A Mine Water Management Strategy for the Extension of an Opencast Colliery in Mpumalanga

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DECLARATION

I, Quintin Nel, hereby declare that the dissertation hereby submitted by me to the Centre for Water Sciences and Management in the Faculty of Natural and Agricultural Sciences at the University of the North West, in fulfilment of the degree of Magister Scientiae, is my own independent work. It has not previously been submitted by me to any other institution of higher education. In addition, I declare that all sources cited have been acknowledged by means of a list of references.

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ABSTRACT

Water is a critical resource and can pose a huge risk to mining operations. Therefore, understanding the hydrogeological conditions at mining sites is essential in minimizing the impact on groundwater, and to develop practical and cost-effective management and mitigative solutions. Mining and mining processes are often associated with Acid Mine Drainage (AMD). These impacts are generally only identified and addressed in the post-mining operational phase. The costing associated with post-mining rehabilitation is often not adequate to address the impact of acid-mine drainage.

This study focused on how an opencast colliery called Mine X in Mpumalanga will behave hydraulically and geochemically during mining, and hydrochemically post-mining if potential decant will occur. Additionally, the study presents a methodology that may be used to predict future mine water decant chemistry and the applicable cost of pH pre-treatment as a condition set by the current RO (Reverse Osmosis) plant. To address the focus of the study, numerical flow modelling, numerical transport modelling, geochemical modelling, statistical analysis and analytical modelling was performed. The results of the above showed that calculated inflows expected during mining will be 653 m^3/day after which, rebound of groundwater levels upon cessation of mining will be approximately 11 years. The post-mining decant volume was calculated at 6 l/s with a calculated starting concentration of 1900 mg/l of SO_4. This was determined using non-parametric multivariate statistical analysis of 48 samples between three similar mining sites which are currently decanting. Using principal component analysis as well as clustered analysis an estimated concentration was assigned to the source term in a transport model with an annual decay rate of 5% p/a based on the work of (Mack & Skousen, 2008). Based on the result of the transport model, a relationship between SO_4 and pH was calculated using 1790 samples from the same sites. Geochemical modelling was subsequently performed to determine pre-treatment product volume requirements for the dynamic pH values associated with the dynamic SO_4 concentrations. pH and SO_4 are dependent variables in the opencast pit but are both influenced by the amount of sulphide materials present in the backfilled opencast mine. Other influencing parameters could include carbonate mineral phases. However, due to the absence of alkalinity in the mine water samples, it was assumed that carbonates are not present or depleted. Therefore, pH and SO_4 in decant water remain independent due to the absence of sulphides and the fact that sulphate is not a pH dependant species.

The results of the statistical model indicated a positive relationship between pH and SO_4 for the study area, and identified that possible statistical relationships between other constituents,

are also likely. Therefore metals concentrations can also be calculated using these relationships, and effectively targeted and treated by passive and/or active methods.

From this a mathematical expression was developed to determine a relationship between the required product volume and the required pH change which is flexible enough to accommodate an evolving source term.

This approach can be implemented in various post mining environments taking the listed parameters into account. This is likely to improve the dynamic apportionment of capital and operational expenditure in the management of post mining hydrochemistry.

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1. INTRODUCTION

1.1 Context of study

Water is a critical resource and can pose huge risk to mining operations. Its management and protection are of paramount importance, and therefore understanding the hydrogeological conditions at mining sites is essential to minimizing the impact on groundwater, and to developing practical and cost-effective management and mitigative solutions.

Part of a mine water management strategy is to conduct a hydrogeological impact study and to determine suitable mitigation measures to address future calculated impacts. The proposed extension of the opencast colliery by conventional opencast truck and shovel rollover method is located in the Mpumalanga Coalfields. Mining of such a coal field is often associated with the release of contaminated mine water (Bell, et al., 2001). When water becomes contaminated through the different processes of coal mining, it has the potential of causing Acid Mine Drainage (AMD). AMD occurs when sulphide minerals in rock are oxidised and exposed to moisture, resulting in the generation of sulphate, metals and acidic conditions that have a negative impact on the environment (Vermeulen & Usher, 2005). The Witbank coalfield geology in Mpumalanga generally includes carbonate mineral phases such as calcite and dolomite, which help neutralize the acidity, resulting in mine waters not necessarily having high acidity, but still maintaining elevated sulphate concentrations (Usher, 2003).

The environmental impacts of AMD and other contaminants from coal mining are in many cases only identified and addressed in the post-mining phase. In today's growing necessity to produce energy, coal resources in the Mpumalanga coalfields are progressively becoming depleted, and future mine closures are predicted to increase (Vermeulen & Usher, 2005). With this in mind, mine management strategies must become more efficient in developing suitable mining techniques and develop adequate mine water management measures that will ensure that collieries keep the impact on groundwater to a minimum.

This study focuses on the opencast colliery called Mine X which commenced mining operation in January 2013. Mine X will exploit three coal seems, namely the No.2U and the No. 2L seams, similar to the current opencast colliery. The opencast colliery is currently fully operational with an estimated life of mine of eleven years. The colliery treats contaminated groundwater through the use of a Reverse Osmosis (RO) Plant on site. The RO plant was designed to treat a feed water flow of 15 ℓ/s, and currently runs at a maximum of 8 ℓ/s. However, the high concentration of contaminants in feed waters cause rapid scaling and fouling of membranes and are highly corrosive, leading to higher capital and operational expenditure. Settling trenches were designed for the pre-treatment of pH before feed water is

supplied to the RO plant. The requirement set by the current RO operations is to pre-treat pH to a minimum of pH 5.2.

Mine X wishes to extend their mining operations which will include an additional opencast pit. The predicted expansion of the opencast colliery will extend approximately 80 hectares. This study will present a mine water management strategy that will demonstrate and predicts the influence that the extension of the opencast colliery will have on the groundwater quality over time, and recommend applicable mitigation measures to limit impacts to the water resource. Additionally, the study will present a methodology that may be used to predict future mine water decant chemistry and the applicable cost of pH pre-treatment as a condition set by the current RO plant.

1.2 Problem Statement

Groundwater quantity impacts are expected during the operation of the mine. This is likely to influence water quantity and associated quality of the groundwater system. Additionally, decant has the potential to occur post-mining, resulting in the discharge of contaminated mine water to the environment. This can lead to various negative environmental effects if left untreated. Therefore, passive or active treatment may be required and the quantification of associated costs. As mine water chemistry is a dynamic system, a dynamic quantification for its pre-treatment may be required for the site.

1.3 Aims and Objectives

1.3.1 Quantify the potential water quantity impacts associated with the opencast coal mine.

What are in the inflow volumes into the mine and how does the cone of depression effect the receptors surrounding the mine.

1.3.2 Quantify the potential post-mining water quantity and quality associated with the opencast mine.

What are the predicted mine water discharge volumes and qualities over time?

1.3.3 Develop a dynamic cost estimate for the pre-treatment and/or mitigation of contaminated mine water discharge.

Quantify the pre-treatment costs as a dynamic mathematical expression which is flexible enough to accommodate the evolution of mine water chemistry over time.

2. LITERATURE REVIEW

2.1 Introduction

The following sections aim to provide a logical and theoretical description of opencast coal mining in the Mpumalanga Province in South Africa. The literature review describes the geological setting of the Karoo Super-group, Opencast coal mining in Mpumalanga, mine water chemistry, rehabilitation methods for collieries, different ways of quantifying impacts and associated water quality, and mine water management.

2.2 Geology and Hydrogeology of the Karoo Super-group, with focus on the Vryheid Formation

The Karoo Supergroup is famous for its fossils, thick glacial deposits and extensive flood basalts with their associated dolerite dykes and sills. The main Karoo Basin covers an area of 700 000 km^2 and attains a total cumulative thickness of approximately 12 km (Johnson, et al., 2006). Figure 1 represents the layout of the Karoo Supergroup throughout Southern Africa.

The sedimentary part of the Karoo Supergroup is subdivided into four main lithostratigraphic units, which form the base up are the Dwyka, Ecca, Beaufort and Stormberg groups (Pinetown, et al., 2007). Below follows a brief description of the underlying Stormberg, Beaufort and Dwyka Groups. More detail and focus was put on the underlying Ecca formations, as the study area rests on this group.
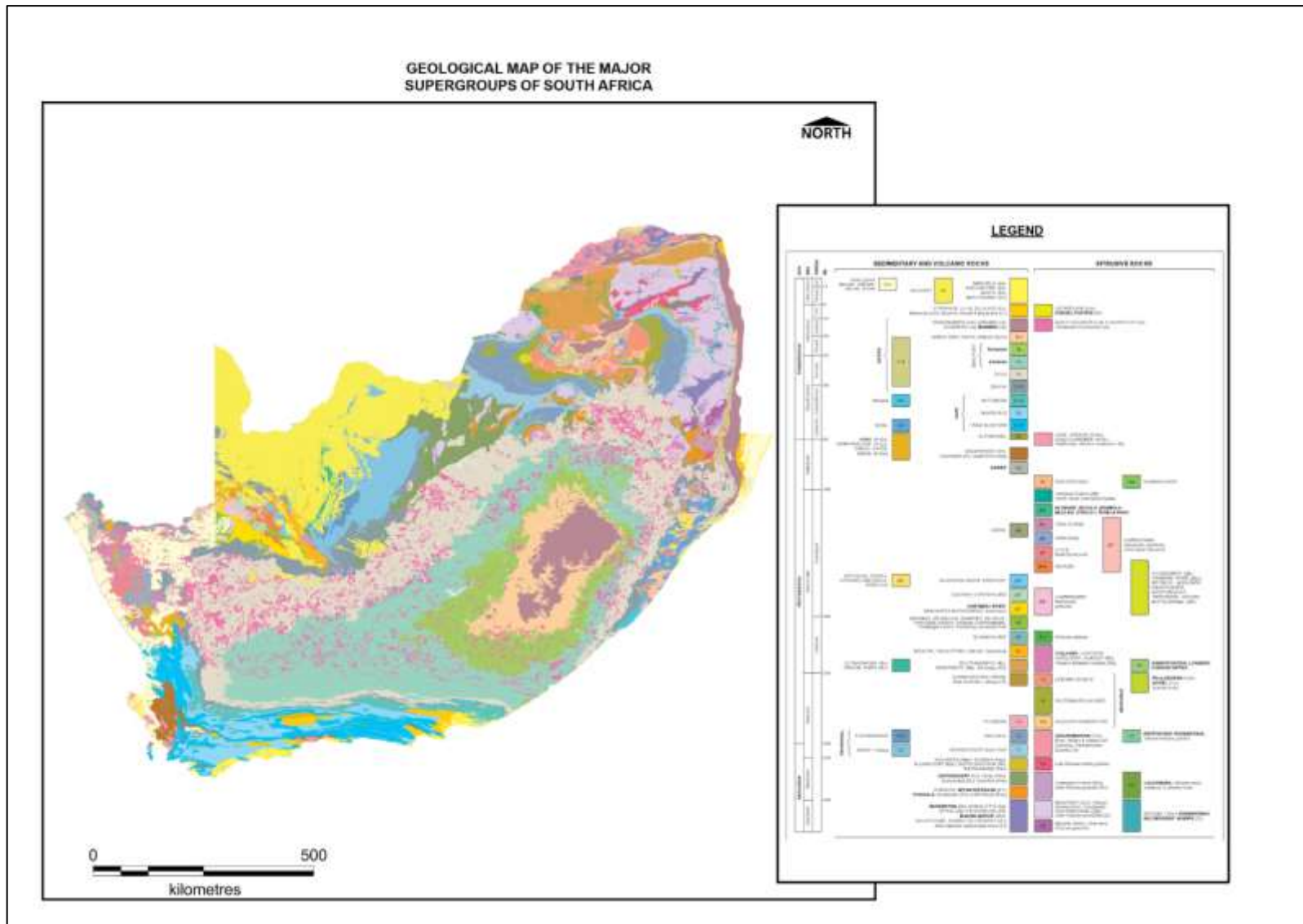
Figure 1: A schematic geological map of the outcrops of the Karoo Supergroup rocks in Southern Africa (Johnson, et al., 2006)

2.2.1 The Stormberg Group.

The Stormberg group represents the Molteno, Elliot and Clarens formations (Rogers & Schwarz, 1902). The Late Triassic Molteno formation comprises of alternating medium to course grained sandstone, with secondary quartz overgrowths providing the sandstones with a distinctive "glittering" appearance. This formation has a maximum thickness of approximately 600 m in the southern outcrop, and as little as 10 m in thickness towards the north. Deposition was predominantly by bedload dominated rivers flowing braided plains from a tectonic active source situated to the south and southeast (Johnson, et al., 2006).

The Elliot formation comprises from alternating fine to medium-grained sandstone and mudrock. These mudrocks can be identified by a red to greyish green colour and typically range in thickness between 25 to 100 m. The sandstone layers are predominantly a yellowish grey to a pale red colour and can be up to 22 m thick. The Elliot formation is typical of fluvial deposits, with flat bedding and trough cross-bedding (Johnson, et al., 2006).

The Clarens formation is a younger formation of the Late Triassic/ Early Jurassic period. This formation is associated with fine-grained aeolian sands with stream deposits. In the north these sand deposits are usually in the order of 100 m thick, and represent once desert-like conditions. In the southern regions of the outcrop, the formation is represents a homogenous siltstone and a silty fine-grained sandstone of approximately 300 m in thickness (Johnson, et al., 2006).

The hydrogeological characteristics and depositional history of the Molteno Formation shows that the formation should form an ideal aquifer. These sedimentary units are more persistent than those of the Beaufort Group and are sheet-like, which represents a more favourable aquifer geometry in terms of groundwater storativity. The Elliot Formation consists of red mudstone thus the formation will form an aquitard rather than an aquifer. The Clarens Formation consists almost entirely of well-sorted, medium to fine-grained sandstones deposited as thick layers. It is the most homogeneous formation in the Karoo Supergroup. Although the formation has a relatively high and uniform porosity it is poorly fractured and has a very low permeability, allowing the formation to store large volumes of water but is unable to release it quickly (Woodford & Chevallier, 2002).

2.2.2 The Beaufort Group.

The Beaufort group represents the transition from subaqueous (Ecca Group) to fully subaerial deposition. It consists of two subgroups, the lower Adelaide and upper Tarkastad subgroup and is comprised of alternating fine-grained lithofeldspathic sandstone and mudstone (Hancox & Gotz, 2014). This formation has a maximum thickness of about 5000 m in the southeast and decreased rapidly to about 800 m in the centre of the basin, and thereafter more gradually to around 100 – 200 m in the north (Johnson, et al., 2006).

The sedimentary units of the group generally have very low primary permeabilities. Aquifers of the Beaufort Group are multi-layered and also multi-porous with variable thicknesses. The contact between two different sedimentary layers will cause a discontinuity in the hydraulic properties of the composite aquifer. The complex nature of the Beaufort Aquifers is further complicated by the fact that many of the coarser and more permeable sedimentary bodies are lens shaped. The life-span of the high-yielding borehole may therefore be limited if the aquifer is not recharged frequently (Woodford & Chevallier, 2002).

2.2.3 The Ecca Group

The Permian Ecca group consists of significant different facies that occur near the centre and towards the edges of the outcrop. The proximal sector (towards the centre) has recently received renewed focus due to the discovery of shale gas in these formations. The Ecca Group can be subdivided into three major formations: the Pietermaritzburg Shale Formation, the Vryheid Formation and the Volksrust Shale Formation (Johnson, et al., 2006).

