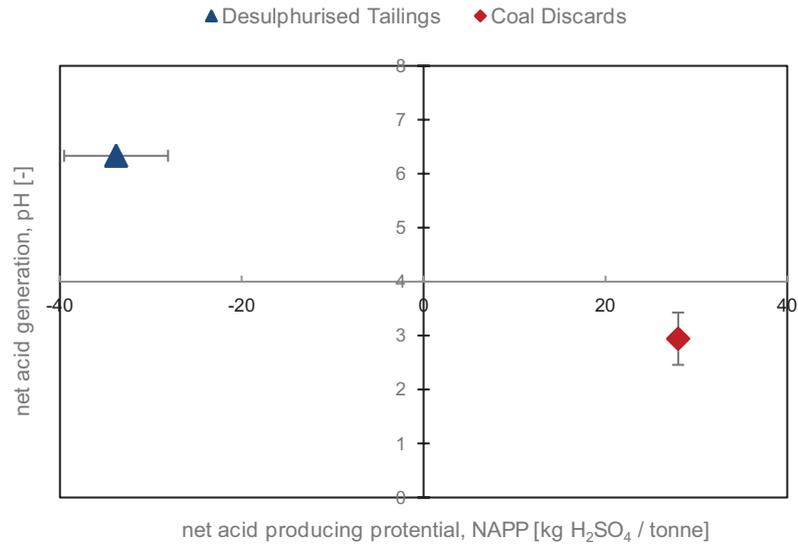


Figure 5-2 Acid generating and neutralising capacity of Waterberg interburden and Witbank desulphurised tailings (Kotsiopoulos & Harrison, 2017).

With these properties, the desulphurised tailings were deemed benign and therefore suitable as a neutralising medium in the potential prevention of ARD. Due to its fine particle size, the benign tailings could concurrently act as a permeable reactive barrier inhibiting the ingress of oxidants to the coal waste rock surface, further inhibiting unfavourable acid generating reactions.

5.2.1.2 Test Columns

Column experiments were performed in the presence of three increasingly aggressive oxidative conditions to establish the neutralising efficiency of the desulphurised tailings on the acid generating interburden. As such, these were demonstrated by the concurrent operation of simulated ore beds in either biotic or abiotic mode irrigated from the top at 30 ml/h with either water or acidified water adjusted to pH 2 with 96-98% concentrated sulphuric acid (Merck). In biotic operation, columns were inoculated with 10¹⁰ cells per tonne of ore of a mixed culture of iron and sulphur oxidising mesophilic microorganisms. Three sets of three vertical PVC columns (H x D = 0.36 x 0.1 m), with an aspect ratio H/D = 3.6, operated at ambient temperature and pressure were used in the initial ARD prevention tests (Figure 5-3).

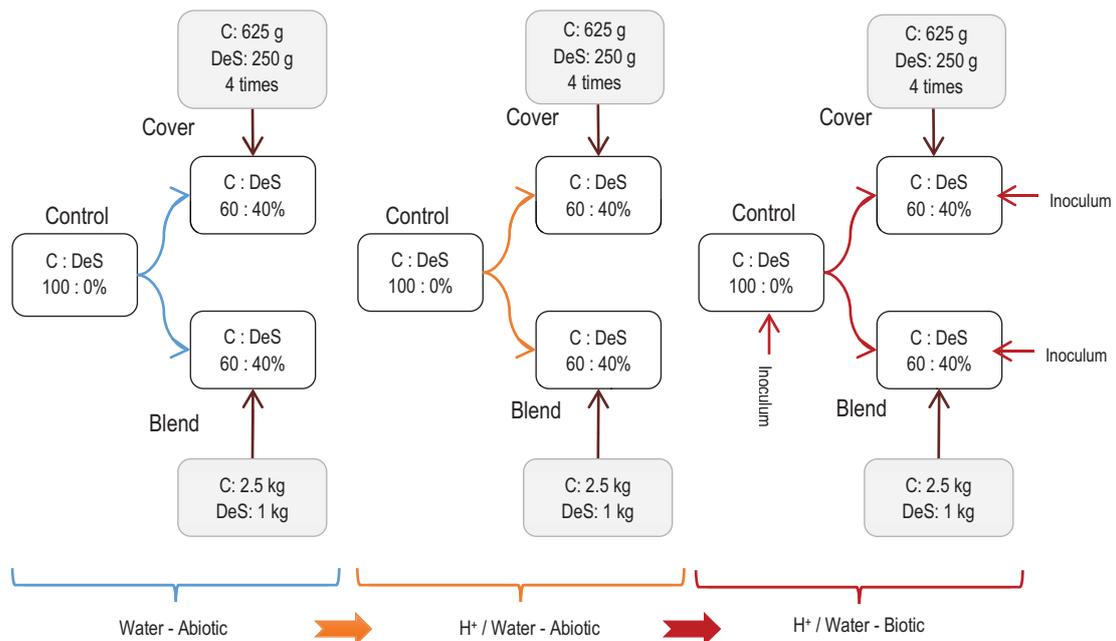


Figure 5-3 Column configurations used for ARD prevention tests. Columns containing interburden (C) and desulphurised tailings (DeS) are irrigated with increasingly aggressive oxidants water and acidified water at pH 2 operated in either abiotic or biotic mode.

The more effective of two co-disposal methods for minimisation of the infiltration of oxidants through the ore bed and mineral voids was determined by varying the packing configurations of the minerals. The PVC columns were packed with a total of 3.5 kg (100%) of either interburden waste rock (control columns) or a combination of 60% interburden and 40% benign desulphurised tailings of the total column mass for in situ neutralisation tests. Four alternating layers of 625 g of coal waste rock and 250 g of benign tailings were loaded into columns investigating the effectiveness of covers in ARD prevention tests while 2.5 kg of interburden was blended with 1 kg of the desulphurised tailings in permeable reactive barrier studies to further minimise pervious channels present in the simulated ore beds.

Aqueous permeate from each of the columns was collected at the bottom of each configuration in 5 l containers and analysed for pH, redox potential (Ag/AgCl, SHE), ferrous ion concentration and total iron concentration. Ferrous ion and total iron concentrations, through the complete conversion of all iron in solution to the ferrous form in the presence of hydroxylamine, were determined spectrophotometrically using the colorimetric 1-10 phenanthroline assay (Komadel & Stucki, 1988). The irrigation medium which did not permeate the column bed was allowed to overflow.

5.2.2 Co-disposal of benign tailings: experimental results

Analyses for column sets irrigated with water and acidified water in abiotic and biotic modes (Figure 5-4) were compared over a period of 200 days. The evolution of pH and redox potential, from initiation of irrigation, was used as a measure of the extent of the oxidation reactions, and indicative of the success of the benign tailings at inhibiting ARD under increasingly aggressive leaching conditions. In all instances, the permeating liquor volume was measured to establish the mean residence time of the aqueous phase. Given the high flow rates of the feed, it was expected that preferential flow paths would initially be established culminating in accelerated flows as these paths altered due to reaction and ore erosion. As such, effluent flow rates were holistically evaluated to determine the efficiency of the packing configuration at reducing the infiltration of oxidants to the acid inducing ore surfaces.

5.2.2.1 Abiotic Columns Irrigated with Water

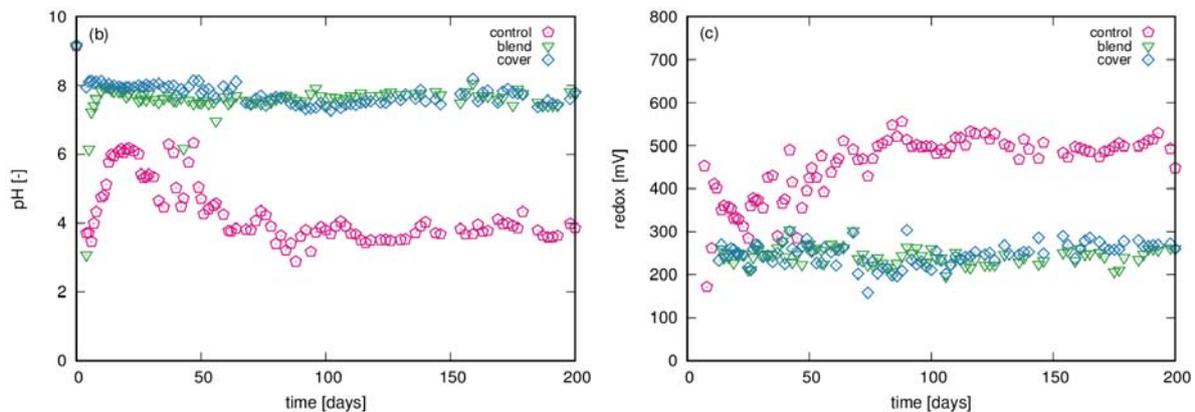
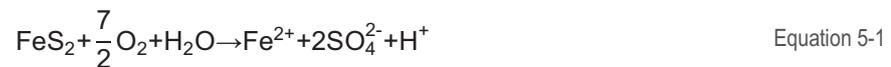


Figure 5-4 Dynamic change in effluent pH and redox potential for columns irrigated with water at 30 ml/h (Kotsiopoulos & Harrison, 2017).

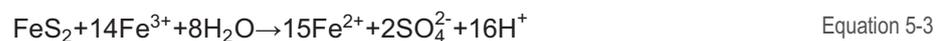
For columns irrigated with municipal water (pH 9.1), the pH decreased significantly to ca. pH 3.0 for both the control and blend columns, while the initial drop in pH for the cover columns was negligible (Figure 5-4a). As pyrite is the only sulphide mineral present in both the coal waste rock and desulphurised tailings, the initial decrease in alkalinity was attributed to the rapid oxidation of fine pyritic particles with both air and water present in the system (Equation 5-1). This was further supported by the initial decrease of the redox potential over the same period due to the liberation of ferrous ions proportionally with hydrons H^+ (Figure 5-4b).



The near instantaneous discharge of Fe^{2+} and H^+ catalysed acid-consuming ferrous to ferric ion turnover reactions increasing the pH and decreased the solution potential over 20 days (Equation 5-2, Figure 5-4). These reactions were most notable in the control columns, with pH and redox recovering almost immediately after the initial acid generating reactions in the blend and cover configurations.



In the absence of any benign tailings, the acid consuming reactions generate sufficient ferric ion oxidant to promote the chemical leaching of sulfidic pyrite minerals present in the simulated ore beds (Equation 5-3). Consequently, a net acid generation, and hence a resulting decrease in pH and increase in redox potential, is noted after 20 days averaging a pH of ca. 3.7 and a redox of ca. 500 mV for the remainder of the experiment (Figure 5-4a).



The net increase in acidity and corresponding increase in redox confirmed that the Waterberg interburden sample was indeed potentially acid producing. However, co-disposed benign tailings obtained from the two-stage flotation process, with coal waste rock, showed promise in neutralising any acid formation in either cover or blend configuration indicating its long-term suitability in ARD prevention applications. The neutralising capacity of these co-disposed columns outperformed the leaching reactions, maintaining the average pH above 7.4 and the redox below 250 mV throughout the experimental period.

5.2.2.2 Abiotic and Biotic Columns Irrigated with Acidified Water

The mitigating efficiency was further tested with increased severity in oxidative conditions. Sets of columns were operated in the absence and presence of microorganisms and irrigated with acidified water at pH 2. For these columns, the potential and rate of ARD generation was determined by measuring the change in system pH and iron turnover (Figure 5-5). Column sets irrigated with water (Figure 5-4) acted as a negative control for the columns charged with acidified water in abiotic mode, which in turn acted as a negative control for the biotic columns.