The Pietermaritzburg Shale formation (Lower Ecca Shales) consists mostly of dark grey, upward coarsening siltstone, mudstone and sandstones. The formation has a maximum thickness of 400 m in the southeast, thinning towards the north with the upper boundary overlying the Vryheid Formation (Johnson, et al., 2006).

The Volksrust Formation (Upper Ecca Beds) have a general thickness ranging between 150 m to 250 m and it is dominated by dark grey-green siltstone and mudstone carbonate concretions (Johnson, et al., 2006).

The Vryheid formation present in the Mpumalanga Province can be subdivided into distinct intervals: a lower fluvial dominated deltaic interval and a middle fluvial interval (Johnson, et al., 2006). The lower fluvial deltaic dominated interval (base) is characterised by an upward coarsening sequence of muddy siltstones deposited in anoxic shelf suspension conditions. These layers are overlain by bioturbated sandstones, siltstones and mudstones. Above this facies, is a facies of mouth-bar formed from a medium-grained sandstone, followed by a ripple cross-laminated fine-grained sandstone and siltstone. A mouth bar refers to section in a river that forms towards the middle of a channel in a river delta, created by mid-channel deposition. The mouth-bar facies is overlain by coarse to pebbly feldspathic sandstone (Johnson et al., 2006).

The middle fluvial interval includes a typically sheet-like geometry. Coarse-grained to pebbly sandstones found at the erosional base typically have an abrupt upward transition into fine-grained sediments and coal seams. Most of the economically important coal seams occur in this interval which grades into deltaic sediments in the southwest (Johnson, et al., 2006).

The Ecca Group consist of thick shales ranging from thicknesses of 1 500 m into the south and 600 m in the north. The porosities of the shale decreases from 0.10 % in the north to 0.02 % in the south and their bulk densities increase from 2000 to 2650 kg/m^3. This means that there is a possibility for economically viable aquifers in the northern parts of the basin (Woodford & Chevallier, 2002).

The deltaic sandstones represent a facies of the Ecca sediments in which one would expect to find high-yielding boreholes. However these sandstones are very low yielding due to the poorly sorted sandstones and that their primary porosities have been lowered by diagenesis (Woodford & Chevallier, 2002).

Table 1 lists common hydraulic properties of Karoo lithologies as discussed in (Botha, et al., 1998). Similar results are published in (Kruseman & de Ridder, 1994) for fractured rock aquifers.

Table 1: Typical Hydraulic Properties of Karoo Aquifers (Botha, et al., 1998).

Layer	Depth, m	K_h, (m · s^{-1})
Upper mudstone layers	8	9.910 × 10-7
	10	6.538 × 10-6
	12	3.601 × 10-6
Carbonaceous shale layer	14	8.796 × 10-7
	16	1.350 × 10-6
Sandstone matrix of the main sandstone aquifer	18	4.055 × 10-5
	20	1.345 × 10-4
Average depth of Mode 1 fracture	22	2.754 × 10-4
Sandstone matrix of the main sandstone aquifer	24	7.878 × 10-5
	26	2.205 × 10-6
Mudstone layers	28	2.309 × 10-7
	30	7.970 × 10-8
	32	4.440 × 10-8
	34	1.234 × 10-7
	36	1.564 × 10-7
	37	2.497 × 10-8
	38	1.740 × 10-8
	40	4.652 × 10-8
Kh=Horizontal hydraulic conductivity:		

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2.2.4 The Dwyka Group

The Dwyka Group is said to have formed during the late Carboniferous to early Permian age, and are the glacial forerunner to the Ecca Group. A number of lithofacies types have been recognised in the Dwyka group, and are considered to have been deposited in a marine basin. The Dwyka consists of three diamictite facies: the massive diamictite facies, the stratified diamictite facies and the massive carbonate-rich facies (Johnson, et al., 2006). The massive diamictite consist mostly of highly compacted diamictite, clast rich, with rounded to angular

striated stones. The stratigraphic diamictite consists of poorly to well-defined bedding planes of diamictite, mudrock, sandstone and conglomerate beds. The massive carbonate-rich diamictite are poor in clasts, and contains smaller angular stones, concretions and irregular bodes of carbonic rock (Johnson, et al., 2006).

The Dwyka Group hydrological characteristics consisting of diamictite and shale, both with low hydraulic conductivities ranging from 10^{-11} to 10^{-12} m/s and have little to no primary voids. This makes the Dwyka Group pertain to low yielding, fractured aquifers that are confined within narrow discontinuities such as joints and fractures. Therefore, aquitards are present in this group rather than aquifers. The sandstones that were deposited in the glacial valleys of the northern facies are very limited in extent and sealed off by the diamictite or mudstone. Most Dwyka sediments were deposited under marine conditions; the water in these aquifers is therefore saline. This makes the Dwyka Group not an ideal unit for a large-scale development of groundwater (Woodford & Chevallier, 2002).

2.3 Open Cast Coal Mining in Mpumalanga

Surface mining can be defined as the exploitation of a mineral at the surface without exposing any miners to underground working conditions. Surface mining methods in Mpumalanga are mainly subdivided into two main classes, mechanical and aqueous extraction methods. Table 2 below summarizes the different classification of mining methods.

Table 2: Classification of Surface Mining Methods modified from (Hartman & Mutmansky, 2002)

Locale	Class	Subclass	Method	Commodities	Relative Cost (%)
Surface	Mechanical	-	Open pit mining	Metals/ non-metals	5
			Quarrying	Non-metals	100
			Opencast strip mining	Coal, non-metals	10
			Auger mining	Coal	5
	Aqueous	Placer	Hydraulicking	Metals, non-metals	5
			Dredging	Metals, non-metals	<5
		Solution	Borehole mining	Non-metals	5
			Leaching	Coal, non-metals	10

The most popular of these surface mining techniques in the exploitation of coal is opencast or strip mining (Hartman & Mutmansky, 2002). Opencast mining differs from open-pit mining in the sense that the overburden removed is not disposed of but rather cast directly into adjacent mined out panels. This process of placing overburden materials in adjacent mined out panels allows the mining activities to be concentrated in a relatively small area, allowing reclamation to follow immediately and achieving higher productivity and often lower costs. The typical dimensions of the opencast strip are 30 to 60 m for the height of the highwall, 23 to 45 m for the width of the open cut, 60° to 70° for the slope of the highwall and 35° to 50° for the slope of the spoils (Hartman & Mutmansky, 2002).

The key to productivity in an opencast mine is the output of the stripping excavator. In smaller scale mines, stripping can be performed using load and haulage equipment. Larger scale mines preferably use draglines for stripping while load and haulage equipment are used strictly to mine the seam. The two major variations of opencast mining are area mining and contour mining. Area mining takes place on flat terrain with flat lying coal seams (Weyer, et al., 2017). Mining cuts are designed in long parallel cross strips. Contour mining usually takes place in hilly terrain, and design follows the contours of the topography. Other variations include box-cut and block-cut methods. These emphasise mining overburden on a smaller scale in rectangular blocks using dozers and scrapers, rather than mining a large parallel strip. In

addition, cast blasting or explosive casting can be used. This method entails blasting to move a portion of the overburden. With proper techniques and design, blasting can cast between 40 and 60% of overburden material into adjacent voids, which may save costs (Hartman & Mutmansky, 2002).

The cycle of operation can be described as follows: Stripping overburden; mining coal; and auxiliary operations. Stripping of overburden is largely determined by the nature of the overburden. Softer soil can be removed by excavation without prior breakage required (Weyer, et al., 2017). Harder rock will require drilling and blasting. Different types of drilling equipment and techniques will be used depending on the type of rock. Blasting transpires commonly using ammonium nitrate emulsions which can be loaded into blasting holes by hand or in bulk by machine. After the overburden has been slackened, excavation takes place usually by dragline or haulage and overburden is casted into the open void.

Once the overburden has been removed, mining of the coal seam can take place. This process may involve ripping, direct loading, drilling or blasting depending on the nature of the coal seam and hauled to the processing plant. Opencast mining of coal in South Africa is especially suited to the deposit due to most of the coal deposits being relatively flat, continuous and shallow. Table 3 summarises the advantages and disadvantages of opencast mining and open Pit mining (Hartman & Mutmansky, 2002).

Table 3: Summary of the Advantages and Disadvantages of Opencast mining and Open Pit mining (Hartman & Mutmansky, 2002)

Mining Method	Cycle of operations	Advantages	Disadvantages
Open Pit Mining	Stripping Overburden: drilling, blasting, excavation, hauling Mining coal: drilling, blasting, excavation, hauling Auxiliary operations: slope stability, dust control, waste disposal, drainage, and transport	High productivity Lowest cost of the broadly used mining methods Low labour costs Ideal for use of large equipment Fairly low rock breakage costs Good recovery	Limited by depth ± 300 m Limited by stripping ratio (0.8-4m^3/ton) High capital investment Requires large equipment to lower cost Weather detrimental Slope stability checks Pit may fill with water after mining
Opencast Mining	Stripping burden: determined largely by the nature of the overburden (soft soil; clays; hard rock) Mining coal: depending on the nature of coal-direct loading; ripping; drilling; blasting Auxiliary operations: reclamation; slope stability; road construction; maintenance; drainage and pumping; communications; dust control and safety.	Highest productivity of any coal activity Lowest cost per ton for coal mining High production rate Rapid exploitation Low labour intensity Able to increase production when needed (flexible) Low blasting costs Simple development and access	Technical limits of equipment impose depth limits (± 90 m) Economics pose limits on stripping ratios (1m – 19 m^3 per ton) Surface damage; extensive environmental reclamation required; often environmental expense is substantial Weather can impede operations; Slopes must be monitored.

2.4 Mine Water and Mine Water Management (Mitigation Measures)

There are several systems available for the treatment and management of mine water (Table 4) and can be divided into two main categories (Geller, et al., 2013) (Younger & Robins, 2002): namely active systems and passive systems.

Table 4: Comparison of Treatment Systems; from ((INAP), 2009)

Criteria	Active	Passive
Period in mine's life cycle	Exploration and operational phase: a workforce is required on site for implementation, control and maintenance. Application in post-closure phase generally only feasible for large volume flows.	Decommissioning, closure or post-closure phases as processes are largely self-sustaining.
Financial consideration	High capital investment and operational cost.	Medium capital cost and low operation and maintenance costs.
Power supply	Mechanical or electrical energy required.	No external power supplies. Use of natural energy sources (solar energy and gravitational flow)
Supervision	High degree of operating supervision and on-going maintenance.	No operators or constant supervision/regular maintenance is required.
Flow rates	Can handle very high flow rates or water volumes depending on design.	Optimum performance at lower flow rates of 0.1 – 2 ML/d. Unlikely to be considered for flow rates > 5 ML/d.
Input material	Generally requires ongoing addition of chemicals, power supply and equipment maintenance.	Natural, prolonged and self-sustaining treatment materials, but certain process technologies will require ongoing addition of chemicals in passive mode.
Treatment range	Can treat any constituent of concern.	Mainly applicable for acidity, metals and sulphate removal (not so much total dissolved solids TDS), electrical conductivity (EC), sodium (Na), and chlorine (Cl)).
Product	Produces very high quality water. Process is more reliable in terms of its output due to control. Product is certain.	Produces water of lower quality than active systems and of variable quality dependant on input water quality.

2.4.1 Active treatment technologies

Active treatment technologies may be defined as physical and/or chemical techniques used to treat contaminated water, and require ongoing human operations, maintenance, and monitoring. Active treatments make use of generated external energy resources and are commonly related to higher costing to construct and maintain. Active treatment is generally required to treat flow volumes in excess of 5 ℓ/s ((INAP), 2009). Common active treatment technologies include: aeration and neutralization (metal precipitation, metal removal, chemical precipitation, membrane processes, ion exchange biological sulphate removal. Identified active treatment technologies are discussed below (INAP, (2009) and Younger et al. (2002).

2.4.1.1 Aeration units

The principal metal contaminant associated with mine water is often dissolved ferrous iron (Fe^{2+}). Aeration is therefore required to increase the level of dissolved oxygen and promote the oxidation of iron and manganese. In water, approximately 10 mg/ ℓ of oxygen can be dissolved, and significantly increases the chemical treatment efficiency which lowers costs. Additionally, aeration contributes to the release of carbon dioxide, and can aid in the increase of pH which decrease the use of chemical reagents ((INAP), 2009) (Younger & Robins, 2002). There are numerous techniques used to apply aeration to mine water. The most common include: gravity (cascading, trickle filtration), mechanical (in-line venturi; mixers) and biochemical oxidation (Younger et al., 2002).

The cost associated with various aeration technologies are mostly related with the size of the plant and the quantity and quality of water needed to be treated. The cost can be broken up into operational cost (electricity consumption, hidden cost such as the dissolution of carbon dioxide from air, which may result in the increase of lime consumption and the increase in sludge production), and capital costs (purchase price of plant/ blowers/ air distribution systems and radial agitators) ((INAP), 2009).

2.4.1.2 Neutralisation/Hydrolysis

Table 5 represents a variety of neutralisation materials used for mine water treatment ((INAP), 2009). These neutralisation chemicals are generally added to mine water as powders, slurries and liquids. The selection of alkali material is based mainly on three characteristics. Firstly, the review of the secondary impacts associated with the chemical used to treat mine water on the surrounding environment. Secondly, the cost of the alkali material and thirdly, the treatment objective and the specific chemical characteristics of the mine water (metals that need to be removed /prove to be problematic) ((INAP), 2009).

Table 5: Alkali Materials and Compounds applied to ARD Treatment (from International Network for Acid Prevention (INAP), 2009)

Alkali Compound/Material	Alkali Requirements (t/t of Acidity)	Neutralisation Efficiency (% of Applied Alkali Used)	Relative Cost ($/t)
Limestone, $CaCO_3$	1.00	30 – 50	10 – 15
Hydrated Lime, $Ca(OH)_2$	0.74	90	60 – 100
Unhydrated (Quick) Lime, CaO	0.56	90	80 – 240
Soda Ash, Na_2CO_3	1.06	60 – 80	200 – 350
Caustic Soda, NaOH	0.80	100	650 – 900
Magna lime, MgO	0.4	90	Project Specific
Fly Ash	Material Specific	–	Project Specific
Kiln Dust	Material Specific	–	Project Specific
Slag	Material Specific	–	Project Specific

- ➢ Lime ($Ca(OH)_2$)

Hydrated lime is one of the most cost-effective ways to treat large-flow, high acidity decant mine water and can be applied either as a dispersion powder, or as a lime slurry. Generally, the powder dispersion application is preferred to the lime slurry as it does not require complicated design and maintenance. Lime neutralisation is most commonly used in industry due to the relative low cost, efficiency, the application thereof, good water and solids separations, and because it is a robust process able to treat variable flow volumes and acidity loadings ((INAP), 2009).

The selection of the most efficient lime treatment technique for pre-treatment and neutralization of pH is site-specific and project related, with an end goal in mind. The following characteristics will effect decision making ((INAP), 2009):

- Mine water (decant) flow rates
- Mine water chemistry (acidity / metals loadings)
- Site location/ site size
- Capital investment for rehabilitation
- Operating/ maintenance costs

Figure 2 represents the typical generic waste water treatment with lime. Lime-slurry and AMD are mixed in sludge conditioning tanks and passed through the neutralization reactors. Treated AMD is discharge, while the sludge is recycled for re-use or disposed of. Lime slurry piping requires careful design and maintenance due to the tendency of the lime to coagulate in the piping system under certain conditions (Skousen, et al., 1998).
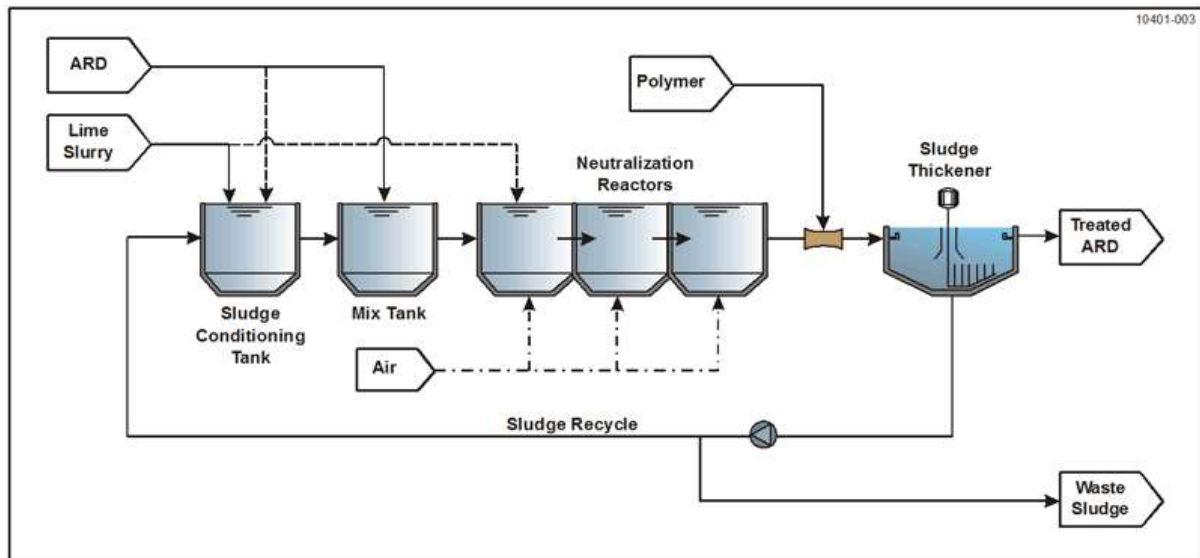
Figure 2: Application of lime to mine water ((INAP), 2009).

The application of lime becomes less cost effective if a very high pH concentration is required to precipitate metals (such as manganese). Applying excess amounts of lime may result in large volumes of unreacted lime which may increase sludge volumes, or raise the pH to such an extent that may result in certain metals such as aluminium to re-dissolve and be an effective hazard to human health ((INAP), 2009). Therefore the application of lime is an ongoing monitoring process that requires the re-evaluation of the quantity applied to mine water.

➢ Limestone

Limestone as with lime, has a low material cost and is the safest and easiest to handle for the treatment of mine water. It is ideally used when the metals of concern are iron and aluminium. However the successful application thereof is limited due to the low solidity and the tendency to form an external coating that limits the reactive chemical potential of limestone. To combat this process, when applying limestone, it should be fined grained increasing the reactive surface area, and allowing the limestone to dissolve before the armour process begins ((INAP), 2009). The applications methods used include diversion wells and trenching.

➢ Other neutralization techniques

Caustic soda (NaOH) and Soda ash (Na_2CO_3) is generally used in remote locations where electricity is unavailable. NaOH is supplied in a liquid and pellet form and is ideally used for

low flow, high acidity solution. Caustic soda is very soluble in water, disperses rapidly and can quickly raise pH. However, the purchase of NaOH comes at a high cost and is considered a hazardous substance to store and handle. Its application is rather for convenience than cost. NaOH supplied in the briquette form is applied by dumping, while the liquid form can be incorporated in a dosing plant. Soda ash is mostly used in emergency situations were cost is not of concern ((INAP), 2009).

2.4.1.3 Metal removal

Metal removal in industry is based on the approach of metal precipitation by chemical treatment. Precipitation of metals allow for the separation of solid metal precipitates from mine water as insoluble hydroxides, carbonates and sulphides. Many metals have an amphoteric property, allowing it to react as both a base and an acid. With the increase in pH, many metals decrease in solubility up to a threshold pH, where thereafter the metal solubility increased again due to the formation of more soluble complexes. Table 6 shows the pH corresponds to the thermodynamic solubility of some selected metal hydroxides ((INAP), 2009).

Table 6: Theoretical minimum metal hydroxide solubility pH altered from ((INAP), 2009)

Metal	pH Corresponding to minimum Metal Hydroxide Solubility
Ferric iron (Fe^{3+})	± 3.5
Aluminium (Al^{3+})	± 4.5
Lead (Pb^{2+})	± 6.5
Copper (Cu^{2+})	± 7.0
Ferrous iron (Fe^{2+})	± 8.0
Zinc (Zn^{2+})	± 8.5
Manganese (Mn^{2+})	±10.6

Metal removal by precipitation typically involves the addition of alkaline material to target pH for the removal of certain metal types. Once the required alkali or chemical has been added to the mine water, it is commonly directed to sedimentation ponds or mechanical thickeners to promote precipitation and settlement. Pre-aeration treatment also has benefits as most metals can exist in more than one oxidation state ((INAP), 2009).

Post chemical treatment, the sludge from the sedimentation ponds are disposed of. Sludge disposal may include leaving the submerged material in the pond indefinitely, sludge hauling from ponds to dispose of in open mine voids or pits, or dumping sludge in tailing facilities (Younger & Robins, 2002).

2.4.1.4 Chemical precipitation for sulphate removal

The desalination process in the mine water treatment process generally focuses on the removal of sulphate, sodium and chloride. Sulphate is typical of many mine drainages and is often considered the primary contaminant. However, during the initial neutralising process by the addition of lime/ limestone, limited sulphate removal takes place as gypsum precipitation. More effective processes such as the addition of barium sulphate or the ettringite precipitation process, also referred to as the SAVMIN process (Younger et al., 2002) has been developed.

The barium sulphate process is based on the addition of a barium salt to re-precipitate sulphate. The resulting sludge is then removed, the accessed barium removed and re-used in the process ((INAP), 2009). The SAVMIN process is described by Younger et al. (2002) as similar to that of the barium sulphate process, with the addition of calcium in order to super-saturate the solution with the addition of aluminium hydroxide in a high pH environment, resulting in the precipitation of ettringite:

Hydronium: protonized water

$$6Ca^{2+} + 3SO_4^{2-} + 2Al(OH)_3 + 38H_2O = Ca_6Al_2(SO_4)_3(OH)_{12} \cdot 26H_2O + 6H_3O^+$$

Hydrated calcium aluminosulphate

Formula adapted from ((INAP), 2009).
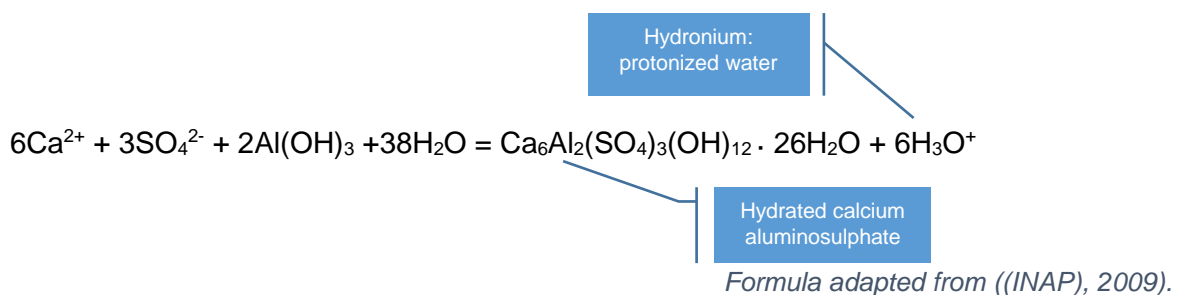
The process proves effective as precipitated gypsum can be sold, and is successful in removing metals such Fe, Mn, and Zn.

2.4.1.5 Membrane Treatment

Membrane treatment is the removal of solutes from mine water by forcing the water at high pressures through a membrane material (Cartwright, 2013). Younger, et al. (2002) classifies membrane treatments as follows:

- Microfiltration: the process by which bacteria are removed from feed water. Membranes used generally have pore sizes ranging between 0.1 and 0.45 µm.

- Ultrafiltration: the process by which colloids are removed from feed water. Membranes used generally have pore sizes ranging between 0.01 and 0.1 µm.

- Nanofiltration: colour is removed from water using this process. Membranes used generally have pore sizes ranging between 0.001 and 0.01 µm.

- Reverse osmosis: this process is used to remove solutes from the feed water. Membranes used generally have pore sizes smaller than 0.001 µm.

Figure 3 represents the typical generic waste water treatment through membrane desalinization. AMD passes through a set of reactors that neutralise and softens the water, before going through the filtration membrane system followed by reverse osmosis. Treated AMD is discharged, while waste water is left to crystallize in evaporation ponds, or discarded as sludge brine.
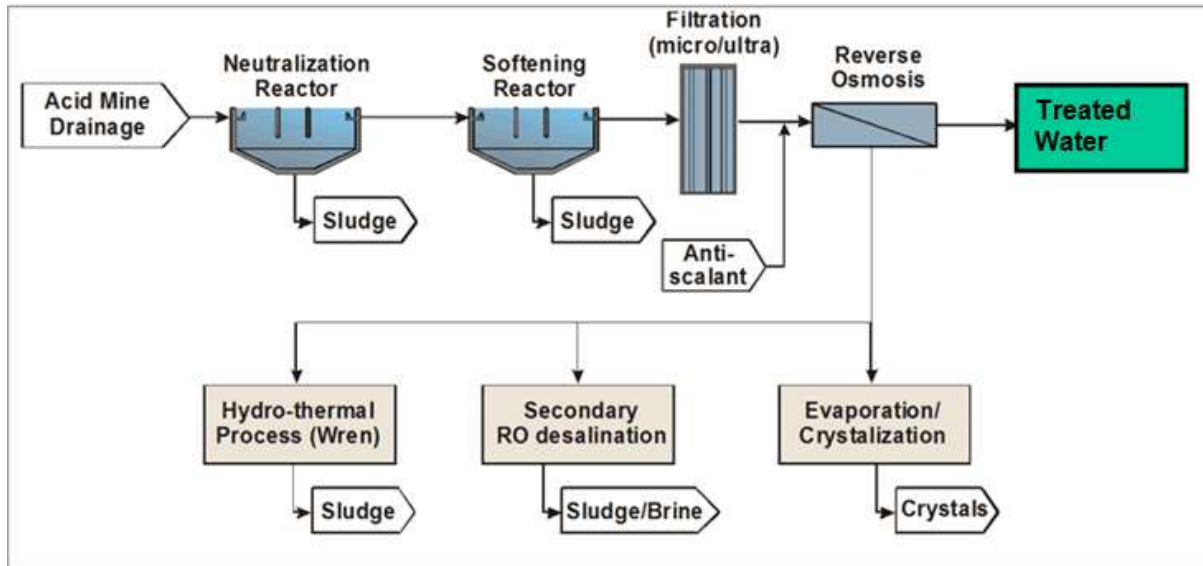
Figure 3: Conceptual High Recovery Membrane Desalination Plant ((INAP), 2009).

In some instances, the application of a reverse osmosis membrane treatment system to mine water is likely to be problematic due to the scaling and fouling potential of mine water. Therefore pre-treatment of mine water is essential before membrane treatment ((INAP), 2009).

Additional, there are higher costs involved to maintain the pressures in the system at which water is forced through the membranes. The process commonly leads to the formation of sludge and brine which requires disposal. However, some membrane technologies produce sellable volumes of precipitated salts from which costs can be recovered when high volumes of highly polluted water are treated (Younger et al., 2002).

The membrane treatment process generally transpires in the following steps:

- The mine water is pre-treated with an alkali compound or material such as lime, limestone or sodium hydroxides to precipitate metals and gypsum, which limits the scaling potential of the water.

- After the removal of residual suspended solids, the pH of the mine water is adjusted and an anti-scaling agent is added. This is generally done using settling ponds, trenches and mine water feed dams.

- Feed water is then forced through the selected membranes at high pressures.

- After the water is treated, simple water quality parameters such as pH may require to be adjusted.

2.4.1.6 Biological sulphate removal

The International Network for Acid Prevention describes biological sulphate removal is generally performed in the following sequence ((INAP), 2009):

- AMD is pre-treated to remove metals by precipitation as sulphides, hydroxides, or/and carbonates.

- The charge of the water is then adjusted. This can be done by adding an electron donor to the water, commonly in the form of sugars, alcohols, hydrogen gas or even sewage sludge.

- Nutrients in the form of nitrogen phosphate, potassium and other trace minerals are added to aid in microbial sulphate reducing conditions.

- The final step includes the reduction of sulphate in an anaerobic reactor. The process is mediated by sulpher reducing bacteria, which ultimately reduces sulphate to sulphide.

2.4.1.7 Sulphide precipitation

Sulphide precipitation is a process that converts soluble metals in solution into more insoluble metal sulphide compounds through the addition of sodium sulphide (Na_2S), sodium hydrosulphide (NaHS), ferrous sulphide (FeS) and calcium sulphide (CaF). The process of sulphide precipitation is effective because it can be applied over a wide range of pH's due to the high reactivity of sulphide with metals in solution. It is however, generally induced under neutral conditions (pH 7.0 to 9.0) ((INAP), 2009).

As with many other treatment technologies, sulphide precipitation is mostly used after the application of a neutralisation agent such as lime, limestone or sodium hydroxides. The advantages of sulphide precipitation treatment includes effective metal removal, low retention time required, and reduced sludge volumes. The disadvantage of the metal sulphide sludge is that it must be removed from the treated water by flocculation, filtration or coagulation. Other disadvantages include potential for toxic hydrogen sulphide gas emission and residual sulphide in treatment effluent ((INAP), 2009).

2.4.2 Passive treatment technologies

The Global Acid Rock Drainage Guide (GARD Guide) describes passive treatment as processes that do not require regular human interference, operations, or maintenance. It generally comprises the use of natural material such as soils, clays, straw, wood chips, manure, and compost, and mostly uses gravity flow for water movement ((INAP), 2009).

Passive treatment systems have low energy requirements and only slight physical processes (mixing, aeration) or chemical processes (chemically induced precipitation, oxidation) are needed, if any. However, because of the low energy requirements, longer retention time is needed ((INAP), 2009). The generic categories of passive treatment systems are listed in Table 7.

Table 7: Generic categories of Passive Treatment Systems ((INAP), 2009)

Passive treatment technology	Application Niche in Mine Drainage
Aerobic wetlands	Net alkaline drainage
Anoxic limestone drains	Net acidic, low Al^3, low Fe^3, low dissolved oxygen drainage
Anaerobic wetland	Net acidic water with high metal content
Reducing and alkalinity producing system	Net acidic water with high metal content
Open limestone drains	Net acidic water with high metal content, low to moderate SO_4

Various passive mine water treatment technologies are available and are listed and described below (International Network for Acid Prevention (INAP), 2009, and Younger et al., 2002) and (Watzlaf, et al., 2004).

2.4.2.1 Aerobic wetlands

Aerobic wetlands are ideal in providing environmental friendly conditions that promote the removal of suspended solids ((INAP), 2009). In addition, selected metals can be targeted. Generally these are ferrous iron that can be oxidised, and ferric iron that can be precipitated through hydrolysis and sedimentation (Younger, et al., 2002). The features associated with aerobic wetlands are:

- Shallow water to allow aeration of mine drainage

- Cascading structures that may enhance aeration

- Wetland vegetation which may enhance aeration and promotes favourable flow conditions

- Longer retention time to enhance treatment reactions

- Promotes the settling and accumulation of precipitates

- Promotes algae progress to increase the pH and facilitate manganese oxidation

2.4.2.2 Anaerobic wetlands

Anaerobic wetland system depends on reduction by chemical and microbial reactions to precipitate metals from solution and neutralise acidity. This is incorporated by allowing water to infiltrate thick vegetative organic material with a high biological oxygen demand ((INAP), 2009). In these conditions, sulphate may be reduced to hydrogen sulphide gas. Additionally, several other chemical reactions can take place within the system that will reduce metals, and the formation of compounds such as iron carbonates and iron sulphide is not uncommon (Younger et al., 2002).