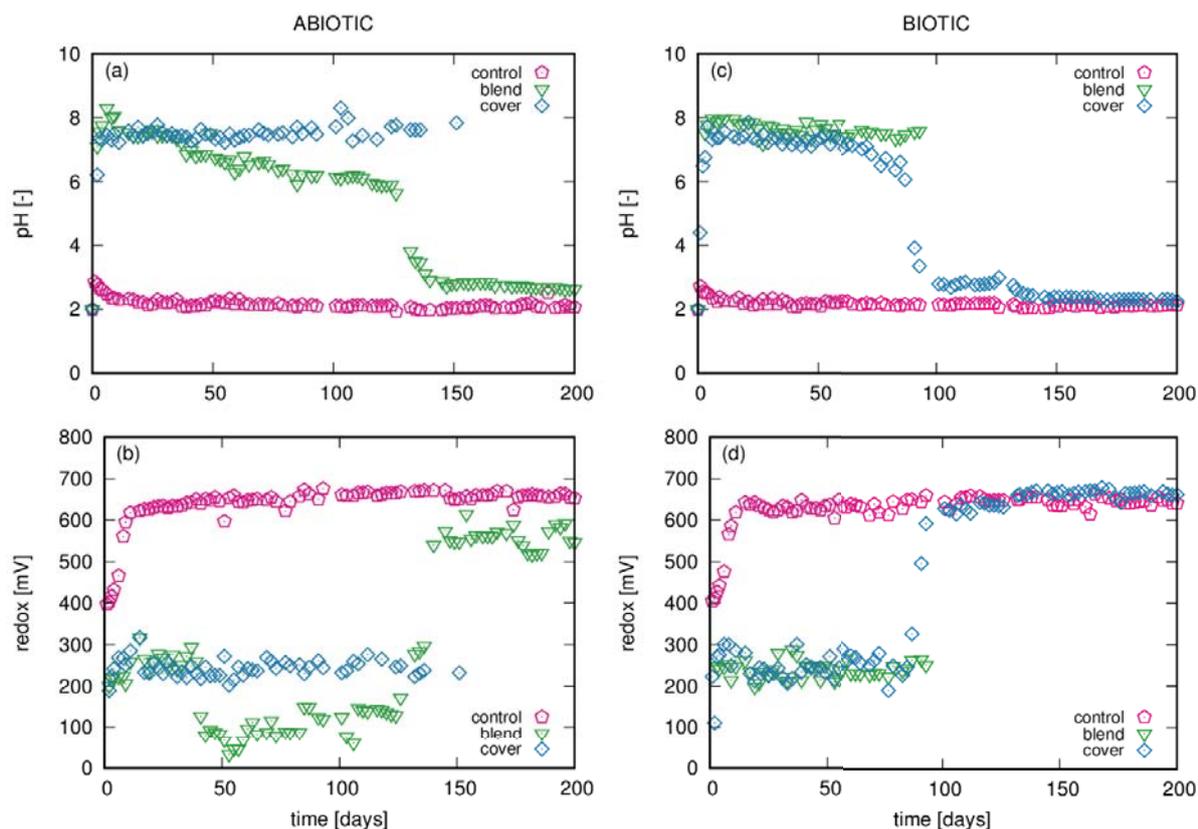


Figure 5-5 Effluent pH and redox potential for columns irrigated with acidified water (pH 2) at 30 mL/h in abiotic (a, b) and biotic (c, d) mode. Biotic columns were inoculated with 10^{10} cells per tonne of ore (Kotsiopoulos & Harrison, 2017).

In comparison with columns irrigated with water (Figure 5-4), the pH of the control columns irrigated with acidified water initially increased slightly (Figure 5-5a and c) and thereafter, over the first 8 days, recovered to stabilize at approximately the same conditions as the inlet. During this period, the influx of acidic medium accelerated the oxidation of any sulphide minerals present, increasing ferric ion production. The redox potential therefore increased rapidly to above 640 mV where it continued for the remainder of the tests in both the abiotic and biotic control columns (Figure 5-5b and d). As the pH remained near the inlet state and did not continue to decrease, balanced acid generating-neutralising reactions were noted. This was confirmed by the deportment of calcium ions into the aqueous phase, where the neutralising carbonates present in calcite sustained the system at the inlet conditions inhibiting further reduction in alkalinity (Figure 5-6).

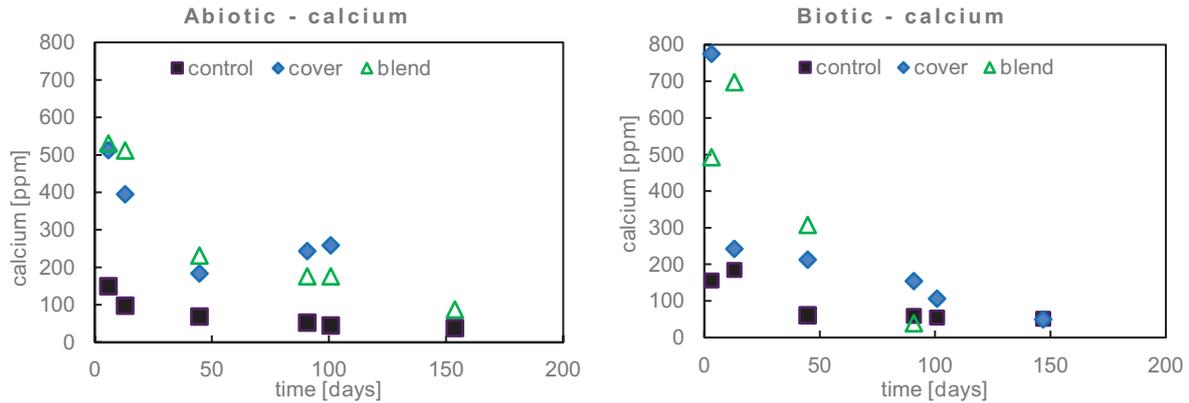


Figure 5-6 Department of Ca^{2+} ions in columns irrigated with acidified water at pH 2 (Kotsiopoulos & Harrison, 2017).

By introducing benign desulphurised tailings to the columns, the alkalinity and redox potential of the effluent remained relatively constant at first at ca. pH 7.5 and 240 mV for the cover and blend columns operating either abiotically or biotically. Both these configurations regardless of the severity of oxidative conditions, i.e. in the presence or absence of microorganisms or inlet pH, continued to show potential in mitigating ARD, similar to what was observed in the water-irrigated columns. By limiting access of ferric ions, protons and oxygen to the mineral surface through layering or mixing benign tailings with the waste rock, oxidation reactions were effectively inhibited. Additionally, the CO_2 required for microbial growth was restricted in these columns suppressing microbially assisted iron turnover reactions (Harrison, et al., 2013). In all columns, packed as either layered cover or blend, operated in abiotic or biotic mode, low effluent flow rates were observed (Figure 5-7). In these columns, excess feed accumulated above the packed beds with the remainder collected via an overflow reservoir. Here, due to their packing efficiency, the columns were saturated, creating an aqueous cover which further enhanced the mitigating efficiency of the desulphurised tailings on ARD generation (Kotsiopoulos & Harrison, 2015). The degree of compactness of the co-disposed minerals thus contributed to limited mass transfer of O_2 and CO_2 and infiltration of the leachate through the ore beds sustaining near neutral conditions in these columns. Microbial activity remained dormant and therefore did not promote accelerated leaching reactions (Kotsiopoulos & Harrison, 2017; Kotsiopoulos & Harrison, 2018).

In the abiotic blend columns (Figure 5-5a and b) and the biotic layered cover columns (Figure 5-5c and d), the neutralising potential of the desulphurised tailings decreased after ca. 40 days and ca. 90 days respectively. Beyond this point, the pH decreased and the redox potential increased, eventually matching that of the control columns.

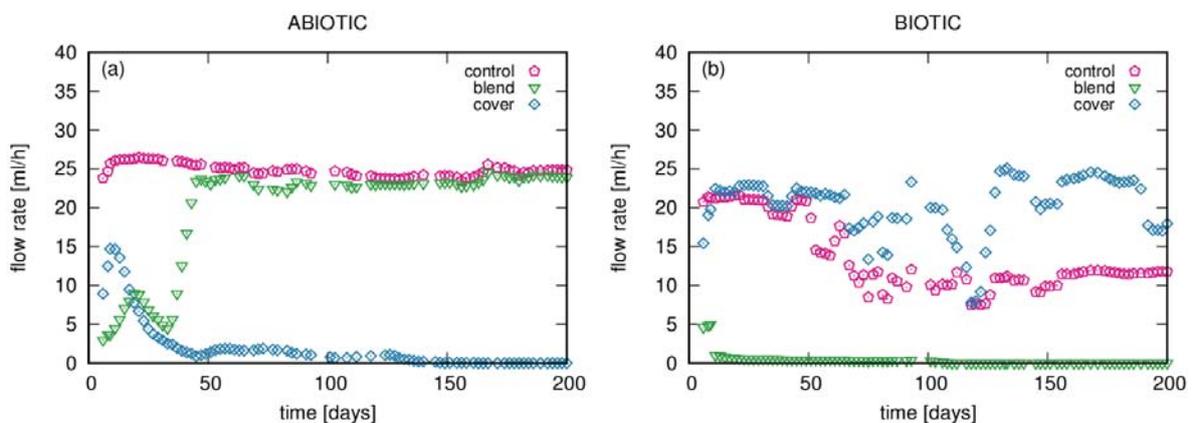


Figure 5-7 Effluent flow rates for columns irrigated with acidified water at pH 2 (Kotsiopoulos & Harrison, 2017).

In the abiotic columns, low effluent flow rates were maintained over 40 days for both packing arrangements. During this period, ARD producing reactions were suppressed by restricting the ingress and development of oxidants through the columns. This is indicated by the favourable outcomes as observed in Figure 5-5a and b. However, in the blend packed columns, the flow rate rapidly increased to ca. 24 ml/h (Figure 5-7a) after which the capacity to neutralise the sulphide tailings decreased (Figure 5-5), signalling the initiation of ARD generating reactions. The potential for the abiotic blend-packed columns to blind the permeable reactive layer of the acid generating waste rock continued to decrease beyond the 40-day period resulting in the pH gradually dropping to ca. 6 at 125 days. The sudden increase in flow rate initiated chemical leaching reactions, postulated as due to a shift in fine desulphurised tailings away from the interburden surface culminating in enlarged flow channels with less restricted flow paths. The increased contact of the ore particles with the flowing acidified water promoted the oxidation of the sulphide minerals resulting in an accumulation of ferrous ions in solution indicated by the gradual decrease in ferric-to-ferrous redox couple, recorded in Figure 5-5b. Consequently, the pH dropped steadily, and the redox potential correspondingly decreased over the same period (Figure 5-5a and b). Beyond the 125 days, the system pH dropped and the redox potential increased. It is clear that ore-associated microbial activity was initiated with the increased production of ferrous ion substrate, which was rapidly converted to the ferric form beyond this point. Microbially assisted leaching reactions accelerated the oxidation of the sulphidic ores thereafter to increase the ARD potential. This resulted in an environment comparable to that observed in the control column.

Prior to the transition from successful neutralisation in the biotic layered cover packed columns to a state of reduced alkalinity, the effluent flow rate remained relatively constant at ca. 23 ml/h (Figure 5-5c and d; and Figure 5-7b). Up to this point the microorganisms introduced into the system remained dormant, showing very little activity in catalysing the oxidation reactions. This is evidenced by the near neutral conditions with low redox potentials seen in Figure 5-5c and d. Preferential flow paths were established early within this column, limiting the dispersion of the leachate throughout the bed surface area. However, with a shift in fine desulphurised tailings over time, these paths altered to eventually recover and build up to accelerated flows after a period of variable flow from 90 to 120 days (as low as 9 ml/h, Figure 5-7b). With the increased exposure of the waste rock mineral surface to the oxidants and increased mass transfer of O₂ and CO₂ within these layers, microbial growth and activity increased resulting in increased leaching reactions with decreased pH and associated increase in redox (Figure 5-5c and d). In the remaining abiotic and biotic columns, this phenomenon was not observed. The mean residence times of introduced liquor remained constant in the abiotic layered cover and the biotic blend columns. These columns became saturated with very little effluent emerging from the column outlets. Thereafter, no further effluent was observed, with the excess feed going to the overflow after 150 days for the abiotic cover and after 100 days for the biotic blend columns. This was attributed to the packing efficiency further impeding flow. No further information was therefore recorded beyond these times. However, based on the recorded data, it appeared that within these columns successful mitigation of ARD generating reactions was achieved, at least, beyond the timeframe noted with their counterparts and could potentially continue beyond the recorded data period. By increasing the solution residence time, an aqueous cover formed evolving into a bifunctional mitigating measure with the desulphurised tailings.

It is clear in the above tests that a relationship exists between fluid flow, and corresponding residence time, with solute transport to the ore surface and through the ore bed and the neutralising efficiency of the desulphurised tailings (Kotsiopoulos & Harrison, 2017; Kotsiopoulos & Harrison, 2018; Kotsiopoulos & Harrison, 2015). Further, the neutralising efficiency of carbonates such as calcite is also dependent on effective contacting of the dissociated anions with the acid generating ore surfaces (Harrison, et al., 2013). As shown in Figure 5-6, even though the concentration of Ca²⁺ (indicative of degree of neutralisation) decreased approaching concentrations comparable with those observed in the control columns, near neutral conditions were still maintained in both layered cover and blend columns operating in abiotic and biotic mode, respectively. Hence, achieving a high degree of compactness of

the co-disposed waste rock with the desulphurised tailings, which effectively restricts the flow and hence contact of the lixiviant to the ore surface, is essential in successful prevention strategies.