2.4.2.3 Anoxic limestone drains (ALD)

Anoxic limestone drains are buried trenches or drains into which mine water discharge is introduced. The ALD is buried to produce anaerobic conditions to prohibit very little available oxygen and to accumulate as much CO_2 as possible ((INAP), 2009). This process allows the limestone to dissolve in the acidic water, adding alkalinity and raising the pH.

2.4.2.4 Reducing and alkalinity producing systems (RAPS)

When mine water contains dissolved oxygen or ferric iron, a reduction and alkalinity producing system (RAPS) will function better and is preferred to the limestone drain ((INAP), 2009). These systems are designed to strip dissolved oxygen from the water by reducing ferric iron to ferrous iron. This allows RAP to treat a wider range of mine water composition (Younger, et al., 2002).

2.4.2.5 Open limestone drains (OLD)

Open limestone drains (OLDs) are intended to introduce alkalinity and potentially raise pH in discharged mine waters. Generally, long channels of limestone are used to transport mine water discharge to a stream, settling ponds or other discharge points ((INAP), 2009). The following features in limestone drains are recommended:

- Steep drain slopes of >20%
- High flow velocities to scour settled solids and clean precipitates from the limestone surfaces
- Ability to periodically flush the OLD and clear the precipitates and solids

2.4.2.6 Passive sulphate removal

Passive treatment systems that make use of high rates of sulphate reduction commonly work on the same principals as anaerobic wetland treatment systems. However, additional features to these systems include ((INAP), 2009):

- Selected organic materials are combined and introduced into the system to hydrolyse ligno-cellulosic materials that will produce volatile fatty acids. These acids will potentially drive the sulphate reduction process.
- Following the sulphate reduction process, the mine water is passed through sulphide oxidizing reactors to partially oxidise hydrogen sulphide to sulphur.

2.4.2.7 Manganese oxidation beds (MOB)

Manganese oxidation beds (MOB) are implemented in the final stages in the mine water treatment process and is similar to alkaline leach beds. These manganese oxidation beds are generally filled with limestone that is not completely engulfed by water to promote oxidation of manganese ((INAP), 2009).

2.5 Qualitative Comparison of Different Categories of treatment

Table 8: Qualitative Comparison of Different Categories of Treatment ((INAP), 2009).

Feature/ Characteristic	Active treatment	Passive treatment	In Situ Treatment
1. Application to phase of mining	Most appropriate to exploration and operational phases because it requires active control and management. Closure and post-closure applications mainly associated with large flows (RO plants).	Most attractive to the closure and post-closure phases, because it requires only intermittent supervision, maintenance and monitoring of self- sustaining process (wetland).	Appropriate to the exploration and operational phases because it requires ongoing operational and maintenance,
2. Operational involvement	Active and ongoing plant operations and maintenance system and personnel.	Constant operations not required, but regular maintenance essential.	Active and ongoing operational personnel required, but permanent presence on site not required.
3. Operational input and materials	Requires chemical, operations staff, maintenance staff, electrical power, continues and/or regular monitoring.	Self-sustaining processes, periodic maintenance, intermittent monitoring. May require replacement or supplement of materials at a low frequency.	Requires chemicals, operations staff, intermittent field maintenance, electrical power and low frequency monitoring.
4. Supply power	Electrical and mechanical energy sources.	Natural energy sources of gravity flow, solar energy and bio-chemical energy.	Electrical and mechanical energy sources.
5. Management and supervision requirements	Ongoing management and engagement, constant facility supervision.	Low level management engagement and low frequency intermittent supervision.	High frequency supervision, but no permanent site presence required.
6. Range of Application • Flow rates • Constituents of interest	Application to all flow rates, especially high flow rates and any constituent of interest.	Mainly applied to low flow rates and acidity, metals, and sulphate removal.	Large spectrum of volume and flow application, mainly to deal with acidity and metals removal.

7.	Treated water quality	Treatment process can be purpose built to deal with spectrum of treatment water requirements.	Treated water quality poorer and more variable than another options.	Treated water quality lower and more variable than active treatment process.
8.	Waste sludge and brine production	Waste sludge and brine are produced, depending on level of treatment, requiring disposal.	No brine production, but longer term liability to deal with accumulation of pollutants in wetlands media.	Sludge and waste production accumulated in-situ, may pose long-term environmental liability.
9.	Capital Cost	High capital investment and periodic capital replacement required.	Moderate capital investment with periodic reinvestment to replace depleted wetlands media.	Low capital investment typically to deal with a short term problem.
10.	Operational and Maintenance Cost	High operation and maintenance cost, with some potential for cost recovery by sale of product water, metals and by-products.	Low operational costs.	Moderate operational costs, but chemical usage may be high due to process efficiency.

2.6 Rehabilitation Methods for Collieries

South African legislation (MPRDA, 2002) currently requires the rehabilitation of all opencast mines. However, technical information regarding the methods used for rehabilitation to avoid or reduce AMD in a South African context is minimal (Tanner, 2007). Several international authors have however published such findings.

The Handbook of Technologies for the Avoidance and Remediation of Acid Mine Drainage compiled by (Skousen, et al., 1998) discusses methods that can be employed to reduce AMD from overburden materials during and after backfilling the opencast mine. These methods include bactericides, alkaline additions, sewage sludge, encapsulation, removal of toxic material, selective handling, and reclamation.

The introduction of bactericides that are anionic surfactants to freshly excavated material in the form of a liquid substrate can suppress bacteriological activity and limit disulphide oxidation. The addition of alkaline materials to backfill is an attempt to raise the pH and potentially prevent disulphide oxidation (Skousen, et al., 1998) (Wisotzky, 2001). This method is also known as blending and can form precipitates which limit water movement through the material.

Limestone is the most readily used added alkaline material due to it being the least expensive, easy to handle and apply, and has a neutralising potential between 75 to 100 percent. The amount of added material is based purely on the Acid Base Accounting (ABA) of the backfill (Skousen, et al., 1998). The application of alkaline material does not necessarily ensure that it will neutralise acidity, and large emphasis has been placed on the method in which the alkaline material is added. For example, if water flow paths are primarily through the permeable acidic rocks, AMD can result where the water may discharge. Surface application of limestone has proven to have limited success. It is based on the premise that rain/irrigation water will dilute the limestone at the surface and infiltrate the alkaline solution to the underlying spoil. Therefore, for more sufficient results, a blending or mixing of alkaline material with the acidic rock should take place (Skousen, et al., 1998).

In the case where limestone is insufficient due to its limited solubility, other alkaline materials containing calcium and magnesium oxides can be used. These materials serve well when the pH has to be raised in excess of 8.3, and Mn concentrations needs to be decreased.

Alternatively, the addition of organic wastes and sewage sludge have been suggested as a stabilizer to acidity producing backfill materials. The bacteriological activity within these additions provide a more reducing environment, decreasing disulphide oxidation and indirectly raising pH (Skousen, et al., 1998).

The manner in which the overburden material is handled and exposed of during the exploitation process may have an influence on the acid producing potential of the materials. The acid producing materials should be disposed of in such a manner that the material is readily inundated after mine flooding. This requires the separation of acid producing material and non-acid producing material during mining, with acid producing material placed at the base of an opencast mine. To achieve the best result for this method, no intermissions of mining activities should occur, with continuous backfilling. This method is highly dependent on the speed of mining to be effective (Skousen, et al., 1998).

Evapotranspiration by plants are one of the more common methods used in minimising AMD formation. The regrading and revegetating promotes evapotranspiration and reduces recharge to the backfilled material. Adding limestone to backfill material, along with this approach, may potentially improve mine water quality to the point of compliance with effluent limits (Skousen, et al., 1998).

Additionally, guidelines by (Tanner, 2007) outlining methods of accepted soil overburden stockpiling and rehabilitation, during and after coal mining in South Africa, discusses the importance of concurrent backfilling of the opencast mine during mining. Backfilling should also be shaped to the desired topographical effect. This topography should be free-draining with slopes designed to ensure the minimisation of erosion, where slopes range between gradients of 1:5 to 1:7 for grazing land and 1:10 to 1:14 for arable land. The topography design should take into consideration the different types of materials, as some could cause expansion and shrinking by 25 and 15 percent respectively.

The current method for soil replacement in South Africa is by truck and shovel methods, replacing the soil in a single lift. A growing technique is the blast cut technique, where blasting is done in such a way that the spoil material fills the box-cut. The best practice for soil rehabilitation in South Africa according to Tanner (2007):

- A detailed soil stripping and stockpiling plan should be drafted prior to the commencement of mining. The soils must also be stripped and stockpiled according to form, prior to mining.
- A reserve of additional soils should be kept to repair areas of localised surface subsidence
- Compaction of soils must be kept to a minimum. This can be achieved by using the appropriate equipment. Additionally, soils should be replaced to the greatest possible thickness in single lifts

- Movement or transport of soils should be performed when the soil is in a dry state. This will aid in the minimisation of soil compaction. Wet soils should be moved strictly by truck and shovel methods

- In the event of restoration of multi-layered soil profiles, compaction of lower soil layers by heavy equipment must be kept to a minimum

- Compaction of soils during smoothing can be minimised by using dozers instead of graders

- Restored soils must be ripped to a sufficient depth for full root establishment of vegetation cover

- In the event that the natural re-vegetation of the area is not feasible, the restored soils must be tilled in order to create a seed-bed for the selected plant species utilised for artificial seeding.

Based on Tanner (2007), ponding of water on surface, increased groundwater make and surface cracking can be deduced in terms of rehabilitated land in South Africa and the associated challenges. The ponding of water on the surface of a post-mining landscape is commonly dealt with by installing trenches and/ or cut-off drains. These drains however have to be inspected on a regular bases to allow for optimum efficiency. The increases in groundwater make can be reduced by planting vegetation to increase the evapotranspiration.

Figure 4 is a representation of possible treatment methods, adapted from ((INAP), 2009)

Mine drainage treatment technologies can be broadly classified into active treatment and passive treatment. To determine whether treatment is required, the samples were evaluated to see to what extend it exceed SANS water quality standards. Should treatment be necessary, active and passive treatment may be decided based on the discharge flow capacity and/or the concentrations of SO_4.
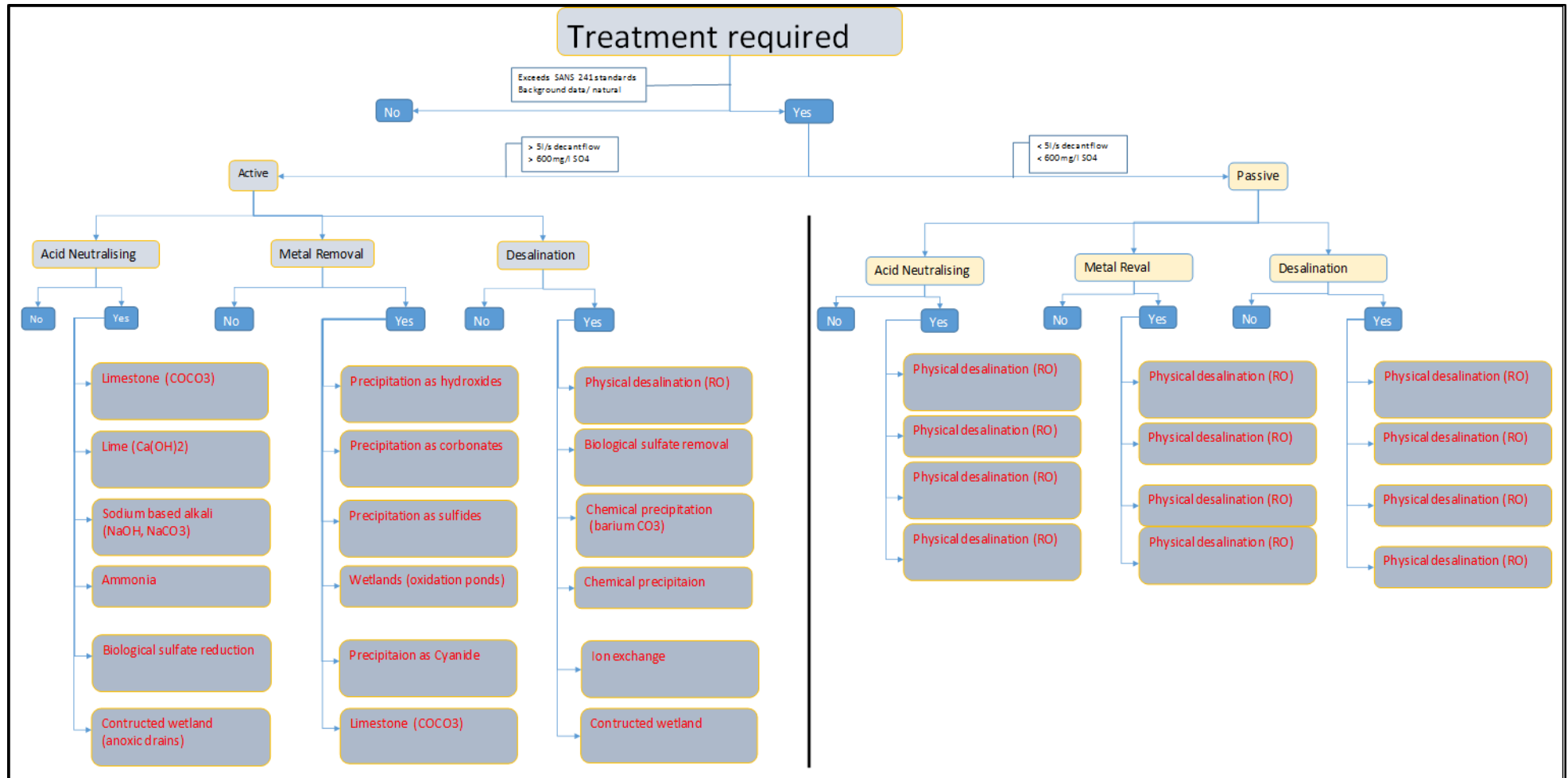
Figure 4: Representation of possible treatment methods, adapted from ((INAP), 2009)

2.7 Pre-treatment of Mine Water (Reverse Osmosis)

Reverse osmosis (RO) is one of many ways to get cleaner water from water that is contaminated or salty. The process forces water under pressure through a semi-permeable membrane/s (Chester, et al., 2016). Three alternatives to RO are briefly discussed below that uses the same process.

Nano-filtration is almost the same as RO with the exception that the membrane pores are slightly larger allowing monovalent ions to pass through while rejecting divalent ions. Ultra-filtration uses extremely small pore space membrane barrier that excludes particles such as bacteria, viruses and colloids from fluids. Microfiltration is a physical filtration process where contaminated fluid is passed through a specific pore size membrane (Chester, et al., 2016).

The primary pollutants in mine water discharge are identified as sulphate, iron, manganese, calcium, magnesium and acidity (pH). Iron and manganese can be removed by neutralisation, aeration and settling (Pinetown, et al., 2006). However, salts, SO_4, Mg and Ca need to be removed by active treatment methods such as RO plants (Chainbelt, 1970).

Most reverse osmosis plants will require some sort of pre-treatment to adjust pH (Chester, et al., 2016). Pre-treatment requirements will depend on the quality of the mine discharge, the volume of discharge, the size of the plant and the type of membranes used in the RO plant. Failure to meet the pre-treatment requirements may result in the damaging of membranes, resulting in poorly treated mine water discharge and high maintenance costs (Chester, et al., 2016).

3. METHODOLOGY

3.1 Hydrocensus

A hydrocensus involves gathering site-specific information on water features, water supply sources, and possible water pollutants. The purpose of the hydrocensus is to develop some sort of site familiarization. This includes identifying and gathering key information on boreholes, wells, springs, and other groundwater related information.

3.1.1 Boreholes

All boreholes in a 2 km radius were identified. A 2 km buffer was decided based on geological structure, water divides, and the estimated radius of influence of the mine. All boreholes within a 2 km radius of the site was visited, whether the boreholes were operational, dry, blocked or never in use. The status and use of boreholes are an important feature as it changes over time. Recording these statuses of each borehole can help to explain some aspects related to the aquifer, e.g. some water uses may result in lower water levels. Therefore, during the hydrocensus study, an effort was made to gather as much information as possible. The following information was recorded were possible:

- Borehole ID or name: The borehole ID or name is usaully written somewhere on the casings of the borehole. If no name could be identified, specific names were given to the boreholes. Naming conversion changes from site to site, but for the purpose of this study the first letters of the name of the farm were used followed by BH (borehole) and the number (PHBH1).

- Coordinates: it is imperative that the exact location of boreholes are recorded, as many of them will be revisited in the future.

- Elevation: Elevation is key in determining surface topography, drainage and water flow direction.

- Owner and Property details: Record the details to who owns the farm and their contact details.

- Borehole features: Borehole features such as casing heights, borehole diameter and casing type, and borehole construction. This information is generally difficult to get hold of.

- Static water level (SWL): The static water level of each borehole needs to be recorded. The result of this study is highly dependent on static water levels in various boreholes to model predicted decant volumes in the future.

- Use: The boreholes intended use was recorded. This included but was not limited to irrigation; livestock; and domestic.

3.2 Drilling

There are several different types of drilling methods used in South Africa. This section covers rotary air percussion drilling, which is a common drilling method used in South Africa. Rotary air percussion drilling was used in drilling of boreholes in this project. Three air percussion boreholes were drilled at strategically selected positions around the planned opencast pit to obtain detailed knowledge of the following site-specific groundwater characteristics:

- The hydraulic properties of the aquifer systems by means of hydraulic tests. This involved slug tests and one pump test to determine the aquifer parameters.

- The current groundwater quality in the vicinity of the site.

- To develop a monitoring network or for abstraction purposes, if required, and to determine the flow drivers to the mine extension area.

The percussion drilling was guided by the "South African National Standard, SANS 10299-4:2003; Development, Maintenance and Management of Groundwater Resources; Part 4: Design construction and drilling of water boreholes". According to the 2009 Environmental Impact Assessment report the local geology as pertained to the coal bearing seams can be described in Table 9.

Table 9: Summary of borehole logs

		Average	Maximum	Minimum
No. 2U TOP SEAM	FROM	17.685	23.000	8.690
	Elev Roof	1563.439	1567.697	1559.056
	TO	18.262	23.860	9.500
	THICK	0.163	0.860	0.000
	ELEV BASE	1562.862	1567.157	1558.930
No. 2U MAIN SEAM	FROM	16.962	24.640	9.060
	Elev Roof	1561.695	1567.758	1552.900
	TO	20.759	29.490	10.390
	THICK	2.434	5.550	0.000
	ELEV BASE	1557.898	1564.024	1551.570
No. 2U LOWER SEAM	FROM	21.28	30.410	11.120
	Elev Roof	1557.142	1564.917	1547.270
	TO	23.693	33.240	13.360
	THICK	1.980	3.9	0
	ELEV BASE	1554.73	1563.027	1544.55

3.3 Water Sampling

Sampling was done by collecting samples with a clean bailer after the boreholes were purged. Additionally, the borehole number, time, date and all other relevant observations were recorded. Before sampling commenced, the collar height (where applicable) of the borehole above general ground level and depth of the static groundwater level was recorded. An EC profile down the holes was done to detect changes in EC. These are then compared to borehole logs and an optimal sample depth is determined where the EC reaches its maximum or minimum.

A bailer attached to a rope was lowered and a water sample collected at a predetermined depth. The samples were submitted to a laboratory for the analysis of the parameters listed in Table 10. Please note this represents the worst case scenario.

Table 10: Parameters to be analysed by laboratory.

Parameter
Total Dissolved Solids
Suspended Solids
Nitrate & Nitrite as N
Chlorides as Cl
Total Alkalinity as $CaCO_3$
Fluoride as F
Sulphate as SO_4
Total Hardness as $CaCO_3$
Calcium Hardness as $CaCO_3$
Magnesium Hardness as $CaCO_3$
Calcium as Ca
Magnesium as Mg
Sodium as Na
Potassium as K
Iron as Fe
Manganese as Mn
Conductivity at 25° C in mS/m
pH-Value at 25° C
pHs by 21° Celsius
Langelier Saturation Index
Turbidity as N.T.U.
Aluminium as Al

3.4 Analytic Water Chemistry Analysis

Water qualities were compared to the SANS 241-1:2015 guidelines (Table 11) for human consumption. The SANS 241-1: 2015 standard is applicable to all water services institutions and sets numerical limits for specific determinants to provide the minimum assurance necessary that the drinking water is deemed to present an acceptable health risk for lifetime consumption.

Table 11: Water Quality SANS 241-1:2015 guidelines for human consumption.

Parameter	Unit		SANS 241: 2015 Recommended Limits	Risk
Physical & Aesthetic determinants				
Electrical conductivity at 25⁰C	EC	mS/m	≤ 170	Aesthetic
Total Dissolved Solids	TDS	mg/litre	≤ 1200	Aesthetic
pH at 25⁰C		pH units	≥ 5 to ≤9.7	Aesthetic
Chemical Determinants - Macro determinants				
Nitrate as N	NO_3	mg/litre	≤ 11	Acute Health
Nitrite as N	NO_2	mg/litre	≤ 9	Acute Health
Sulphate	SO_4	mg/litre	Acute Health ≤500; Aesthetic ≤250	Acute Health/Aesthetic
Fluoride	F	µg/litre	≤1500	Chronic Health
Ammonia as N	NH_3	mg/litre	≤ 1.5	Aesthetic
Chloride	Cl	mg/litre	≤ 300	Aesthetic
Sodium	Na	mg/litre	≤ 200	Aesthetic
Zinc	Zn	µg/litre	≤5000	Aesthetic
Chemical Determinants - Micro determinants				
Total Iron	Fe	mg/litre	Acute Health ≤ 2.0; Aesthetic ≤0.3	Acute/Aesthetic
Total manganese	Mn	mg/litre	Acute Health ≤0.4; Aesthetic ≤0.1	Acute/Aesthetic
Aluminium	Al	µg/litre	≤ 300	Operational

3.5 Aquifer Parameter Testing

3.5.1 Introduction

There are many methods to determine the physical properties of an aquifer. Some of the methods have limitations in certain aspects, while others are reliable but are time consuming and expensive. The aquifer parameter test methods utilized for this study was slug testing, and a constant rate test. A brief summary is discussed below on each of these methods.

3.5.2 Slug Tests

Literature defines a slug test as a simple method to determine localized aquifer parameters on a localized scale and is a short term method to determine whether further, more complicated tests are needed to be carried out, which are more time consuming and costly (Kruseman & de Ridder, 1994).

Slug tests were performed on one of the newly drilled boreholes. In summary the procedure was completed by removing a column of water from the borehole in a set time period while measuring the negative displacement of water and subsequent record the recovery to equilibrium to give the first estimation of the transmissivity of the aquifer surrounding the immediate vicinity of the borehole.

AqteSOLV was used to interpret the slug test to determine the hydraulic conductivity.

3.5.3 Constant Rate Test

The constant rate method can determine a number of aquifer parameters but is the most time consuming and the most expensive. Drawdown is measured against time for pumping a borehole at a constant rate. After the constant rate test was completed, a recovery test may be performed to determine the time it takes to recover to approximately 90% of the drawdown of the borehole and a sustainable yield may be calculated.

Cook (2003) explains that pumping tests either inject or remove fluid from a borehole and measure the response (change in pressure) of the aquifer in the same or in nearby observation boreholes. A model is used to estimate the hydraulic conductivity and transmissivity of the aquifer. The values of the aquifer properties that are obtained will depend on the model that is used to interpret the test results. Most models for interpreting pumping tests represent the aquifer as a homogeneous, isotropic porous medium, and adopt one of the three basic geometries (Campbell, et al., 1990). The constant rate test helps determine the sustainable yield of the borehole, one of the most important reasons a geohydrological investigation is done (Campbell, et al., n.d.).

A constant rate test was conducted on one of the newly drilled boreholes. The borehole was pumped to such an extent that an obvious observation can be made that the aquifer is

stressed. Thereafter, a recovery of the aquifer was recorded. AqteSOLV was used to interpret the results.

3.6 Conceptual Hydrogeological Model

The methodology used for the developing of a conceptual hydrogeological model for the rehabilitated opencast coal mine is discussed below with special focus on sulphate contamination.

A conceptual model is constructed as a simplified representation of reality. A conceptual model allows for the understanding of the hydrogeological processes active at a specific site for that specific period of time. However, the conceptual model is a simplified presentation of a system and must be updated regularly as more data becomes available (Huisamen, 2017).

During mining, an open void in the subsurface allows for the depressurization in the subsurface, causing a change in groundwater flow directions. Therefore a cone of depression forms towards the opencast pit. For continual safe work and productivity at the opencast mine, dewatering is imperative to drain the void of fluids. In South Africa, the technique used most entails stage mining of a single strip being mined and subsequently backfilled at any given time (Hartman & Mutmansky, 2002).

After the final strip of the opencast mine is backfilled, the backfill material has undergone several weathering and oxidation processes while exposed to surface conditions in overburden dumps. This results in secondary mineral formation within the spoils. Once mining operations has fully ceased and the opencast mine is backfilled, groundwater will rebound in the opencast mine and flooding in the backfilled opencast will occur. This process occurs due to groundwater flow characteristics, higher material hydraulic conductivity and recharge from rainfall (Tanner, 2007).

The backfilled opencast mine will therefore have a higher porosity and hydraulic conductivity that generally exceeds that of the surrounding hydrogeological aquifer system. This results in faster flow velocities and could result in flooding (Gomo & Vermeulen, 2013). This process results in rapid weathering of secondary minerals and the formation of AMD, until the backfilled mine is fully flooded. Once fully flooded the AMD process de-accelerates until steady state conditions are reached. The addition of water to the opencast mine through recharge and groundwater inflows, will cause the potential contaminants to dilute. This process will continue until the sources of contamination are depleted or chemical equilibrium is reached. Other processes such as dispersion and advection into the surrounding aquifer will occur which can generate a contamination plume migration which may impact sensitive receptors (Hartman & Mutmansky, 2002).

Table 12 summarises the input variables used to design a basic conceptual model represented in Figure 5.

Table 12: Input variables for the design of a basic conceptual site model.

Location	BH ELEVATION	SWL MAMSL	SWL (MBGL)
PBH3	1591.432	1586.217	5.215
PBH2	1586.619	1585.309	1.31
Start of Pan	1585.754	1585.954	-0.2
Centre of Pan	1585.334	1586.334	-1
End of Pan	1588.203	1588.303	-0.1
PBH1	1588.037	1586.362	1.675
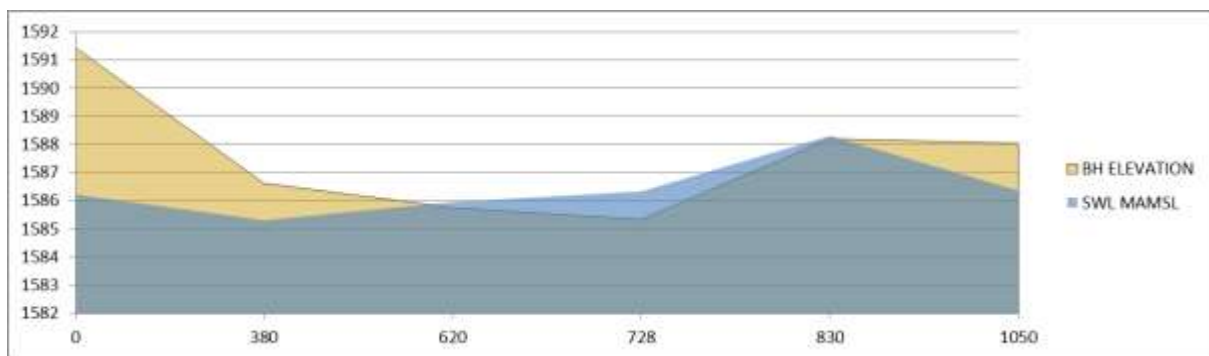
Figure 5: Basic conceptual site model

3.7 Analytical Model

3.7.1 Assumptions and Limitations

Analytical models are mathematical models that have a closed form solution, i.e. the solution to the equations used to describe changes in a system can be expressed as a mathematical analytic function (Marineli & Niccoli, 2000).

The (Marineli & Niccoli, 2000) analytical model is based on the following assumptions:

- Homogeneous and isotropic hydraulic conductivity distribution.

- Horizontal flow conditions.

- Infinite horizontal extent or limited boundary conditions.

- The geological materials are homogenous and isotropic.

- Groundwater flow is steady state, unconfined, horizontal, radial, and axially symmetric.

- Therefore, sustainable yield can be estimated without accounting for aquifer boundary conditions, aquifer geometry, and the spatial distribution of hydraulic properties of the system.

- Recharge is unfortunately disturbed at the water table and all recharge within the radius of influence is captured by the pit.

- Pit walls are approximated as a right circular cylinder.

- The static pre-mining water table is approximately horizontal.

- The base of the pit is coincident with the base of the aquifer, and there is no flow through the pit

3.7.2 Model Setup

Analytical solutions are most useful for evaluating simplified groundwater systems, and are based on a number of limitations, and can be useful in estimating mine water inflows and drawdowns when very little site specific data is available (Marineli & Niccoli, 2000).

The steady state two dimensional model from (Marineli & Niccoli, 2000) was used to estimates radial groundwater flow towards the opencast mine pit using the following formula:

$$h = \sqrt{h_p^2 + \frac{W}{K_h}\left[r_i^2 \ln\left(\frac{r}{r_p}\right) - \left(\frac{r^2 - r_p^2}{2}\right)\right]}$$

where

      h   = is saturated thickness above the pit base at r (distance from pit centre)

      h_p = is saturation thickness above the pit base at r_p (mine wall)

      W   = is distributed recharge flux

      K_h = is horizontal hydraulic conductivity of surrounding geological materials

      R_i = is radius of influence (max cone of depression)

      R   = is radial distance from pit centre

      r_p = is effective pit radius.

3.8 Numerical Flow Model

3.8.1 Introduction

The Groundwater Modelling System (GMS) was used to perform groundwater simulations. It consists of a graphical user interface (GUI) and a number of numerical codes. The GMS interface was developed by the Environmental Modelling Research Laboratory of Brigham Young University in partnership with the U.S. Army Engineer Waterways Experiment Station and is currently under the ownership of Aquaveo.

Numerical groundwater flow models were constructed to aid in the understanding the hydrogeology of a mining site and to act as transport models using selected geochemical modelling results as input. Additionally, the purpose of the numerical models was to develop a tool than can be used to assess the various hydrogeological influences on the aquifer as well as predict transport of contamination into the aquifer system.