5.2.2.3 Influence of Ore Bed Cross-Sectional Surface Area

To determine the influence of flow and ore bed surface area on the neutralising efficiency of the benign tailings in mitigating ARD, the preceding experiments were repeated in wide glass columns with significantly larger column diameters ($H \times D = 0.18 \times 0.16$ m). These wide test columns, with an appreciably lower aspect ratio of $H/D = 1.12$, were irrigated with acidified water at pH 2 at 30 mL/h. In abiotic and biotic mode; all columns were packed with a total of 3.5 kg of either interburden (control column) or a blended mix or alternating layers of a combination of desulphurised tailings (1 kg) with interburden (2.5 kg). Layered covers in these columns were limited to two alternating layers to maintain the total mass of co-disposed benign tailings with waste rock consistent with ARD experiments in narrow columns but sufficiently thick to have an appreciable neutralising potential. As with the narrower PVC columns, biotic columns were inoculated with 10^{10} cells per tonne ore at the start of the experimental runs.

For comparison with the high aspect ratio PVC columns ($H/D = 3.6$), the low aspect ratio glass columns ($H/D = 1.12$) were operated for the same period of 200 days. As before, the evolution of effluent pH and redox potential were recorded as a measure of the extent of ARD generating reactions. It was expected that the same principles of ARD would apply in these columns with the added influence of increased ore bed cross-sectional surface area. As ARD generating indicators have already been elucidated in the previous subsection, this section will focus on the comparative influence of increased surface area on the initiation of oxidative reactions between the narrow ($H/D = 3.6$) and wide columns ($H/D = 1.12$) and the contribution of flow through each of these systems.

Control Columns

In both aspect ratios, the pH and redox potential trends for the control sets overlapped, rapidly reaching pH levels of around 2 and redox potentials above 640 mV (Figure 5-8). Comparatively, the near identical change in alkalinity and solution potential over the same period indicated that the change in ore bed surface area with a decrease in column aspect ratio had little effect under the applied operating conditions. Similarly, the effluent flow rates for both the narrow ($H/D = 3.6$) and the wide ($H/D = 1.12$) columns for the abiotic systems remained relatively constant for the duration of the respective experiments averaging 25 ± 5 mL/h with no notable changes in exit flow rates (Figure 5-9a and c). Although fluctuations in the effluent flow rates were observed in both column aspect ratios operated in biotic mode, with significant decreases noted in the narrow columns after around 50 days, no significant changes were observed in ARD potential (Figure 5-9b and d). This supports the hypothesis that the dissolution of sulphide minerals is accelerated with solution contact with the surface of the acid generating waste rock resulting in conditions conducive to ARD generation (Kotsiopoulos & Harrison, 2018).

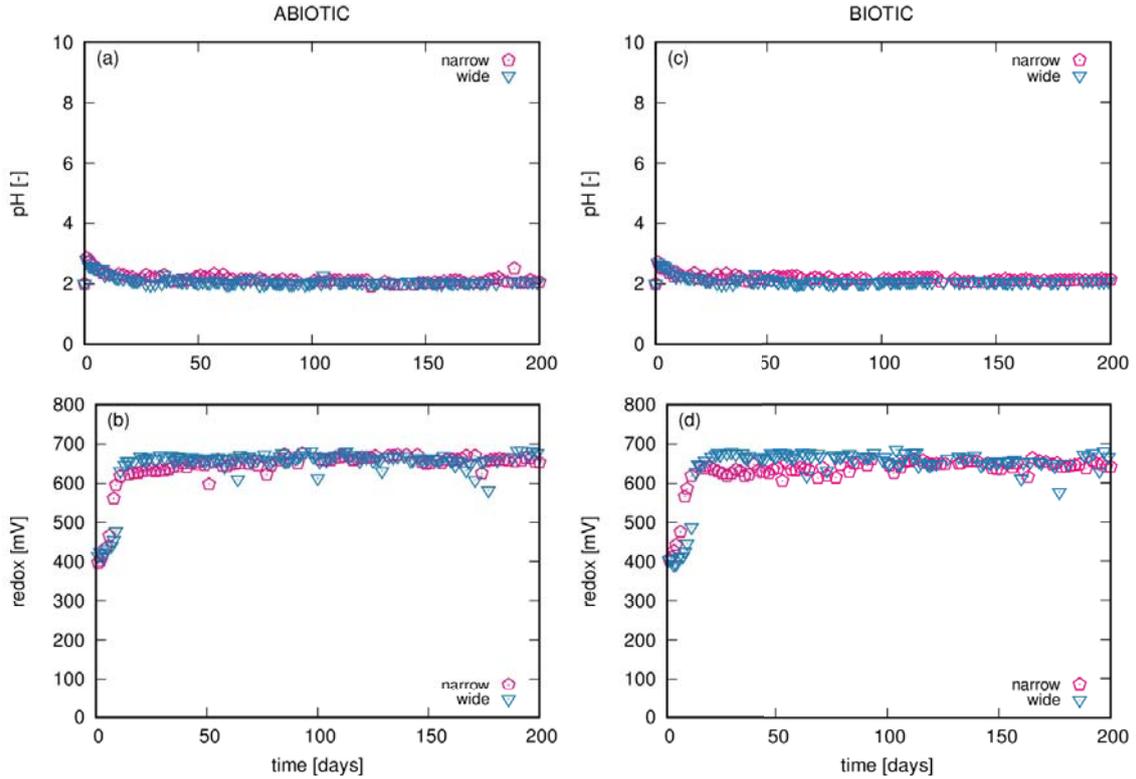


Figure 5-8 A comparison of effluent pH and redox potentials for narrow ($H/D = 3.6$) and wide ($H/D = 1.12$) control columns operating in abiotic (a, b) and biotic (c, d) mode containing 3.5 kg of coal waste rock. Columns are irrigated with acidified water at pH 2 at 30 ml/h (Kotsiopoulos & Harrison, 2018)

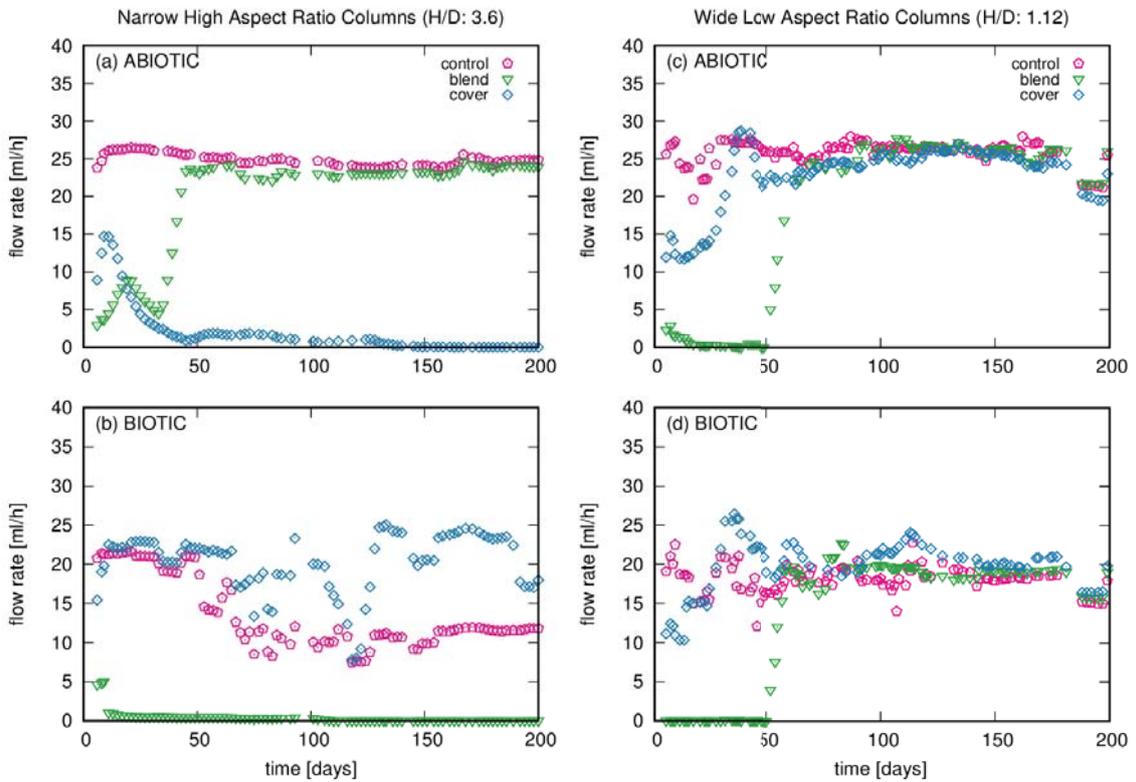


Figure 5-9 Effluent flow rates for narrow ($H/D = 3.6$) and wide columns ($H/D = 1.12$) operating in either abiotic (a, c) or biotic (b, d) mode containing 3.5 kg of coal waste rock. Columns are irrigated at 30 ml/h with acidified water at pH 2 (Kotsiopoulos & Harrison, 2018).

Blend Columns

In the columns packed with blended ore and benign tailings, both abiotic and biotic columns gave results for the pH and redox potential which were comparatively similar for high and low aspect ratios. In the wide columns, very low effluent flows were observed (Figure 5-9). This was due to the immediate collapse in the ore bed with decreased aspect ratio. This was followed by a visible shift in a large fraction of the desulphurised fines, with low hydraulic conductivity, to the bottom of the columns and into voids between the ore bed (Figure 5-10). Consequently, high mean residence times were noted over the first 50 days in these columns.



Figure 5-10 Wide columns with low aspect ratio ($H/D = 1.12$) showing the shift in fine desulphurised tailings through the blend columns (a) before and (b) after initiation of the experimental runs.

In abiotic mode, the pH rapidly increased after which it stabilized and gradually decreased after ca. 40 days (Figure 5-11a). However, the progressive decline in pH and corresponding increase in redox terminated over a shorter period in the wider columns compared with the narrow columns, falling rapidly to pH values comparable with those observed in the control columns from very early in the course of the run. Due to the increased ore bed surface area, it was expected that the same degree of compactness achieved in the narrow columns could not be replicated for the same quantities by mass of both benign tailings and interburden. Due to the increased ore bed cross-sectional surface area, the bed voidage increased with an implication of a higher probability of a shift in benign fines, effectively exposing the surface of the waste rock to the aqueous oxidants. As a result, the initiation of the oxidation reactions occurred earlier in the run with the increase in bed surface area, voids and available lateral plane, enhancing mass transfer and particle-solution contacting. This was validated by the gradual decrease in pH in the restricted flow regime followed by the delayed change in alkalinity with the rapid increase in the effluent flow rate after 50 days in the wide columns indicating a sudden shift in flow paths through the ore bed. Further validation came from the washout of fines observed in the effluent stream (Figure 5-9c and Figure 5-10b). Similarly, for biotic columns, the pH and redox potential culminated in conditions that promote ARD in the wide columns (Figure 5-11b and d). Although, the exit flow rates from these columns remained comparatively constant, the increase in column diameter in the wide columns from the narrow columns promoted increased lateral dispersion and hence contact of the leachate with the acid generating interburden. These results were consistent with observations by

Erguler, et al. (2014). These authors similarly noted that with increased column diameter, the oxidation reactions initiated earlier during the course of testing.