Therefore, a constructed groundwater flow model will be used to estimate flow volumes entering and exiting the rehabilitated opencast pit, groundwater flow directions and flow rates, potential future discharge volumes and plume migration.

The following approach was used:

a. Identify the assumptions and limitations.

b. Construction of the conceptual model.

c. Setup of the numerical model. This included a description of the boundaries used and subdivision of the model into discrete finite difference cells.

d. Determine model parameters.

e. Model calibration.

f. Prediction of the groundwater drawdown due to dewatering at the Pit.

3.8.2 Assumptions and Limitation of Numerical Model

- Pit geometry and backfill porosity are highly variable, and best estimates were used on available data and literature.
- A steady state system was assumed based on minor to no disturbance to the aquifer system for the pre-mining scenario.
- Hydraulic properties of each geological layer are homogeneous considering the representative elemental volume (on a large enough scale the fractured aquifer will behave hydraulically as a porous medium).

- Limited static groundwater levels from monitoring data in the area of Mine X, and therefore the volume of water level data is a limitation on model accuracy.
- The pumping test and slug test properties were extrapolated to the entire model.
- Uniform recharge was assumed across entire model (for the pre-mining scenario).
- Recharge to the backfilled pit was assumed uniform (post mining) and estimates based on the values of (Hodgson & Krantz, 1998) were used.

3.8.3 Numerical Model Setup

Numerical groundwater modelling is considered to be the most reliable method of quantifying the likely impacts on the groundwater system.

The following hydrogeological characteristics were considered for the investigative area:

- First layer: The upper most lithological layer consists of a completely weathered material (soil layer), as shown in drilling and exploration logs. This highly weathered soil layer has a high hydraulic conductivity and groundwater is predicted to follow the topography.
- Second layer: The next twenty meters are divided into two predominant aquifer systems. The first, a highly fractured sedimentary aquifer system consisting of shale, mudstone and sandstones with a low hydraulic conductivity. The second will be the opencast mine backfill material. This backfill material will have a high hydraulic conductivity with preferential pathways.
- Generally, backfilled opencast mines hydraulic conductivity increases by an order of magnitude compared to the surrounding geological layers, and often decreases exponentially with depth (Jiang, et al., 2009).

3.8.4 Recharge

Groundwater recharge is one of the most important and most difficult parameter to calculate when evaluating a groundwater resource (Ping, et al., 2014). Groundwater recharge can be defined as any addition of water from surface, unsaturated zone or surface water body's sources. Several techniques can be used to estimate groundwater recharge. These include geochemical methods, isotopic methods, physical measurements, water –balance techniques, temperature profiling, electromagnetic and modelling approaches (Ping, et al., 2014). For the purpose of this study, the mass chloride method was used.

3.8.4.1 Chloride Mass Balance (CMB) Method

From a hydrochemical perspective, chloride is one of the most informative ion due to its conservative nature, and is the most popular method used to calculate groundwater recharge (Ping, et al., 2014). To calculate and quantify recharge, the following equation was used:

$$R = \frac{(P)(Clwap) + (Clwap)(Cldep)}{Clgw} \; X \; \frac{100}{P}$$

R   =   Recharge to groundwater (mm/y)

P   =   Mean annual precipitation (mm)

Cl_{wap} =   Weighted average chloride in precipitation (mg/l)

Cl_{gw} =   Average chloride in groundwater (mg/l)

Cl(dep)=   Dry deposition of Cl (mg/l)

Table 13 documents the input parameters for the recharge calculation

Table 13: Input parameters were used recharge for the study area

Parameter	Unit	Value	Listing
Precipitation (P)	(mm/year)	726	Calculated from (DWS, 2017)
Chloride in precipitation	(mg/l)	0.6	Cl in rain for inland
Chloride in groundwater	(mg/l)	0.06	0.1*(Cl in rain)

$$R = \frac{(P)(Clwap) + (Clwap)(Cldep)}{Clgw} \; X \; \frac{100}{P}$$

$$R = \frac{(726)(0.6) + (0.6)(0.06)}{12.59} \; X \; \frac{100}{726}$$

R = 5.2 %

3.8.5   Aquifer Parameters

Although the most relevant aquifer parameters are optimised by the calibration of the model, many parameters were calculated and/or judged by conventional means. The fixed assumptions and input parameters used for the numerical model are summarized in Table 14. A detailed description for the listed parameters, with referencing, is provided in Section 5.7.

Table 14: Input parameters to the numerical flow model

Model Parameter	Description
Recharge to the aquifer	Calculated using the chloride method (5% or ± 0.0001 m/day)
Recharge to backfill pit	20% or ±0.0004 m/day (Hodgson & Krantz, 1998)
Evapotranspiration	Estimated from literature (DWS, 2017)
Horizontal hydraulic Conductivity	Slug test and pump test data; and (Kruseman & de Ridder, 1994) to verify.
Hydraulic conductivity of backfill material	Estimated from literature
Effective porosity	Estimated from literature
Model Boundaries	Identified during desktop study
Geological layers	Estimated from literature and drill logs
Head error range	Calculated as 10 % of the difference between the maximum and minimum calculated head elevations

3.8.6   Model Boundaries

Model boundaries were identified on the premise that natural topographical features such as water divides and rivers will serve as either no-flow boundaries or constant head boundaries. To the north and west, a no-flow boundary was selected on the boundary of the Olifants Water Management Area. To the east and the south, constant head boundaries were applied for the Wilge River.

The Digital Elevation Model (DEM) was obtained from the ASTER Global Digital Elevation Model which is generally accurate to within five meters and users a ninety meter grid cell spacing. In addition, model grid refinement at the mining area allowed for a more accurate representation of reality. Figure 6 shows the DEM applied to the study area, while Figure 7 indicates the model boundary and refine grid where the void of the opencast mine will be located. The vertical delineation is represented in Figure 8.

Figure 6: Elevation points from DEM data
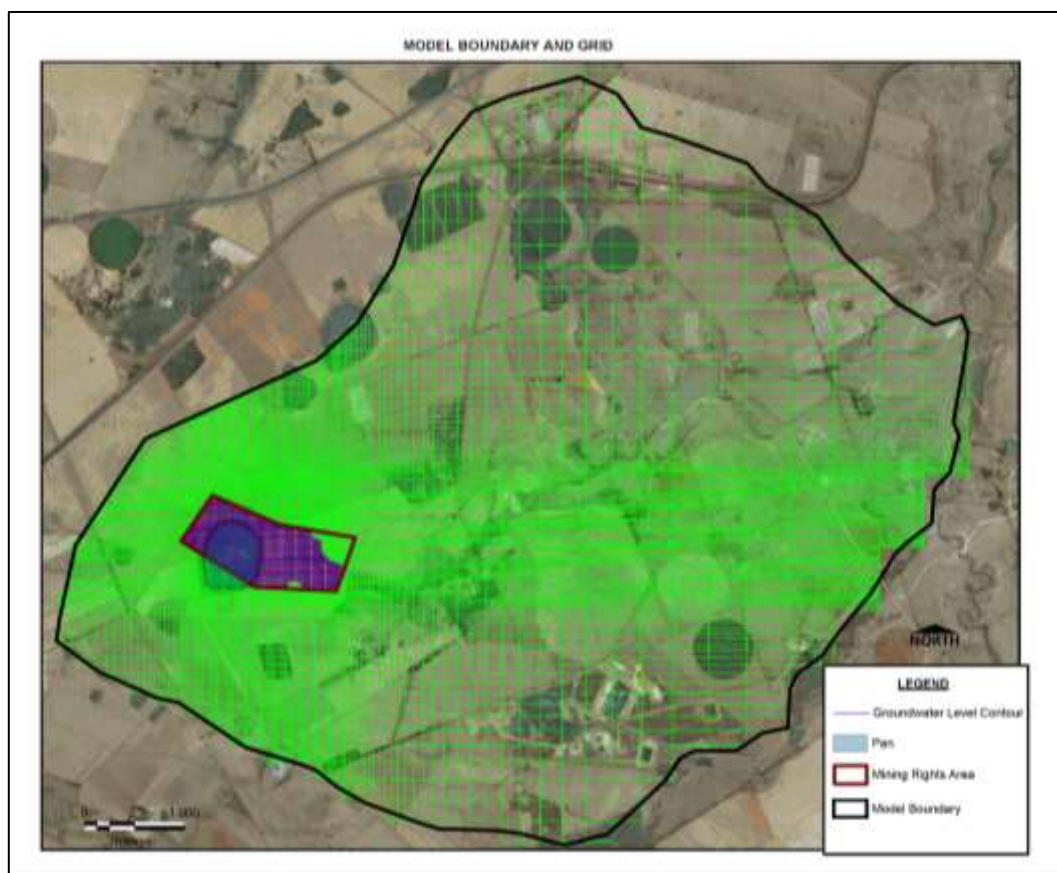
Figure 7: Model boundary and refined grid over the opencast pit
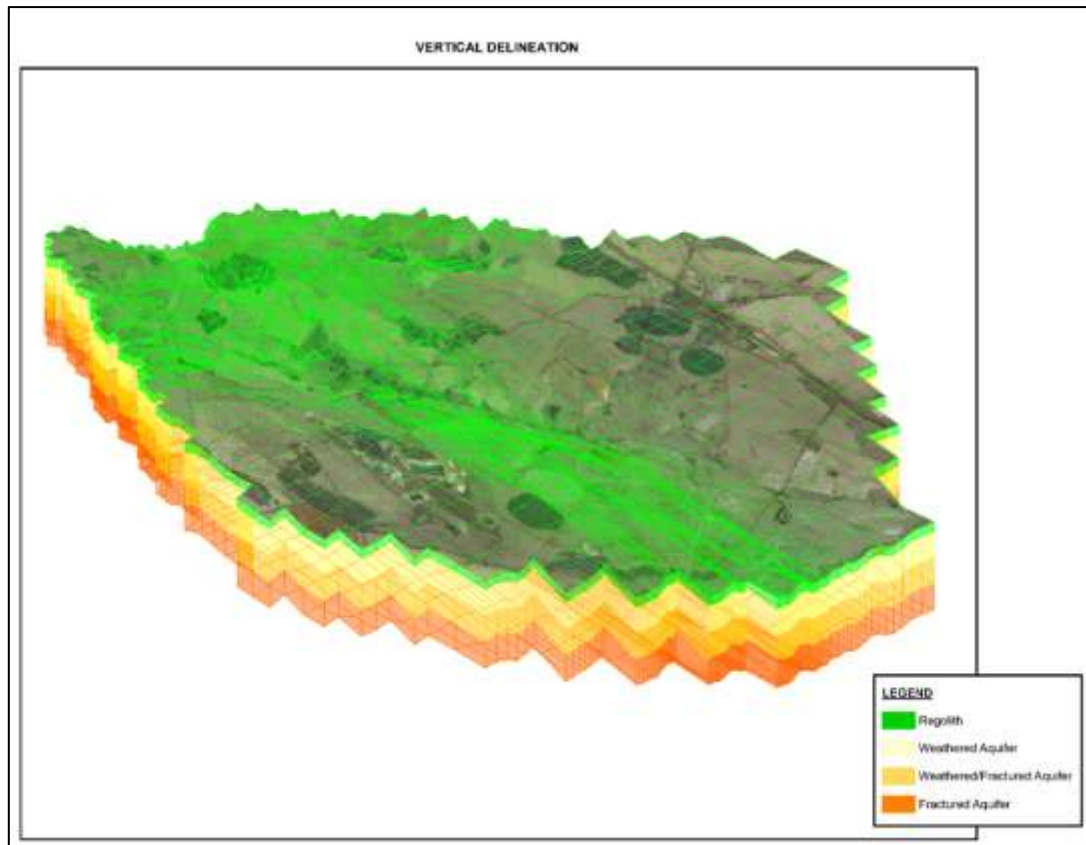
Figure 8: Vertical delineation of the study area

3.8.7 Calibration

To calibrate the model, the National Groundwater Archive (NGA) was searched and water level data collected during the hydrocensus was used. NGA borehole and hydrocensus boreholes were used for the calibration. Calibration was performed for steady state conditions. Water levels were calculated before mining, and compared to actual measured data. If these values fall within the allowed calibrated error range (10% of the maximum minus the minimum calculated groundwater head elevation) the model is successfully calibrated (Diersch, 2014). Drawdown was calculated using a steady state simulation for the "during" mining scenario. It was assumed that the entire pit will be dewatered simultaneously to represent a worst case scenario and achieve the maximum drawdown for the simulation. Post mining water levels were then calculated using a transient simulation to ensure that the complete flooded period is simulated. The calibration graph represented in Figure 9 shows the computed water levels versus the measured water levels at the study area. All values fall within the 10% of the maximum and minimum calculated groundwater head elevation, and the model can therefore be declare calibrated.

Limitation: Limited groundwater elevation data was available to calibrate the model.
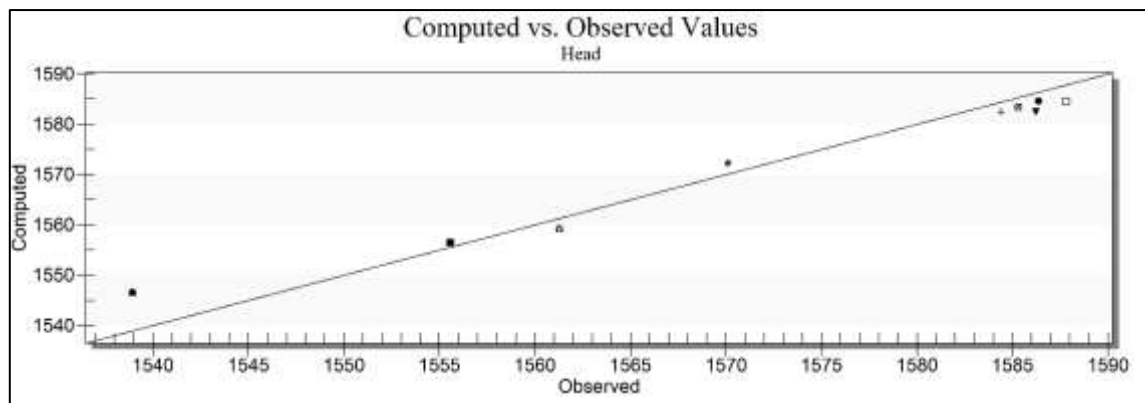
Figure 9: Transient head calibration graph for the numerical model constructed for Mine X.