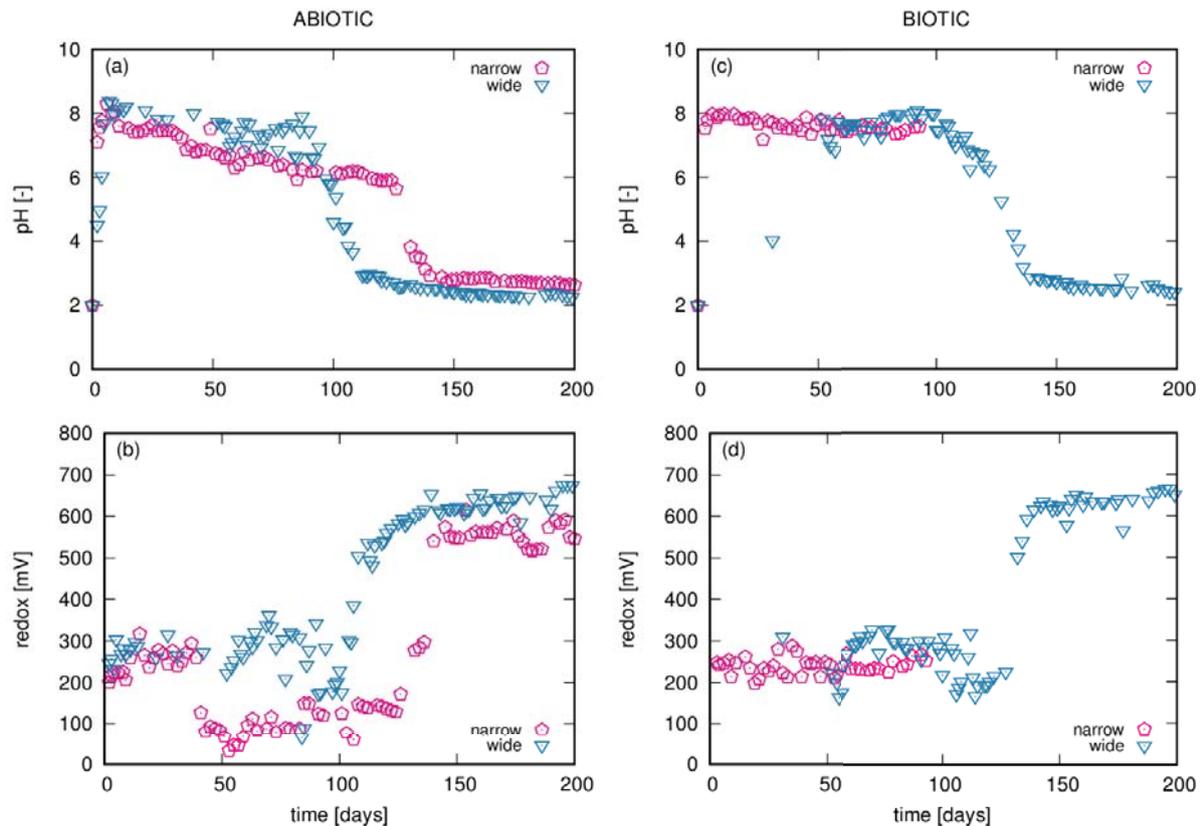


Figure 5-11 A comparison of effluent pH and redox potentials for narrow ($H/D = 3.6$) and wide ($H/D = 1.12$) columns operating in abiotic (a, b) and biotic (c, d) mode with a blend of desulphurised tailings (1 kg) and interburden (2.5 kg). Columns are irrigated with acidified water at pH 2 at 30 ml/h (Kotsiopoulos & Harrison, 2018).

Layered Cover Columns

Like the wide blend columns, the wide layered cover columns resulted in pH and redox potential trends in the wide columns similar to those observed in the narrow columns for about 70 and 50 days for the abiotic and biotic tests, respectively (Figure 5-12). Thereafter, the pH rapidly decreased and the ferric-to-ferrous ion ratio, as indicated by the redox potential, rapidly increased in the low aspect ratio columns. There was an initial variation in flow during the first 40 days, but no major changes in effluent flow rates were noted beyond this point. With variability in flow maintained to within ± 5 ml/h in the wide abiotic columns, the resultant change in solution conditions was mostly due to the increased exposure of the waste rock surface to oxidants with increased column diameter (Figure 5-12a and b). However, while the favourable conditions were not maintained for the same period in the biotically operated wide columns as in the high aspect ratio columns (Figure 5-12c and d), the associated decrease in pH and rapid increase in redox could both be attributed to the variation in flow rates up to 40 days corresponding to the initialisation of ARD generating reactions (Figure 5-9d). Once again, this was due to reduced compactness of the alternating layers of benign tailings and the increased lateral space for aqueous dispersion allowed for multiple points at which oxidants could permeate through subsequent layers through the bed with time. Also, with the increased bed cross-sectional surface area, diffusion of gaseous oxidants through the bed was less inhibited resulting in microbially accelerated leaching reactions.

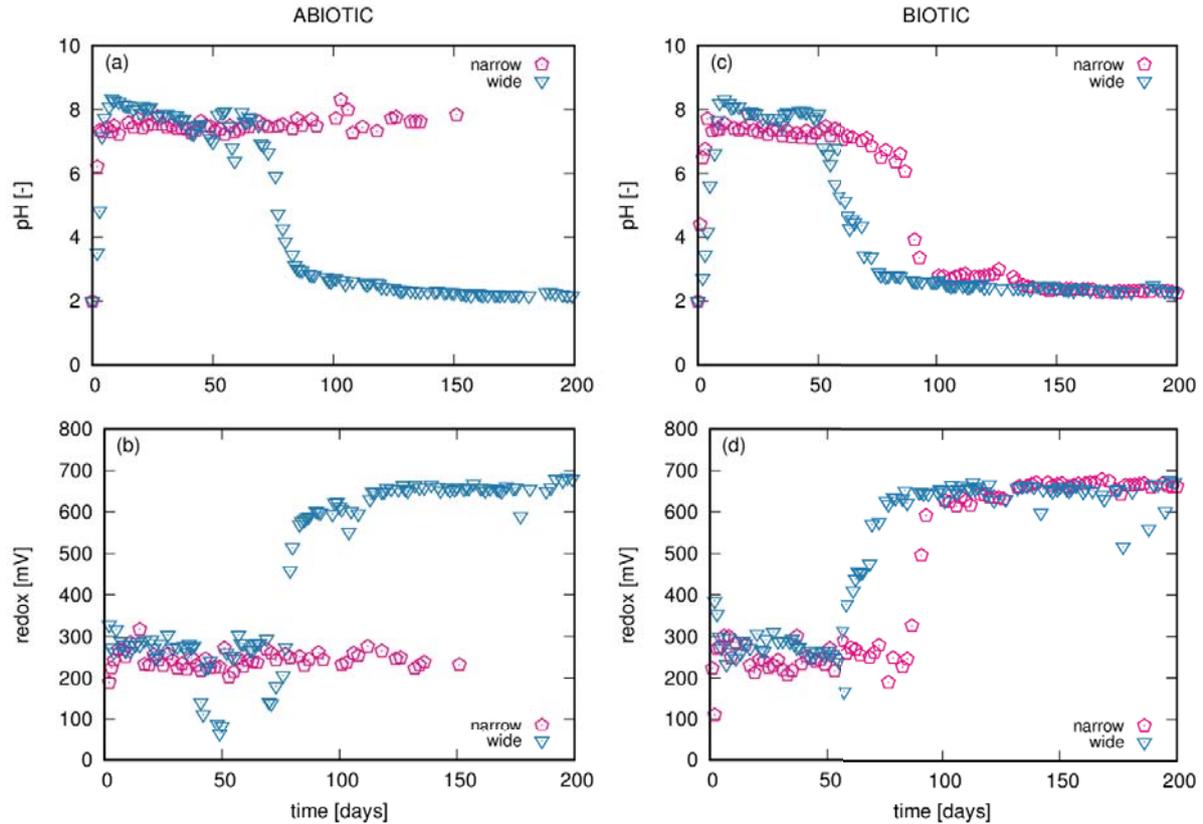


Figure 5-12 A comparison of effluent pH and redox potentials for narrow ($H/D = 3.6$) and wide ($H/D = 1.12$) columns operating in abiotic (a, b) and biotic (c, d) mode with 2 alternating layered covers of desulphurised tailings (0.5 kg) and interburden (1.25 kg). Columns are irrigated with acidified water at pH 2 (Kotsiopoulos & Harrison, 2018).

5.2.3 Conclusions on the use of benign tailings for co-disposal

The applicability of co-disposing benign desulphurised tailings with interburden waste rock as either layered covers or blends was investigated for the minimisation of the infiltration of oxidants to the porous ore and void space within the bed. Aqueous permeate was found to be inhibited by the blend of benign fines with the interburden which reduces cavities in the blended mixture. Successful reduction in ARD generation was observed in all columns irrigated with water.

However, the neutralising efficiency was not sustained in blended abiotic columns or in columns packed as covers when operating in biotic mode. This was attributed to variation in permeate flow paths through the columns. Rapid increases in flow were postulated to be due to a shift in the fine desulphurised tailings in these columns resulting in exposed active sites, hence promoting the oxidation of the exposed sulphide mineral surfaces. In columns where firm packings were sustained, as indicated by inhibited effluent flow rates, ARD potential was minimised maintaining near neutral pH conditions with low associated redox potentials.

Wide glass columns with significantly larger circumference than the high aspect ratio columns were used to establish the influence of increased ore bed cross-sectional surface area on the degree of packing, and accordingly the liquor holdup and the onset of leaching reactions. Observed results indicated that effective in situ neutralisation of waste rock co-disposed with desulphurised benign tailings under extreme oxidising conditions was achieved for shorter periods than those observed in high aspect ratio columns. In comparison with the observed changes in effluent flow rates in the narrow columns, effluent flows in the wide columns increased, with an observed shift or washout of fines. Thus, in both biotic and abiotic modes, the reaction environment appreciably changed over time, evolving to increased acidity and elevated redox potentials indicative of ARD formation.

The terminal neutralising point of abiotic and biotic modes for the low aspect ratio columns ($H/D = 1.12$) preceded that of columns with higher aspect ratios ($H/D = 3.6$) and greater packing efficiencies occurring at about 60 days instead of about 100 days. This indicates that column H/D ratios, and corresponding ore bed surface area, significantly influences the mitigating efficiency of the co-disposed benign tailings. Mass transfer limitations imposed by the compactness of blended columns in both biotic and abiotic modes appear to inhibit microbial activity, enhancing the neutralisation strategy. The correlation between flow, the initiation of leaching reactions and microbial activity was further validated in these low aspect ratio column tests.

In accordance with findings by Harrison, et al. (2013) and Erguler, et al. (2014), long-term results confirmed that, in addition to the neutralising efficiency of the benign tailings, the effectiveness of the preventative measure is largely dependent on the restriction of flow, which is related to the degree of compactness of the ore bed (and maintenance thereof) in the test columns, irrespective of the severity of the oxidising conditions.

5.3 Benign Tailings Used in Fabricated Soils

The large volume of finely divided particulate material comprising the sulphide-lean fraction of the fine coal processing waste is considered as the substratum for manufacture of fabricated soils, with potential for use as top soil in the rehabilitation of mine sites in the first instance. The concept is in the inception stage worldwide with no current research presented on South African materials and limited research presented globally.

5.3.1 Introduction to fabricated soils

Different terminologies have been used for the final product of “manmade” soils. These include: artificial soil (Cox and Whelan, 2000; Haraldsen and Pedersen, 2003; Reynolds, et al., 1999), manufactured soil (Darmody, et al., 2009), anthroposol (Curcio, et al., 2004; IUSS, 2014), technosols (IUSS, 2014), fabricated soil (Kefeli, et al., 2008; Kalevitch, et al., 2008) and minesoils, the last if they are specifically related to the mining activities (Sencindiver and Ammons, 2000). With slight differences, all definitions refer to a product containing an appropriate amount of organic carbon, macro- and micronutrients, porosity and texture capable of supporting adequate plant growth (Darmody, et al., 2009).

Natural soils are porous media created at the land surface through weathering processes mediated by biological, geological, and hydrological phenomena over time. Five factors are considered to be responsible for soil formation: parent material, relief, climate, organisms and time (Sposito, 2008). Given the diversity of such factors along earth’s crust, soils are heterogeneous in three dimensions with important consequences for soil fertility. The optimum nutrient status of a soil alone will not ensure total soil fertility and productivity, since nutrients are only one aspect of soil fertility. Other factors such as absence of contaminants, moisture, temperature, structure, acidity, salinity and microbial population also play an important role in soil fertility (Malherbe, 1948). The fertility of a fabricated soil has to be understood in the same way: chemical, physical and biological aspects have to be addressed together. From an environmental perspective, a fabricated soil should mimic the conditions of surrounding desirable soils. Reproducing micro-indigenous conditions is congruent with the goals of ecological restoration (Falk, et al., 2006). For agricultural areas, fabricated soils have to supply nutrients and water for the chosen crop, rather than for native vegetation.

In constructing fabricated soils, the addition of amendments to a bulk particulate material is required to manipulate the chemical, physical and biological aspects of the material, producing a good soil. Just like natural soils, “manmade” soils depend greatly on their parent material constituents to define their geomorphology, hydrology and the kind of vegetation which can be introduced; these are strictly related to biochemical and physical properties (van Ham, et al., 2007; Tordoff, et al., 2000). This is, of course, applicable to constructing soils using mine waste as the main substratum with clayey material, a

balanced amount of appropriate organic matter, possibly chemical fertiliser and an agent to adjust the pH or buffer the soil forming the usual amendments.

5.3.2 Coal waste as bulk substratum for fabricated soils

This report focuses on modified coal mine waste, in the form of the benign tailings from the two-stage desulphurisation process, as the substrate for the fabricated soil. Studies on the use of coal waste as the main substratum for fabricated soil is limited worldwide to a few instances, notably Firpo et al. (2014) and González Cañibano (1995).

González Cañibano (1995) presented a variety of uses of coal waste, including use as a plant growth substratum. He provided the first report of coal mining wastes as substratum in hydroponic culture and for growing ornamental plants. Experimental plant growth studies resulted in the same crop performance as obtained using traditional substrata. Based on these results, a greenhouse for the production of tomatoes using coal waste as the substratum for hydroponic culture and a facility for the similar production of ornamental plants were constructed. However, the detailed study on experimental test conditions, parameters monitored and performance of these facilities was not reported.

Firpo (2015) proposed a procedure to fabricate topsoil using desulphurised Brazilian coal mine wastes: carbonaceous and pyritic materials were removed from coal discards coming from a jig processing circuit using a dense medium separation method. The low-sulphur material (S_{total} 0.8%) was amended into three different mixtures with: i) steel slag according to ABA analysis to adjust pH at 7.0; ii) sewage sludge to achieve a final content of 3.0% organic matter based on C-org; and iii) 2 wt% rice husk ash to improve physical structure. The graminoid *Avena strigosa* was planted in 2.5 kg soil mixtures and presents a growth compatible to plants grown in natural soils. Soil chemical analysis showed pH, macro- and micronutrients in accordance with plant needs and no contaminants were found in plant tissue.

Key issues for all soils, including those derived from mine wastes are soil acidity and neutralisation reactions, macro and micronutrient availability, organic matter content, microbial community department, heavy metals and phytotoxic compounds mobility, and physical structure. Daniels and Zipper (2010) and Sheoran et al (2010), through a review and a case study, established selection criteria for topsoil substitutes for minesoils including: acidity generation, rock type, extractable nutrients, pH, soluble salts, degree of weathering and oxidation before mining. These parameters are relevant to plant development since high acidity, high concentration of soluble salts, lack of bioavailable nutrients and phytotoxic elements compromise plant growth while contaminants may accumulate in animal or human tissue (Darmody et al., 2009).

Long term studies and investigation of a diversity of plant species still need to be performed. Comminution and amendment addition to coal waste must be analysed case by case, South African sulphide-lean tailings, for example, may not require comminution or an alkalinity source.

5.3.3 Altering the properties of fabricated soils

5.3.3.1 Soil acidity and neutralisation reactions

Pyritic materials control acidification (acid potential) and associated metal mobilisation in all fabricated soils made with coal waste or tailings. Neutralisation reactions will be achieved through lime addition or another acid-neutralising mineral amendment. The decision regarding which neutralising agent should be used considers: solubility rate relative to soil acidification, pH, neutralising potential, particle size distribution, Ca:Mg ratio, cost and purity (Douglas, et al., 2012; Haibin & Zhenling, 2010; Munn, 2005; Ziemkiewicz & Skousen, 1998). The use of desulphurised fine coal waste as proposed in this study avoids or reduces the necessity for alkaline amendment addition.

5.3.3.2 Macro and micronutrient availability

In natural and fabricated soils, macro and micronutrients are required for plant growth. Plant nutrients used in large amounts are classified as macronutrients (H, C, O, N, S, Ca, Mg, P, K) and when used in

smaller amounts as micronutrients (B, Cl, Mn, Fe, Cu, Zn, Mo) (Malherbe, 1948). Hydrogen, carbon and oxygen are provided by photosynthesis; soils must provide all the other nutrients. Three nutrient aspects are important: concentration, availability, and stability/leaching. If any aspect of any nutrients is inadequate, soil conditions can be adjusted or nutrients can be added through fertilisation.

Among macronutrients, nitrogen-based compounds can be added via chemical fertiliser or by planting legume species which are able to fix atmospheric N in the soil. The use of organic nutrient-rich material, like sewage sludge is also a possibility (Belyaeva & Haynes, 2012; Corrêa, 2004). Nitrogen needs to be quickly incorporated into plant or microbial biomass and soil organic matter to avoid losses by leaching (Daniels, 1996).

Like nitrogen, phosphorus availability is strongly related to soil organic matter and mineralogy. If not “stored” in soil organic matter or biomass, phosphorus can be lost by leaching if negative charges predominate or by adsorption if positive charges predominate depending on soil mineral composition. For instance, iron and aluminium oxides under acidic conditions or calcium under alkaline conditions strongly adsorb phosphate ions making phosphorus unavailable to plants (Huang, et al., 2015; Daynes, et al., 2013; Sposito, 2008).

Calcium, magnesium, and potassium availability to plants is strongly dependant on soil pH and cation exchange capacity (CEC) even though the exact mechanism with respect to each element differs. Generally, macronutrients are more bioavailable from circumneutral to higher pH values and in soils with higher CEC which prevents them from being leached. Micronutrients (manganese, iron, copper, zinc, and boron) are generally more bioavailable from circumneutral to lower pH (Kabata-Pendias, 2011; Marschner, 1995; Sposito, 2008). Macro- and micronutrient concomitant availability in circumneutral pH values is one of the reasons soil pH adjustment is important.

5.3.3.3 *Organic matter content*

Soil organic matter (SOM) influences nutrient concentration, availability, and stability/leaching. It promotes nutrient stability and prevents leaching in two ways: incorporating nutrients into its structure or binding them to functional groups (e.g. proteins, phosphate groups). Addition or development of organic matter in the soil increases CEC through the presence of complex macromolecules such as humic and fulvic acids which maintain soil functionality across a very wide range of pH values (Sposito, 2008). SOM has a point of zero net charge (PZC) around pH 3.0, which means that its surface is predominantly negatively charged whenever soils' pH is higher than that. Also, as plants or soil biota decay, microbial communities mineralise nutrients releasing them back to soil solution, favouring nutrient cycling and microbial biodiversity (Daniels, 1996; Liu & Lal, 2014; Sheoran, et al., 2010; Sposito, 2008; Ussiri & Lal, 2005; van Deventer & Hattingh, 2004).

SOM is also responsible for soil aggregation since its functional groups build “bridges” with metals holding minerals together, thereby improving soil structure. Indirectly, SOM improves soil physical properties by stimulating microbial activity which exudate molecules, secretions, hyphae, etc., promoting soil aggregation (Daynes, et al., 2010; van Deventer, et al., 2008). Further, SOM can store water in its hydrophilic sites, thus increasing soil water holding capacity in the soil profile.

Studies have demonstrated that biosolids and sewage sludge (van Ham, et al., 2007), algae (Pulz & Gross, 2004) and sugarcane debris (Li, et al., 2015) can be used as organic matter source in fabricated soils. Nevertheless, addition of organic rich material is not a complete solution. Li et al. (2014) revegetated tailings amended with woodchips as a source of organic material and significant increases in microbial diversity and biomass were found; however, these positive changes did not modify the tailings sufficiently to meet the requirements of a functional soil.

5.3.3.4 *Microbial communities*

The microbial community can act as source and sink of plant nutrients: nutrients available in soil solution are used by the microbial community (sink) however, as microbial communities die they release

nutrients back to the soil solution (source), as part of nutrient cycles. In early stages when moisture levels are high bacteria play an important role in organic material decomposition and are substituted by fungi over time (Geisseler, et al., 2010; Juwarkar & Jambhulkar, 2008; Sheoran, et al., 2010).

An important example of nutrient cycles being related to microbial community is nitrogen: its incorporation into soil organic matter from atmosphere and subsequent release to soil solution depend on a series of reactions controlled by bacterial activity (Geisseler, et al., 2010). Another example is mycorrhizal fungi communities and their relation to phosphorus bioavailability. Such fungi are responsible for solubilizing inorganic phosphate and transporting soluble forms directly into the roots of plants; their biomass also helps to improve soil aggregation via charge, adhesive and enmeshment mechanisms.

To start cycling nutrients and accumulating soil organic matter, fabricated soils should be populated with diverse and healthy biological communities as soon as possible (Huang, et al., 2015; Kumar, et al., 2010; Waterhouse, et al., 2014). Unfortunately, microbial community restoration may be slow, uncertain and often costly (Nortcliff, 2002).

5.3.3.5 Heavy metals and phytotoxic compounds

According to (Sposito, 2008), a toxic element is bioavailable if it is in a readily absorbed chemical form that can affect plant's life cycle. With respect to fabricated soils, the possible presence of highly concentrated soluble salts and contaminants among its constituents must be addressed during characterisation since a toxic environment for plant roots can reduce plant growth and associated SOM accumulation (Neel et al., 2003; Daniels, 1996). Moreover, fabricated soils should not introduce contaminants into the food chain.

Considering SOM's CEC and its ability to bind and immobilise heavy metals, organic amendments have been widely used to control phytotoxicity. Juwarkar and Jambhulkar (2008), for example, used a combination of sewage sludge, biofertilisers and mycorrhizal fungi along with suitable plant species to reduce heavy metals in coal mine spoils.

5.3.3.6 Physical structure, including porosity and permeability

Soil texture, or distribution of particle/pore size, regulates water and air supply in soils. Macropores (pores > 0.06 mm) are responsible for air diffusion and micropores (pores < 0.06 mm) hold water under gravitational forces, with micropore volume equivalent to water content in soils at field capacity. Macro- and micro-porosity correspond to total porosity. Soils with a majority of macropores (sandy soils) will store small amounts of water and allow easy air diffusion. On the other hand, soils with a majority of micropores (clayey or compacted soils) will store water easily, but air diffusion will be compromised. Typically, the requirement for plant development is for at least 10% the soil volume to comprise gas-filled pores at field capacity, with at least 10% of the gas being oxygen (Dexter, et al., 1988).

Other physical parameters are bulk density, structure, and mechanical resistance to plant root penetration, among others. As a rule, the best balance between water, air supply and mechanical resistance is considered to be between soil field capacity and wilting point. Critical density depends on soil texture, for clay loamy soils, for example, it corresponds to 1.4 to 1.5 g·cm⁻³ while for loamy soils it varies from 1.7 to 1.8 g·cm⁻³.

Micropores are related to the space between soil particles, so that the total space available is difficult to change. Macropores, however, exist between aggregates, thus anything that changes soil structure can change macropore distribution. Soil organic matter, amendments (organic waste, agricultural lime, slag, etc.), vegetation and best management practices are among factors that change soil structure as well as bulk density, aggregate stability and resistance to erosion (Tordoff, et al., 2000; van Ham & Teshima, 2005).

It is important to recognise that there is no single physical amendment that gives the best result under all management conditions.

Fabricated soils, or more specifically minesoils, lack structure since they are an artificial mixture of different materials, thus, choosing a particle size distribution that will promote a balance between macro and micropores along with organic matter content is critical for its establishment as a functional soil (Daniels, 1996; Ussiri & Lal, 2005). Clayey material or alternative ultrafine material is used to manipulate the particle size distribution of the fabricated soil to establish the appropriate permeability and hydrology. The clay material may be sourced from "borrow pits", requiring both transportation costs and the need to rehabilitate the borrow pits. Many different materials can be combined in order to improve or create a functional soil as long as the aspects previously mentioned are taken into account. In this context, the use of coal mine waste appears as an opportunity to amend and compose soils, especially during environmental remediation activities (Tordoff, et al., 2000; van Ham, et al., 2007; van Ham & Teshima, 2005) which prevents environmental damage elsewhere by the use of "real soils" and offers the benefit of recycling urban and industrial wastes (US EPA, 2007).

5.3.4 Opportunities for use of fabricated soil for mine site rehabilitation

By law, mined land has to be rehabilitated to a stable state, compatible with the surroundings and restored to its pre-mining agricultural land capability through its integration into the landscape by the establishment of a vegetative cover (Bell, et al., 2001). Many sites for disposal of mining waste are poorly rehabilitated (Håkan Tarras-Wahlberg & Nguyen, 2008; Liebenberg-Weyers, 2010; Beukes, et al., 2007), giving rise to environmental pollution and land degradation. Sites rehabilitated according to best practice based on the prevailing knowledge and available characterisation of the site at the time, may not be environmentally benign over the long term (Lottermoser, 2011; O'Kane, 2003) and further understanding is required to enable long-term predictions of status. Rehabilitation of impacted areas involves the planting of vegetation, hence the presence a layer of functional soil is essential to maintain permanent plant growth. This may be provided through transporting a topsoil or fabricating a soil.

Legacy mines, waste piles and long-term mining operations usually do not have enough soil of sufficient quality for restoration purposes (Beukes, et al., 2007; Platt, 2009). This is a particular issue in South Africa, where historical practices have left a legacy of abandoned and unrestored waste disposal sites. For instance, according the Department of Mines and Energy (2001), since the 1985 inventory annual discard production has increased from 43.6 million tonne (1985) to 66.2 million tonne (2001) covering hundreds of km², most of which are located in close proximity to growing residential areas, sensitive agricultural areas or water bodies. However, through introducing this concept in the earlier phases of the mining project design, the production of major substratum for soil fabrication for rehabilitation can be conducted in situ.