Based on the calibrated model, calculated water levels where compared with measured water level. All data fell within the defined error range (Figure 10).
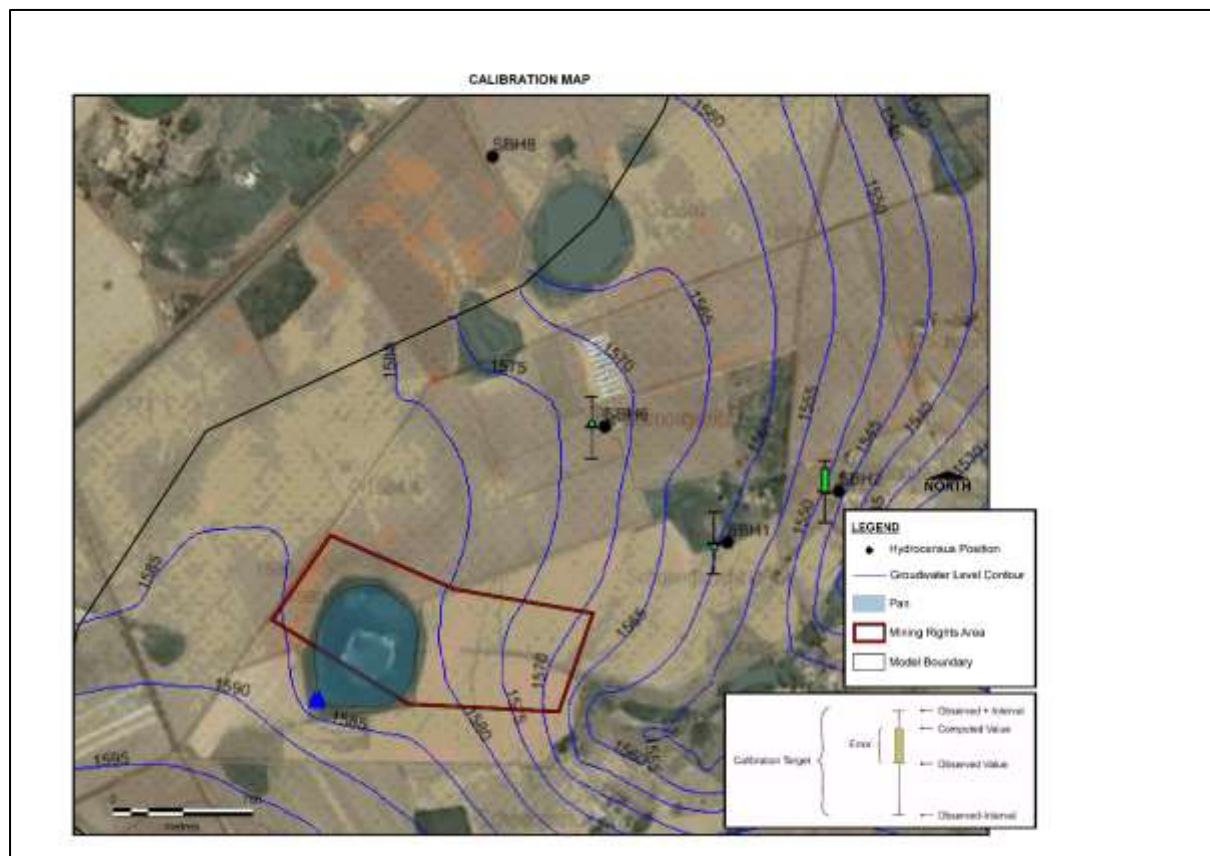
Figure 10: Calibration of the numerical model (calculated vs. measured water levels)

3.9 Statistical Analysis

Three decommissioned opencast collieries in the Witbank Coal Field with similar hydrogeological characteristics as Mine X, were identified to compare background hydrochemical data. These three mines were selected based on that they were similar in size, similar in the mining technique used, site location, geological makeup of the mining area, the climatic setting, and have similar aquafer parameters. All three mines are currently fully flooded and decanting. The comparison between the case study mines' background and decant hydrochemical data is probable to show that the expected Mine X decant hydrochemical signature will be similar to that of the case study mines.

For the statistical analysis, the data from three case studies (Mine A, Mine B and Mine C) were used to see if there are statistical relationships between the hydrochemistry of the different mines in terms of pre-mining water quality as well as post mining water quality.

To help determine any obvious and hidden relationships in groundwater quality data when comparing the three case study mines, statistical analyses will be used. IBM's SPSS 20 will be used to identify the constituent relationships in the geochemical make-up of the groundwater samples. To illustrate these relationships, a dendrogram will be created using Ward's method and the between groups linkage method with a square Euclidean distance interval with no transformation of values. A dendrogram shows clusters of samples which are grouped according to the degree of correlation between the constituents that make up the hydrochemistry of each sample.

The results of these statistical analyses will indicate if the hydrochemistry of the background samples of all four mines are similar. The same process will be repeated for the three case study mines' decant hydrochemistry. It is assumed that sample groups that show a comparative variance of 25% or more are significantly different from each other with regards to hydrochemical properties and composition.

In the event that the decant samples' chemistry show a variance of less than 25% between samples, the assumption will be made that the decant chemistry of Mine X is likely to have a similar hydrochemistry. This will then be used as input to a geochemical model to simulate potential treatment methods and the efficiency of these methods.

Additionally, a statistical analysis will be used to determine the relationship between pH and SO_4, and how these correlate over time. The same approach was used as above, with data from three case study (Mine A, Mine B and Mine C) was used. The results of these analysis will help estimate/predict the dynamic incline of pH of the decant concentrations from the opencast mine. More is discussed on this process under Geochemical Model.

3.9.1   Limitations and Assumptions

- Volume of data: increasing the volume of data will always almost with certainty, influence the accuracy of the statistical model.

- Different cluster analysis method give approximately the same results which lead to the assumption of data representativeness.

- In the statistical analysis a variance of 25 % was considered to be significant.

- If the groundwater chemistry data corresponds between the sites, it is assumed that the mathematical expressions developed subsequent to the statistical analysis will be accurate.

- From a review of Mine A, Mine B and Mine C, it is evident that all of these mines in the Mpumalanga coal field have similar geology, climate, mining techniques, size, life of mine, similar overburden exposure time and similar backfilling techniques. For the purpose of this study it is therefore assumed that if pre-mining hydrochemistry is similar, so too will be the decant water hydrochemistry.
  - Eliminates the necessity for in-pit geochemical modelling due to measured values.

3.10   Geochemical Model

The Geochemists Workbench® version 9.0 will be used to model and interpret hydrochemical data to aid in identifying possible the most effective pre-treatment chemicals, the success of these chemicals, as well as the quantities of chemicals needed. To determine the input concentrations to the model, the average decant concentrations of three case study mines namely Mine A, Mine B, and Mine C as calculated in the statistical model will be used.

The aim of the geochemical model is to help determine the most successful pre-treatment chemicals and the quantity thereof, and determining the most cost effective for Mine X. Once the most successful and cheapest has been identified, the application of these methods will be taken into account.

3.10.1 Assumptions of Geochemical Model

Although all efforts will be made to base the geochemical model on the most representative data, some limitations and certain assumptions will be made:

- Pit geometry and backfill porosity are highly variable parameters and almost always different from site to site. The average concentrations of the decant concentrations calculated in the statistical model is only a theoretical estimate based on available data.

- The geochemical model assumed that the equilibrium groundwater hydrochemistry, pre-mining, as well as mining processes between sites are statistically similar based on the statistical model.

- It was assumed that the initial mine water composition is representative based on the statistical analysis.

- The geochemical model assumed an equilibrium model. (Assumed full reaction of all reagents and stoichiometrically balanced reaction). The geochemical model used assumed that all the chemicals added will react fully.

- The geochemical model assumed that there will be no plating of the treatment product added to decant; and the precipitation of any mineral phase is uninhibited.

- The numerical transport model assumed a 5% decrease in the release rate of SO_4 in the pit on a cumulative annual basis 5% decay rate (Mack & Skousen, 2008).

- The inflow and outflow of the investigated backfilled opencast mine was estimated using a numerical model. Although this may not be perfectly accurate and only estimates based on available data, defensible results can be obtained based on measured monitoring data and model calibration tests.

- pH and SO_4 are independent variables in decant water but are both influenced by the amount of sulphide materials present in the backfilled opencast mine. Other influencing parameters could include carbonate mineral phases. However, due to the absence of alkalinity in the mine water samples, it was assumed that carbonates are not present or depleted. Therefore, pH and SO_4 in decant water remain independent due to the absence of sulphides and the fact that sulphate is not a pH dependant species.

The geochemical model will be completed by following the discussed criteria below.

3.10.2 Geochemical Statistical Analysis

The information required to predict the future water chemistry and the potential treatment thereof will be determined by a non-parametric, multivariate statistical model of the hydrochemistry of a population group (samples from Mine A, Mine B and Mine C) and compared to the modelled hydrochemistry of the site (Mine X). The method followed is documented in Sections 3.10.2.1 to 3.10.2.4.

3.10.2.1   Determine the relationship between pH and SO_4 from measured data.

To determine the statistical relationship between pH and SO_4, a total of 1790 samples will be analysed. This statistical analysis will help identify "hidden" relations to determine if there is any correlations between pH and SO_4 and if so to what variation.

To determine this statistical relationship between pH and SO_4, pH will have to be converted to a hydronium ion (H^+) to compare concentrations on the same scale. pH is a log scale while H^+ is a linear scale equivalent to the SO_4 measurement. The following formula will be used for the conversion:

$$pH = 10^{-x}$$

Once pH is converted to a concentration it can be correlated to its corresponding SO_4 concentration using a scatter plot. To determine the correlation fit a linear trendline, assuming a fit (R2) greater than 0.7 is significant.

A mathematical expression of the relationship between the two parameters will then be represented by a straight line formula. The straight trendline equation will be used to obtain calculated H^+ concentrations based on measured SO_4 concentrations. pH will then be re-calculated based on the mathematical expression and compared to the measured pH to determine the variation from measured values.

3.10.2.2   Determine the relationship between pH and the volume of pre-treatment product

The straight line equation will be used to calculate H^+ concentration relative to SO_4 concentration from the numerical transport model. Once the H^+ concentrations are calculated, it can be convert and compared to the modelled pH. For a more accurate estimation, the mean variance for modelled data will be added to obtain a more accurate correspondence to measured data.

3.10.2.3   Determine the dynamic volumes of treatment product to be added.

To estimate the volume of pre-treatment product needed to treat pH, a pH "jump" will be calculated. The weight of lime needed for each pH jump can then be calculated using Geochemist's Workbench.

This will be calculated by determining the total volume of product required to raise pH from initial concentration (discharge quality at the start of mine decant) to required concentration. This will be achieved by examine various pH values relative to reaction progress, assuming 100% reaction of pre-treatment product. The fraction reacted vs. pH raise will then be graphically displayed on a scatterplot to determine the relationship between pH change and product mass. A trendline (polynomial) will then be fitted to the data to mathematically display the relationship and express the product mass vs. pH raise using trendline equation. This trendline equation will be used to calculate the required product for various calculated pH values based on model SO_4 concentrations

3.10.2.4   Determine the dynamic pre-treatment cost.

The calculated product volumes to be added per volume of decant (mg/L) will be converted to tonnes per year. This dynamic change in pre-treatment product will arise in a dynamic costing, changing every year.

3.11   Cost analysis

To estimate the cost of the pre-treatment product for the treatment of decant mine water, the following steps will be followed.

Coceptual Model / Site characteristics	• Site Layout, operations, topography, neighbouring properties.
Statistical Model	• Dendrogram to identifying the relationship of hydrochemical concentrations and correlation thereof from four differnet mines. • Prove that the decant chemistry of Mine X is likely to have a similar hydrochemistry.
Numerical Model	• Water balance. • Decant volumes. • SO_4 concentration.
Geochemical Model	• Simulate possible treatment chemicals and the amount of each chemical to reduce pH, remove metals and to desalinate the decant water.
Possible Treatment Methods	• Once the chemicals are identified, the most effect and the cheapest will be compare to each other.

3.11.1 Summary of the cost of applicable pre-treatment additives.

A summary of the costs applicable to the selected pre-treatment products will be tabulated.

4. DESCRIPTION OF THE STUDY SITE

4.1 Site Location

The proposed mine will be developed on the extension purchased towards the south west of the current mine location, located on farm lands near Delmas in the Mpumalanga province, South Africa.

4.2 Topography

The topography (Figure 11) can normally be used as a good first approximation of the hydraulic gradient in the unconfined to semi-confined aquifer. This discussion will focus on the slope and direction of fall of the area under investigation, features that are important from a groundwater point of view.

The area is characterised by a gentle undulating topography and in the area of the mining site the slope is more or less in the order magnitude of 1:82 (0.01).

Local drainage is towards the pan which is situated in the middle of the proposed mining area. The pan is not regarded as a sensitive area. On a larger scale, drainage occurs eastward towards the generalised flow of the Wilge River which is from south west to north east (6.5 km from the site).
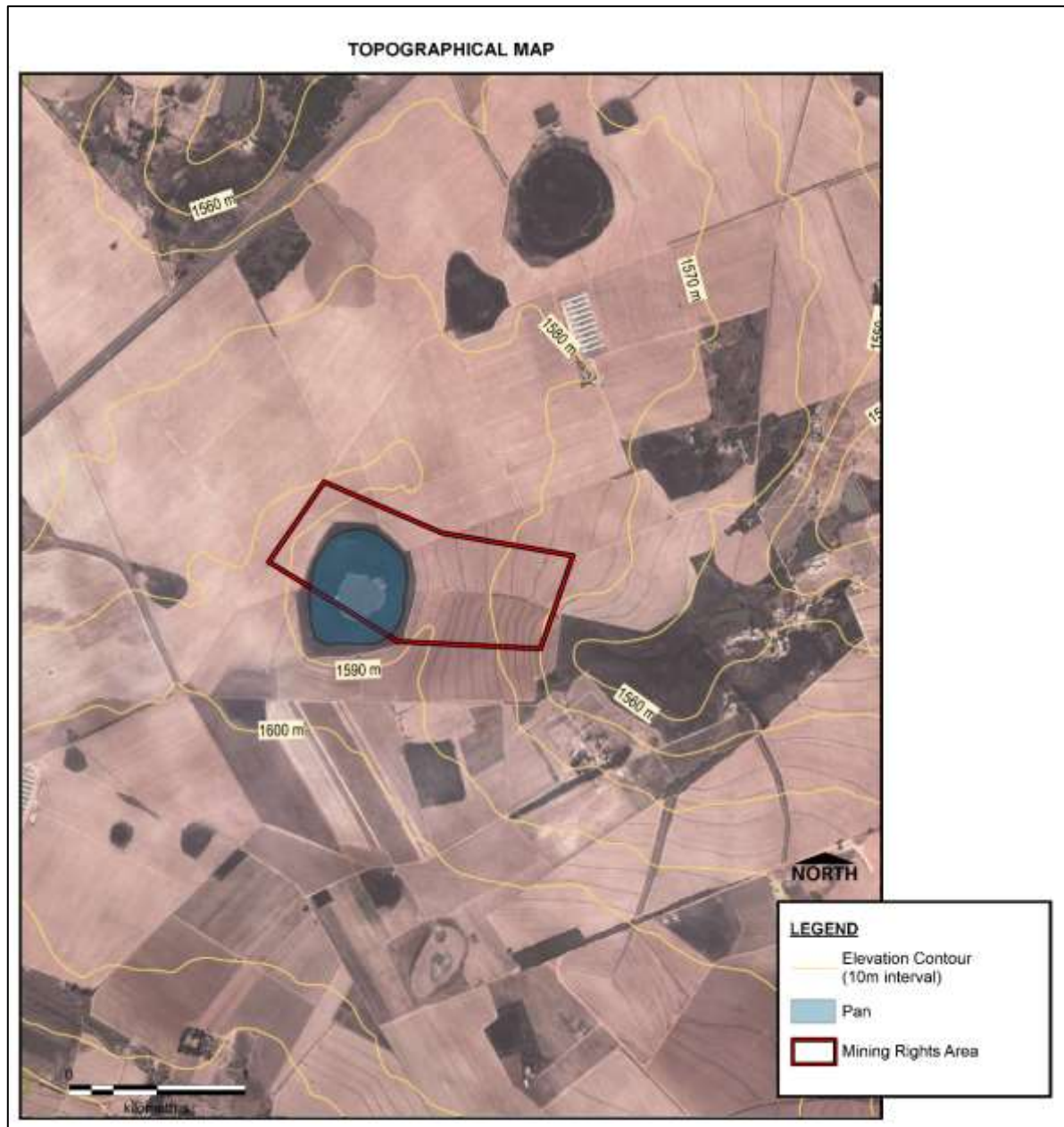
Figure 11: Topography map of Mine X