5.3.5 Soil evaluation experiments

The experimental plan was designed according procedure described by Firpo et al. (2014) and consisted of, material selection and preparation, material characterisation, and plant growth experiments. The final results are shown in terms of final size per plant and total biomass after 90 days (Figure 5-13).

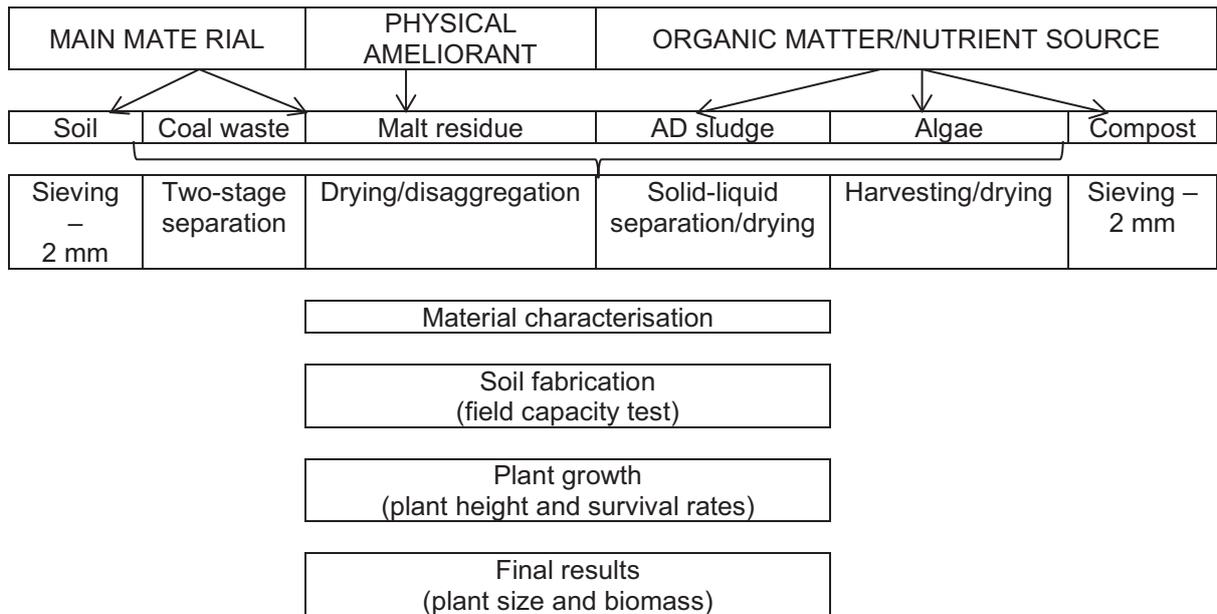


Figure 5-13 Flowchart of the materials and operations conducted in the fabricated soil experimental study

The main substratum was composed of low sulphur “benign” tailings from two-stage desulphurisation of ultrafine coal waste (CW) and soil (S) mixtures in two different ratios: 1:1 and 3:1. Coal waste and native topsoil were collected in the Middelburg area. The control used 100% native topsoil.

In order to improve physical soil structure some tests were conducted using malt residue (2% w/w) as a physical ameliorant. The malt residue was collected during a brewing process performed at UCT, just after the process called lautering.

Organic matter and nutrients were supplied to coal-waste/soil mixtures by comparing the addition of three amendments (3% w/w): AD sludge, microalgae and compost. The anaerobic digested sludge used in this work is from a local brewing effluent treatment plant, the microalgae was produced in house and the compost was obtained from local nursery. Since the coal waste available was limited, experiments using algae as organic matter and nutrient source were carried out with a 1:1 ratio of CW:S. Figure 5-14 shows the experimental delineation and the six materials used.

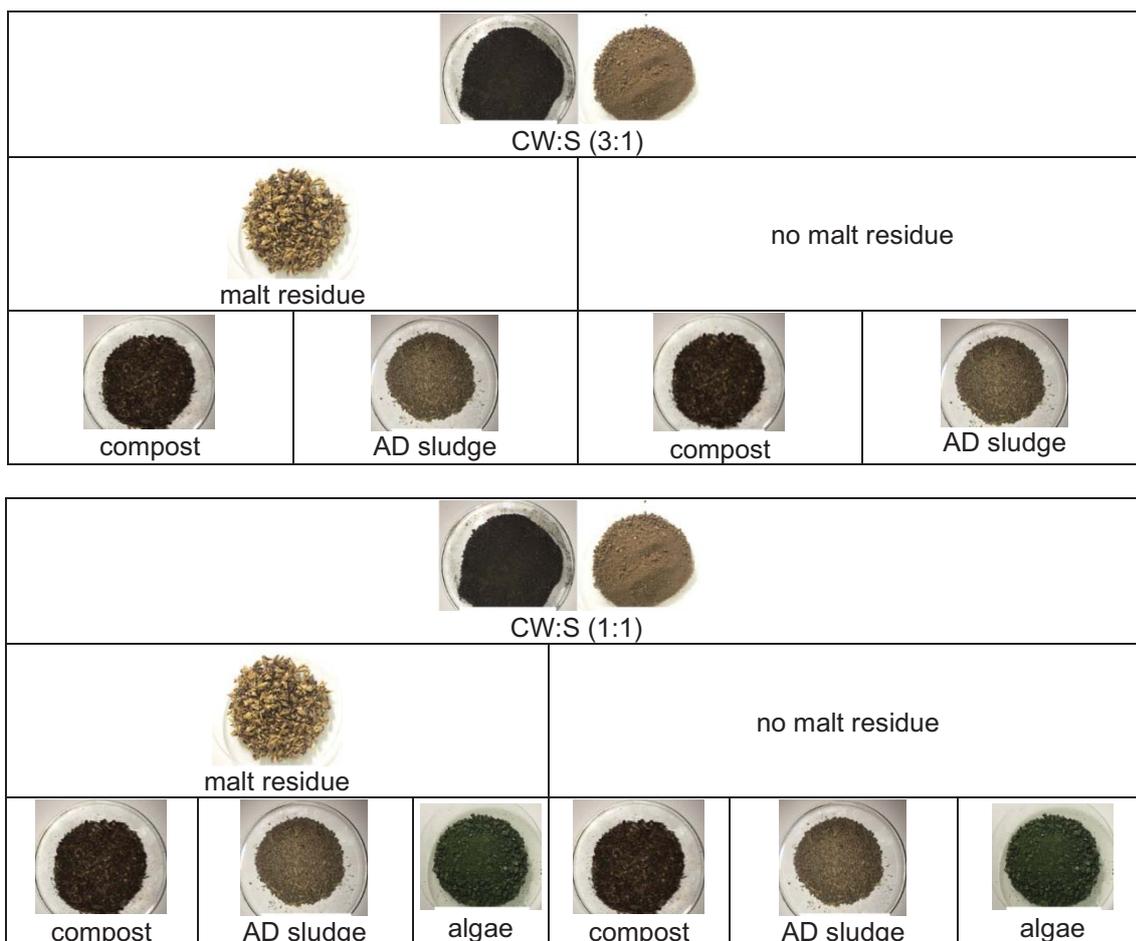


Figure 5-14 Main material, physical ameliorant and organic matter and used in this study

All the materials were characterised according to Table 5-2 before being mixed to fabricate the soils. The field capacity is the amount of water held by the soil after excess water has drained and this was determined for each mixture as part of the characterisation.

Table 5-2 Characterisation of materials used to fabricate soils

	Method	coal waste	organic matter	physical ameliorant	native soil
Particle Size Distribution	Malvern nanozetameter	X	-	-	X
Ash	ASTM D3174 – 12	X	X	X	X
Total Sulphur	ASTM D 2492	X	X	X	X
Total Organic Carbon	Hach 10173	X	X	X	X
Elemental quantitative	EPA 3052	X	X	X	X
Total Phosphate	Hach 8114	X	X	X	X
Total Kjeldahl Nitrogen	Hach 8075	X	X	X	X

Plant growth experiments were conducted, in triplicate, monitoring the germination and growth of *Eragrostis teff* seeds during a 90 days period. *Eragrostis teff* is a typical grass used for rehabilitation in South Africa. Plants were irrigated daily with tap water. Each plant container received 6 seeds which were allowed to grow freely for 31 days (established germination time). The number of plants per

container at 31 days was recorded to give the germination rate. After 31 days, in order to normalise the growth experiments, plants were reduced to two per container before the following 60 days. After 90 days (total), the above ground and below ground plant tissue was cut, dried over 24h at 60°C in a constant flux oven, and then weighed separately, with the total biomass results presented in grams. The final plant height above ground was measured and the results are presented in centimetres. All the experiments were conducted in triplicate and the average and standard deviation of all results were calculated.

5.3.6 Results of soil characterisation

Material characterisation is presented in Table 5-3. The native topsoil a has high mineral content (93.7% ash) and is considerably depleted in sulphur (0.03% sulphur). Particle size distribution showed that coal waste (D_{90} 400.3 μm) is significantly finer than native topsoil (D_{90} 888.9 μm). In terms of elemental composition, it can be observed that coal waste and soil are both high in iron. Coal waste presented higher amounts of potassium, total organic carbon (TOC) and total Kjeldahl nitrogen (TKN). In contrast, native soil presented higher amounts of calcium and total phosphate. The amount of total organic carbon in the soil and coal waste samples are consistent with the ash results obtained for both fractions. However, the high amount of TOC found in the coal waste is not necessarily available for the plants since it is related to the presence of coal particles. Due to the complexity of the organic chains in coal this organic matter will not be available in early stages of the soil development (Firpo, et al., 2014).

Due to the ultrafine characteristics of the coal waste, malt residue was added with the aim of improving the physical structure of the fabricated soil. However, the characterization results demonstrated that the malt residue has similar or higher TOC, total phosphate and TKN compared to the organic matter and nutrient source material (compost, AD sludge and algae).

The compost utilised in this study has a high ash and correspondingly low TOC. In contrast with algae and AD sludge that presented significant higher amounts of TOC, TKN and total phosphate. Regarding the microalgae, it is possible to see that the metals Zn, Al, Mn, Cr and Fe occur in low amounts. The high TOC content in the algae samples is consistent with the low ash content.

Table 5-3 Results of the characterisation of the materials used to fabricate soils

Parameter	unit	Native Soil	Coal waste	Malt residue	Compost	AD sludge	Micro algae	
Ash	%	93.7	63.4	3.3	67.5	45.8	10.0	
Total Sulphur	%	0.04	0.3	0.4	0.23	1.07	0.55	
Ca	%	0.66	0.43	0.77	3.60	1.00	2.48	
Cr	%	0.03	<0.02	<0.02	<0.02	<0.02	<0.02	
Al	%	2.85	5.13	<0.04	1.20	1.58	<0.04	
Fe	%	2.66	2.59	0.03	0.52	0.54	0.07	
K	%	0.27	0.55	0.17	0.71	0.60	1.12	
Mg	%	0.16	0.14	0.34	0.37	0.34	0.43	
Mn	%	0.03	0.05	<0.02	<0.02	<0.02	<0.02	
Zn	%	<0.02	<0.02	<0.02	<0.02	0.60	0.080	
Total Organic Carbon	ppm	9 407	69 259	330 741	59 630	104 444	411 481	
Total Phosphate	ppm	1 825	750	4 363	1 600	5 113	6 225	
Total Kjeldahl Nitrogen	ppm	1 175	5 750	15 750	5 550	23 625	25 250	
Particle size distribution	D10	μm	59.4	16.9	-	-	-	-
	D50	μm	301.6	140.9	-	-	-	-
	D90	μm	888.9	400.3	-	-	-	-

Figure 5-15 shows the amount of water content in the soil after excess water has drained away and the rate of downward movement has decreased (field capacity). It can be observed that all the mixtures show a significantly higher field capacity (FC) than the soil-only control. Experiments using algae demonstrated to retain more water than the other experiments. Higher superficial area derived from the presence of fine grained materials in coal waste contributed to the increase in field capacity in mixtures. As for the addition of amendments, mixtures with algae showed higher capacity to store water. Mixtures using algae together retain 10% more water than the amount of water held by the soil only control, which is important during drought periods. However further experiments should be conducted to evaluate the amount of water available for the plants in the different soils mixtures.

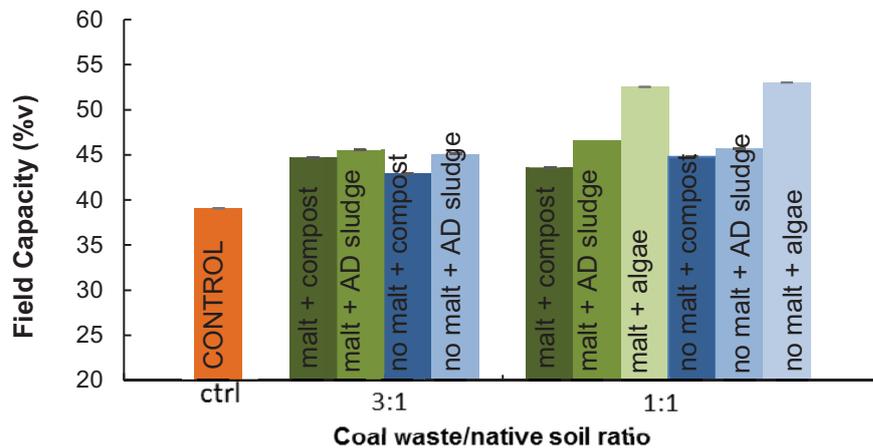


Figure 5-15 Field capacity (%v) for the different amendment combinations; average with standard deviation green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only); 3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil

5.3.7 Results of plant growth experiments

As described in section 5.3.5, initially 6 seeds were sown per container, plants were reduced to 2 in each container at day 31 and then tended to day 90. Figures 4 and 5 present the number of plants per container after 31 and 90 days of experiment respectively.

Only in the native soil did 100% of the plants survive to 31 days. As observed in Figure 5-16, for all the mixtures presented, except the fabricated soils using compost and no malt residue, there were two or more plants per container, in average, after 31 days. This was at least two plants in every container after 31 days. The results for the compost mixtures without malt are consistent with the characterisation results shown in Table 2, suggesting that the poor quality of the compost used in this study did not provide conditions for seed germination in either the 3:1 or 1:1 CW:S mixtures, with only 3 plants and 2 plants, respectively, surviving out of 18 possible plants at 31 days. Results indicate that soils with high concentrations of organic matter and nutrients do not provide enough conditions to the early stage plant development.

On Day 31, in all pots with more than 2 plants, the plants were thinned to 2 per pot. This left pots with the no malt and compost mixtures with 2 pots with 0 plants and 1 pot with 3 plants in the 3:1 CW:S and 1 pot with 0 plants and 2 pots with 1 plant in the 1:1 CW:S going forward to 90 days.

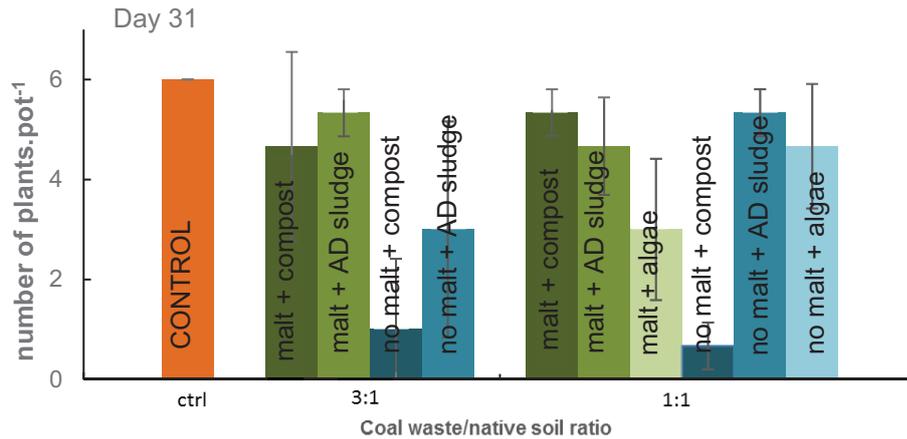


Figure 5-16 Average number of plants per pot at 31 days, average and standard deviation
green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only);
3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil

At the end of the 90-day experiment, the control and four of the amendment combinations still had two plants per pot (Figure 5-17). In the case of the 3:1 CW:S ratio, the only 100% survival rate from day 31 to day 90 was for the compost and malt residue mixture, with the malt and AD sludge mixture performing at 67% plant survival. For both the 3:1 CW:S with no malt and compost or AD sludge began this phase with a total 2 plants and 4 plants respectively, so that the very low survival rates translate to a hidden 50% survival rate for this phase.

In contrast, the 1:1 CW:S ratio had a 100% survival rate not only with malt and compost, but also with no malt and AD sludge, and, in addition, the no malt with algae performed at 100% survival here too. The 1:1 CW:S with compost and no malt retained all the plants present at the start of this phase, giving a hidden 100% survival in growth phase only. The 1:1 CW:S mixture including malt residue and AD sludge had a more than 80% survival rate with only a single plant not surviving; however, the mixture with malt and algae had only 50% of the plants surviving.

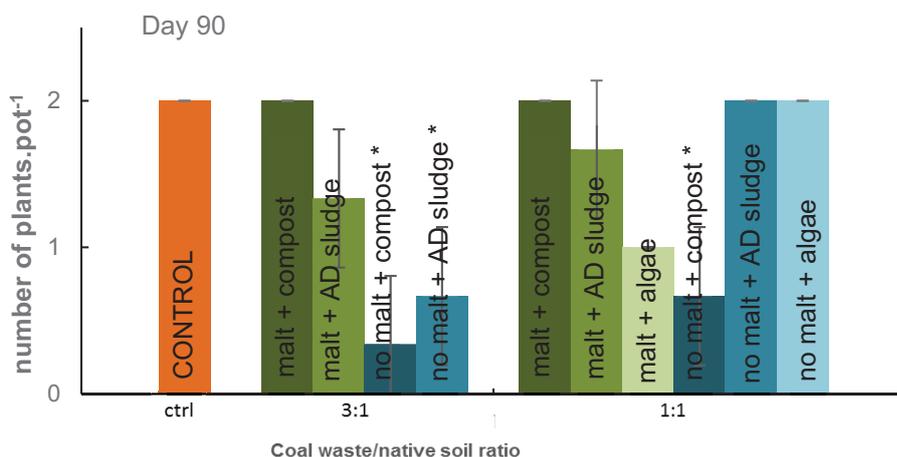


Figure 5-17 Number of plants per pot at 90 days, average and standard deviation
green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only);
3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil
*tests with less than 2 plants per pot at 31 days

In terms of size of plants, it is observed that the control presented a final size of approximately 76 cm which is a higher performance than most of the fabricated soil mixtures (Figure 5-18). As for mixtures with coal waste, the higher were those with a 1:1 ratio, algae and malt in their composition (81.5 cm in

average), but with only one plant per container after 90 days. In general, all mixtures using a 1:1 coal waste/native soil ratio demonstrated a better performance in terms of plant size.

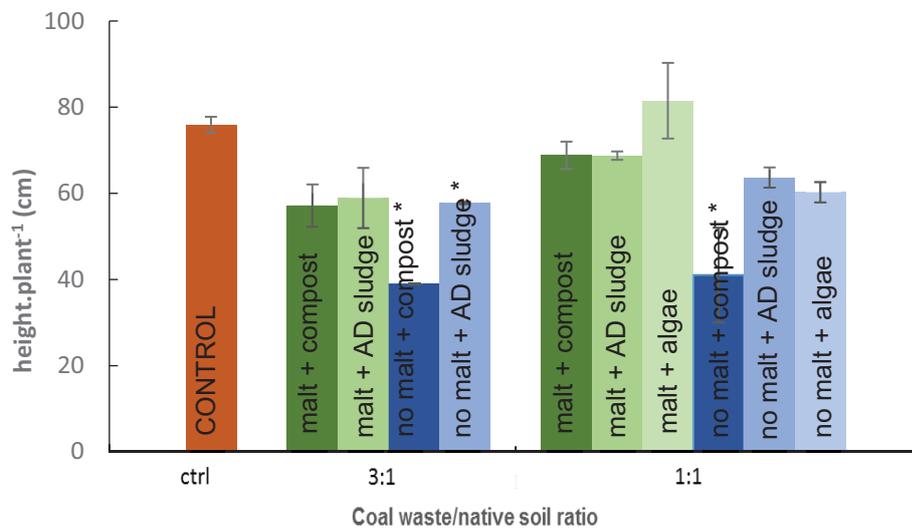


Figure 5-18 Height per plant at 90 days; average and standard deviation
green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only);
3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil
*tests with less than 2 plants per pot at 31 days

In contrast to the plant size, total biomass per plant in fabricated soils, after 90 days, showed better results than the control except for two treatments: no malt with compost for both CW:soil ratios (Figure 5-19). As in the plant size results, once again the fabricated soil made with 1:1 coal waste/native soil ratio, algae and malt residue performed better when compared with the other fabricated soil mixtures. This includes a major improvement over the equivalent mixture with algae and no malt, indicating the possible role of the malt. However, it is important to note that this fabricated soil mixture presented only one plant per pot, after 90 days.

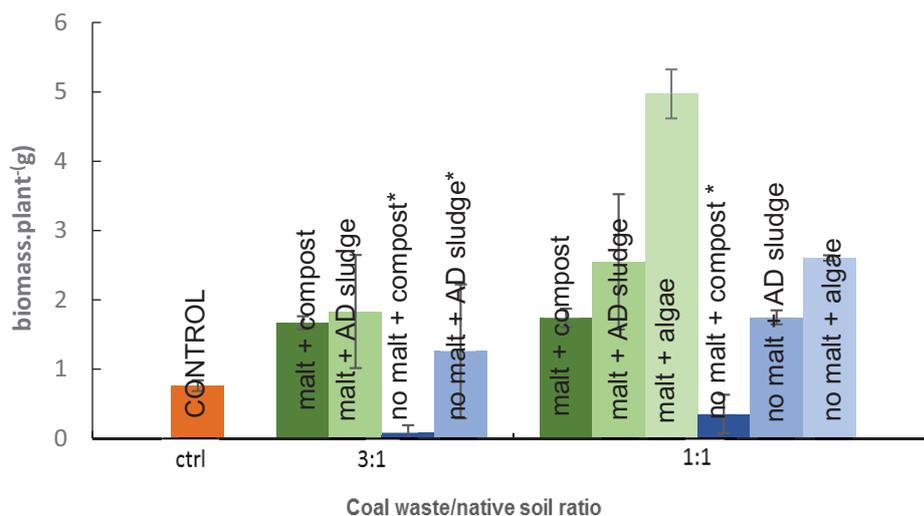


Figure 5-19 Total biomass per plant at 90 days, average and standard deviation
green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only);
3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil
*tests with less than 2 plants per pot at 31 days

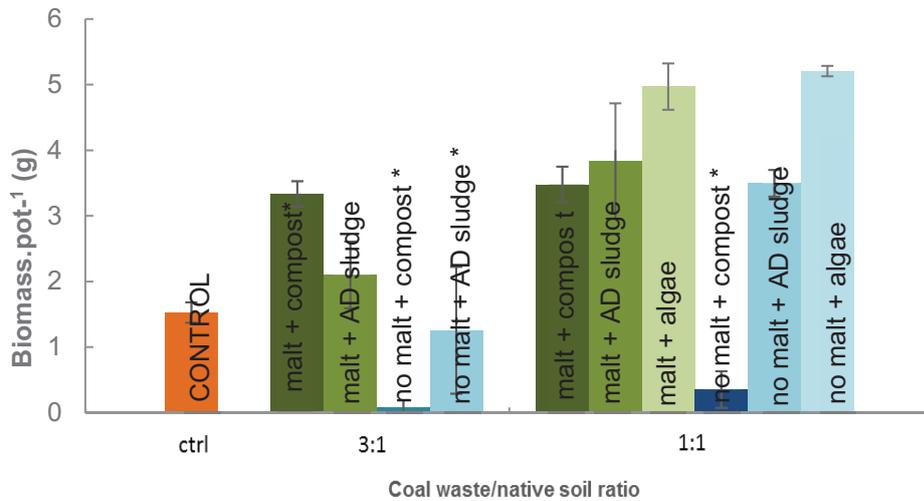


Figure 5-20 Total biomass per pot at 90 days, average and standard deviation
green = physical ameliorant added; blue = no physical ameliorant added; orange = control (native soil only);
3:1 = three parts coal waste and one part native soil; 1:1 = one part coal waste and one part native soil
*tests with less than 2 plants per pot at 31days

Considering total biomass per pot, the results showed in Figure 5-20 indicate that the fabricated soil mixtures using algae as organic/nutrient source have a better performance than the rest of the mixtures. Also, in the tests using algae, the total amount of biomass per pot are the same independent of the number of plants at 90 days, as well as for the AD sludge mixtures. The only mixtures which underperformed in comparison to the control were those using compost with no malt residue and the 3:1 mixture with AD sludge and no malt residue. This means that the malt residue can contribute significantly to final biomass values. It also shows that the compost used in these experiments added virtually nothing to the overall growth. It should be noted however that all three of the underperforming mixtures started at day 31 with fewer plants overall, indicating that the growth issue began during the germination phase.

Table 5-4 shows *Eragrotis teff* plants at 90 days, and also the summarised final results for each different mixture. The growth results using compost mixtures with no malt residue addition, demonstrated that the use of only malt residue as amendment is promising.

Table 5-4 *Eragrotis teff* in soil mixtures with summarised results at 90 days

Coal waste/soil ratio = 3:1		<p>Nutrient source = compost Physical ameliorant = none Field capacity = 95.9 mL Height.plant⁻¹ = 39 cm Biomass.pot⁻¹ = 0.1 g Plants out of 6 = 1</p>		<p>Nutrient source: compost Physical ameliorant: malt residue Field capacity = 100.8 mL Height.plant⁻¹ = 57.2 cm Biomass.pot⁻¹ = 3.3 g Plants out of 6 = 6</p>
		<p>Nutrient source: anaerobic digested sludge Physical ameliorant: none Field capacity = 99.5 mL Height.plant⁻¹ = 57.8 cm Biomass.pot⁻¹ = 1.3 g Plants out of 6 = 3</p>		<p>Nutrient source: anaerobic digested sludge Physical ameliorant: malt residue Field capacity = 105.5 mL Height.plant⁻¹ = 58.9 cm Biomass.pot⁻¹ = 2.1 g Plants out of 6 = 4</p>
Coal waste/soil ratio = 1:1		<p>Nutrient source = compost Physical ameliorant = none Field capacity = 89.4 mL Height.plant⁻¹ = 41.0 cm Biomass.pot⁻¹ = 0.4 g Plants out of 6 = 2</p>		<p>Nutrient source: compost Physical ameliorant: malt residue Field capacity = 90.8 mL Height.plant⁻¹ = 68.8 cm Biomass.pot⁻¹ = 3.5 Plants out of 6 = 6</p>
		<p>Nutrient source: anaerobic digested sludge Physical ameliorant: none Field capacity = 92.6 mL Height.plant⁻¹ = 63.7 cm Biomass.pot⁻¹ = 3.5 g Plants out of 6 = 6</p>		<p>Nutrient source: anaerobic digested sludge Physical ameliorant: malt residue Field capacity = 98.0 mL Height.plant⁻¹ = 68.8 cm Biomass.pot⁻¹ = 3.8 g Plants out of 6 = 5</p>
		<p>Nutrient source: microalgae Physical ameliorant: none Field capacity = 117.2 mL Height.plant⁻¹ = 60.3 cm Biomass.pot⁻¹ = 5.2 g Plants out of 6 = 6</p>		<p>Nutrient source: microalgae Physical ameliorant: malt residue Field capacity = 117.1 mL Height.plant⁻¹ = 81.5 cm Biomass.pot⁻¹ = 5.0 g Plants out of 6 = 3</p>
Control		<p>Nutrient source: micro algae Physical ameliorant: malt residue Field capacity = 68.1 mL Height.plant⁻¹ = 75.9 cm Biomass.pot⁻¹ = 1.5 g Plants out of 6 = 6</p>		

5.3.8 Conclusions on the use of benign tailings in fabricated soils

The mixture of coal waste with native soil and the addition of amendments has been demonstrated as a feasible substitute for native soils for rehabilitation activities using *Eragrotis teff* plants. The use of these soil substitutes could reduce the amount of topsoil used in land restoration by up to 75%.

In terms of the amendments investigated for organic nutrient addition, the low quality compost used in this work did not provide adequate conditions for seed germination or plant development. In contrast, AD sludge and algae both enhanced growth in most scenarios. The addition of the physical ameliorant, malt residue improved growth significantly except in the case where the nutrient additive was algae where the growth was excellent with or without the physical ameliorant.

Fabricated soils using three parts desulphurised coal waste and one part native soil amended with malt residue had better performance than the topsoil from Middleburg area in terms of plant biomass production. Three parts of desulfurised coal waste with one part of native soil and malt residue can facilitate vegetation establishment by supplying nutrients, especially phosphorus, to the soil and vegetation with low environmental risk.

The use of coal waste as fabricated soil represents a potential substitute for natural soils. This use would significantly reduce the land-use footprint and the social impact of mining activities, thus promoting circular economy zero-waste strategy.

5.4 Reviewing Options for Re-purposing of Benign Tailings

There are a number of possible applications for the bulk benign tailings stream from the two-stage desulphurisation process, reviewed in Table 5-1. These include use in construction, either directly or in manufacture of basic construction materials, the former possibly of direct use on the mine-site complex. Further uses are in prevention of acid rock drainage, in stabilisation of mine workings and in the fabrication of soil. Three of the applications are directly related to activities which are part of the normal mining cycle.

It was these three applications which were investigated further in this report, with alternative applications being delayed to the follow-on study in WRC K5/2761. Here, the option of use tailings as aggregate in cemented paste backfill has been demonstrated in Chapter 4; it was concluded that the use of benign tailings in this application would be more advantageous than further work on disposal of the sulphide-rich concentrate for CPB. An experimental study, presented in Section 5.2, demonstrated the efficacy of benign tailings in co-disposal with sulphidic waste rock for the minimisation of ARD. This should be further investigated. The creation of fabricated soil using the benign tailings as bulk material, particularly with use in rehabilitation of mine sites as final application in the first instance, was explored. In the initial experimental study presented in Section 5.3, one of the grasses frequently used in mine site rehabilitation was successfully cultivated in several different fabricated soil mixtures, equalling and surpassing native topsoil in terms of grass biomass over several months' growth. This application should be further investigated for optimisation and rollout at pilot dimensions, including in rehabilitation for natural veld, for pasture and for agricultural use.

Further, extended studies on re-purposing of sulphide-lean fine mine wastes for uses beyond the mine fence, such as in construction materials and polymers, require attention.

6 CONCLUSION

Several key environmental advantages arise from the two-stage flotation process for separation of coal waste into coal, sulphur-rich and sulphur-lean fractions investigated in this project. These are facilitated by the different characteristics of the resultant final sulphur-lean tailings and the composite fine coal waste which forms the mine tailings in the absence of this process. The first key improvement, which forms a long-term environmental imperative, is that the bulk tailings stream is non-acid forming. Removing the ARD generating factor allows for the safe disposal of the sulphur-lean tailings, with the concomitant reduced potential for economic, environmental and social harm which is associated with reduced long-term environmental risk and associated liabilities. The second advantage is that the sulphur-rich stream is at most 25% of the original volume of coal waste, resulting in much reduced long-term disposal costs for containment even if this stream is not valorised. The third advantage is that the total combined fine 'waste' stream for disposal or re-purposing is smaller in quantity, since the coal-rich fraction recovered from the first stage flotation may form 15 to 65% of the original coal waste (Table 3-3). The reduction in space footprint holds economic, environmental and social gains.

In this project, we have demonstrated using LCA and risk assessment tools that, even in the absence of tailings valorisation, the environmental impact of the fine coal waste is reduced by recovering the available coal fraction and by disposing of the high sulphide stream separately from the low sulphide stream. This presents through a reduction in water pollution (salinity, ecotoxicity and metal risks) and a reduction in the consumption of natural capital (land, water and fossil fuels). While climate change indicators and acidification increase owing to electricity consumption for aeration of the flotation cells, this reflects a small overall change as the electricity consumption of the two-stage process accounts for less than 1% of that required for coal recovery.

A techno-economic study showed potential for the two-stage separation route, even in the absence of further stream valorisation beyond coal, i.e. depending on coal revenue only. However, the economic neutrality was impacted by both the coal price (showing profit on first analysis is 2013 when the coal price was high and some cost with the reduced coal price of 2016) and coal grade of the fine coal waste. The potential to improve the economics by integrating this separation into the overall plant train, rather than considering it as a post-processing add-on is noted. Further potential reduction in operating costs through reducing reagent costs will enhance feasibility. In all such analyses, it is essential to evaluate the new options against the cost of the 'as is' approach, particularly the remediation costs associated with the 'as is' approach. Further value must be assigned to the avoidance of ongoing environmental, and potentially social, liability and its associated cost.

This project seeks to investigate the greatest minimisation of waste possible and its associated legacy, i.e. it drives towards a zero-waste approach. As such, it is desirable to consider every stream for re-purposing and possible valorisation, in preference to limiting benefit to achieving minimised disposal impact. To work towards this, we have sought and evaluated re-purposing routes with potential for valorisation and burden reduction for both the high sulphur and low sulphur streams.

To valorise the high sulphur stream, re-purposing its reactive pyrite component formed the focal point. Potential products from pyrite were identified together with the quality of feed pyrite required and their proposed flow sheet. Where low grade pyrite was appropriate as the feed material, the process flowsheet was evaluated using technical, social, environmental and economic criteria and a group of technical experts in the field. Four processes were selected as promising: soil amelioration for correction of acidity and supply of sulphur (and key metals), cemented paste backfill, ferric sulphate coagulants for water treatment and heat generation in heap leaching. The first three were evaluated in detail. While soil amelioration shows significant potential, in South Africa the lack of proximity between mine site and suitable soils implies the transportation costs outstrip the benefit over CaSO_4 . However, the pyritic fraction does have benefits of slow release and hence less frequent application. Cemented paste backfill has promise, but the limited material available as sulphide rich fraction in South Africa

and the unresolved impact on whether sulphate will be formed imply that the use of sulphide lean tailings, or untreated tailings, is preferable to sulphide rich tailings for CPB. While the uses and production potential of ferric sulphate were established, the technoeconomics of this process requires further validation.

In the re-purposing of sulphide-lean tailings we have chosen to evaluate uses on mine site in the first instance. These include CPB (discussed above), soil fabrication and co-disposal with waste rock. Alternative uses in construction and formation of polymeric materials are considered in further studies (WRC K5/2761).

The demonstration of the co-disposal of acid-neutralising sulphur-lean tailings and sulphidic waste rock was demonstrated to reduce ARD formation. The lack of sustainability of the neutralising function is highlighted, indicating the importance of maintaining the integrity of an impermeable ore bed over the long term as bed integrity to minimise permeate flow is essential for performance. While demonstrating proof of concept, the study highlights the need for good tools to characterise the ore bed in terms of achieving its stability, integrity and low permeability. Further scale-up studies are needed owing to dependence of bed stability on geometry.

The third approach to re-purposing sulphur lean tailings is soil fabrication. Here the need to validate the benign status of tailings before use is highlighted. It has been demonstrated that a technosol can be formed with a fine coal content of 75%. By desulphurisation, acidification is prevented. Improved plant growth was observed in the presence of the physical ameliorant malt extract and nutrient sources such as anaerobic digester sludge and algal biomass. While ongoing work is needed to optimise the soil composition and to generate understanding of the change in soil quality and metabolic activity with time, the demonstrated concept shows great potential in both reduced topsoil burrowing and in having greater availability of soil for mine site rehabilitation. Reduction in topsoil burrowing reduces both ecological impact and transportation costs.

In conclusion, our study shows the potential for reduction in environmental, and social, liabilities through appropriate separation of waste streams to reduce the material for disposal and enable appropriate disposal choice. Economically, a sound argument can be made for this approach. Further, re-purposing and potentially valorising the high sulphur and low sulphur tailings streams provides opportunity to enhance resource efficiency, reduce demand on natural capital, reduce land use, reduce ongoing liabilities and improve economic outcome. Potential exists to move towards a zero-waste strategy with significant benefits for mine closure. Ongoing study to achieve this aim is recommended.

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APPENDIX A: CAPITAL COST ESTIMATION GUIDE

Capital cost method	Factored cost estimate	Weiss	Plant component cost ratio	Peters and Timmerhaus
Parameter	Factors			
Delivered equipment costs	-	x	x = 1.03 *(equipment costs)	15-40% of total
Equipment installation	x = 1.43 *(equipment costs)	40% * x	17-25% * x	6-14% of total
Piping process	10-30% * x	20% * x (flotation)	7-25% * x	3-20% of total
Electrical	25-100% * x	-	13-25% * x	2-10% of total
Instrumentation	5-10% * x	5-15% * x	3-12% * x	2-8% of total
Process buildings	20-60% * x	15-40% * x Buildings and site	33-50% * x	3-18% of total (includes services)
Auxiliary building	-	-	7-15% * x	-
Utility lines	5-15% * x	5-25% * x	-	-
Plant utilities	Included under electrical	50-100% * x	7-15% * x	8-20% of total
Field expenses – construction management	-	-	10-12% * x	4-16% of total
Site improvements	-	-	3-18% * x	2-5% of total
Sub-total	y	y	-	-
Project management, engineering and construction	20-35% * y	50% * y	30-33% * x	Engineering: 4-21% of total Contractors fee: 2-6% of total
Contingency	20-30% * y	20% * y	-	5-15% of total
Escalation	-	annual % *(completion yrs)*(sum of items listed above)	-	-
Size factor – small	5-15% * y	-	-	-
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