4.3 Surface Drainage

The proposed mine is situated in the Upper Olifants sub-catchment of the Olifants Water Management Area (WMA), in quaternary catchment B20E (Figure 12). The Upper Olifants Sub Area is divided into twenty six quaternary catchments (B11A – B11H and B11J-B11L, B12A- B12E, B20A – B20H and B20J and B32A). The Olifants WMA area is divided into four sub-catchments that is the Upper Olifants, Lower Olifants, Middle Olifants and the Steelpoort Sub Catchment Areas (DWS, 2017).
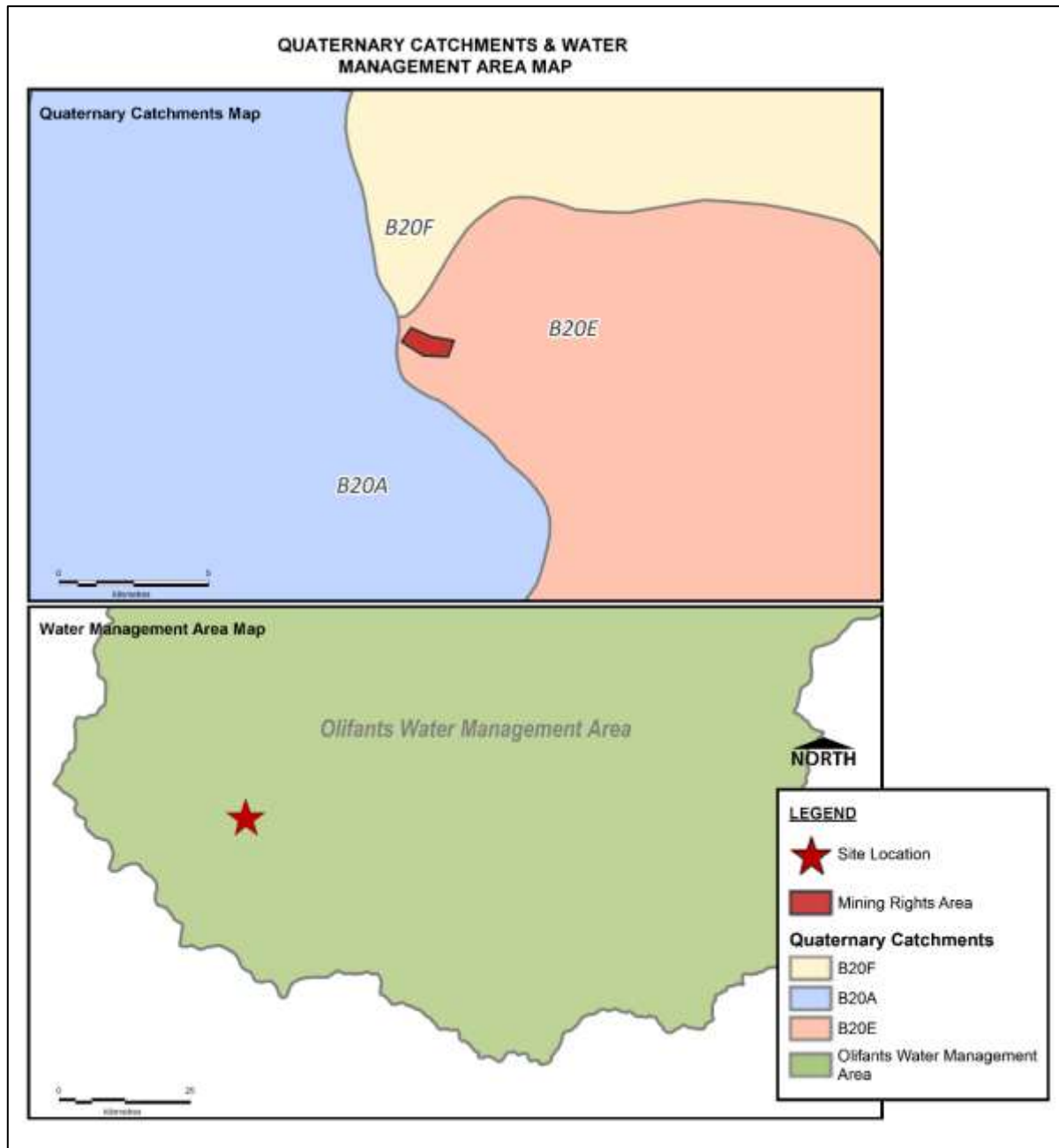
Figure 12: Quaternary Catchment Boundaries and Water Management are of the Mine X.

4.4 Geology

According to the 1:250 000 Geological map series of the Republic of South Africa, the investigated area falls within the 2628 East Rand geology series maps. An extract of these maps is shown in Figure 13.

The mine extension geological characterization consists of consolidated layers of the Karoo Supergroup, consisting of sandstone, shale, and coal beds of the Vryheid Formation of the Ecca Group, and is underlain by the Dwyka Formation.

The Vryheid Formation of the Ecca Group comprises mostly of sediments deposits in shallow marine and fluvio-deltaic environments. The accumulation of coal as peat in swamps and marshes are also associated with such environments. These sandstone and coal layers found on top of the shale, acts as reasonable aquifer systems while the shale serves as an aquitards.

The underlying Dwyka Formation consists of consolidated product of glaciation. These products usually have a high clay content components and are normally considered to be an aquiclude.

According to the 1:250 000 Geological map series of the Republic of South Africa, there are no distinct geological structures such as faults or dykes identified within the study area. However, no geophysical surveys were completed to confirm if there were no geological structures present within the study area.

The extent of the coal seams are generally controlled by the pre-Karoo topography (bedrock). The Karoo Supergroup sediments are horizontally deposited with gentle regional slope to the south
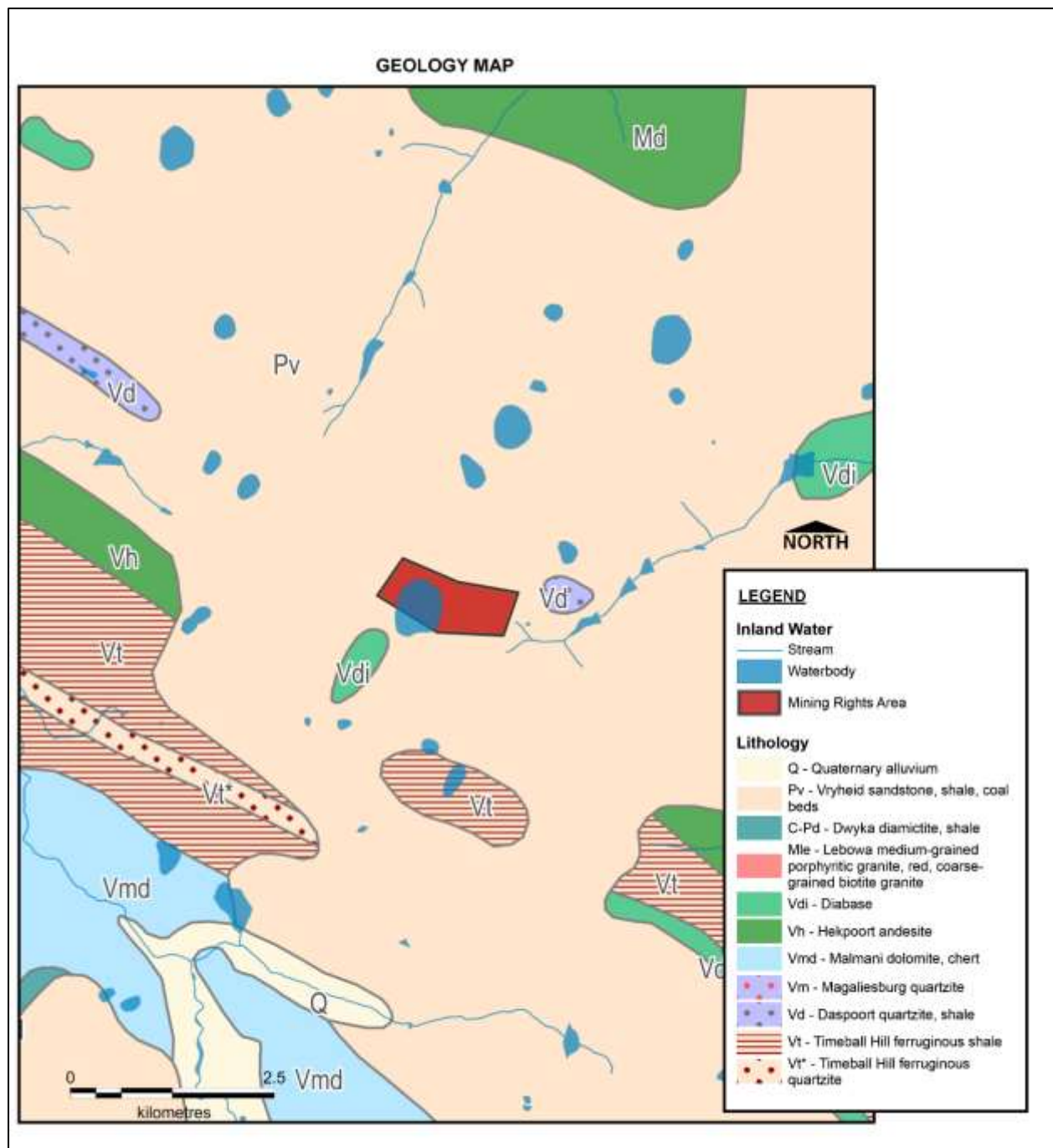
Figure 13: Geological Map of the Lithologies encountered at Mine X

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4.5 Hydrogeology

According to the 1:500 000 Hydrogeological map series (2526 Johannesburg) (DWS, 2017) (Freda & Schalk, n.d.) the area is underlain by an intergranular and fractured rock aquifer type with expected median borehole yields of 0.5 to 2.0 ℓ/s.

However, in these formations a multi layered weathering system could be present. This means that there could be two aquifer systems present in the form of a shallow regolith aquifer, and a fractured bedrock aquifer. The shallow regolith aquifer with a weathered intergranular soft rock base is associated with the contact of fresh bedrock and the weathering zone; whereas the fractured bedrock aquifers are associated with deeper formations. A brief description of these aquifer systems are discussed below.

4.5.1 Shallow Regolith Aquifer

Shallow regolith aquifers, are aquifers limited to the top zone of geological lithologies. The main source of recharge into the shallow aquifers is rainfall that infiltrates the aquifer through the unsaturated (vadose) zone. Vertical movement of water is faster than lateral movement in this system as water moves predominantly under the influence of gravity (Walker, 2016). The near surface weathered zone is composed of transported colluvium and in-situ weathered sediments and is underlain by consolidated sedimentary rocks. Groundwater flow patterns usually follow the topography, often coming very close to surface in topographic lows (springs) (Botha, et al., 1998). Recharge to these shallow aquifers are approximately 5% of the Mean Annual Precipitation (MAP).

4.5.2 Fractured Bedrock Aquifer

The host geology of the site area contains consolidated sediments of the Karoo Supergroup and consists mainly of sandstone, shale and coal beds of the Ecca Group. Groundwater movement is largely associated with secondary structures in this aquifer (fractures, faults, dykes).

Both the porosity and the hydraulic conductivity of the Ecca Group in fractured aquifers are known to be low. (Kruseman & de Ridder, 1994) states that porosity and permeability for the rock types, are 0 – 30% (porosity) and 10^{-7} – 1 m.d^{-1} (hydraulic conductivity).

4.6 Climatic Condition

Climatic data was obtained from the Department of Water and Sanitation (DWS) weather station Groenfontein at Bronkhorstspruit Dam (rainfall data and evaporation data) for the Delmas area (Table 15). The opencast colliery extension is located in the summer rainfall region of Southern Africa with precipitation usually occurring in the form of convectional thunderstorms. The average annual rainfall (measured over a period of 47 years) is approximately 726.6 mm per annum as presented in Table 16, with the high rainfall months between Octobers and March (Figure 15).

Table 15: Climatic data for Mine X (DWS, 2017).

Month	Average monthly rainfall (mm)	Mean monthly evaporation
January	138.86	164.87
February	84.26	144.22
March	93.58	135.340
April	43.09	104.07
May	14.43	84.98
June	7.014	65.40
July	2.54	73.65
August	7.54	103.45
September	22.53	141.57
October	76.76	165.93
November	112.30	163.75
December	123.63	175.64
Annual	**726.56**	**1522.91**
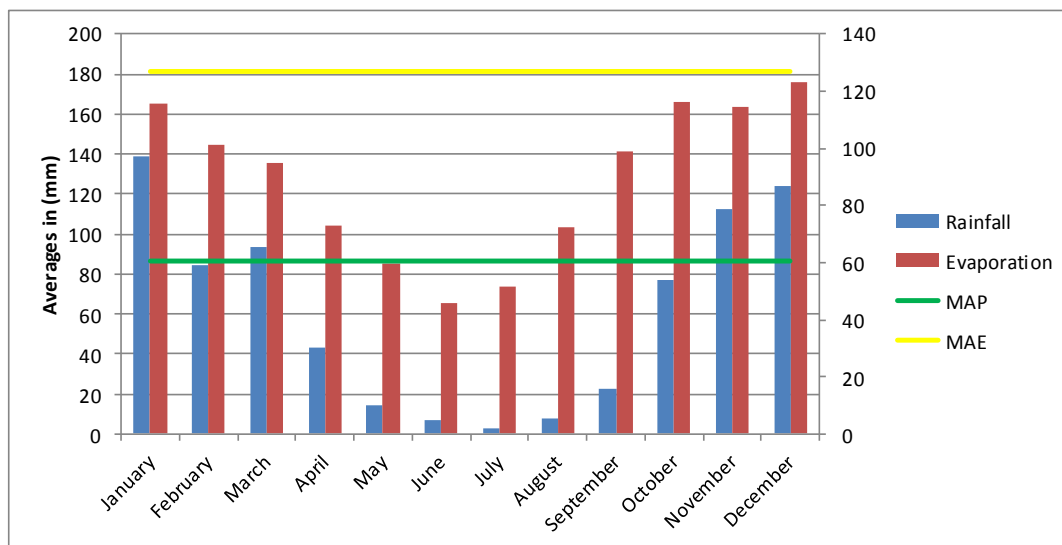
Figure 14: Climatic data representation (DWS, 2017).

4.7 Mining

The Colliery currently extracts coal by opencast mining methods (truck and shovel). Mining commenced in January 2013 and consists of mining three coal seems, namely the upper, middle and lower seams. The mine is currently fully operational.

The predicted expansion of the opencast colliery will extend approximately 80 hectares. By identifying potential contamination sources (opencast pit, overburden dumbs, stockpiles, tailings, pond dams, and facility processes) the impact to the groundwater and surrounding environment can be assessed and suitable mitigation measures can be put in place to limit or eliminate possible contamination.

4.7.1 Potential sources, receptors and potential impacts

Potential contamination sources for the proposed mine can be identified as the opencast mining, storage of fuel and waste rock dumps. Possible sensitive receptors can be identified via the desktop study and hydrocensus. Potential receptors could include (but not limited to) surrounding boreholes and an unnamed perennial river to the south of the opencast.

Following the period of mine closure, possible seepage of AMD will increase in concentration of sulphate in and around the area of the opencast pit. The concentrations will be evaluated and calculation regarding the potentially reach of the plume to receptors is necessary. A description of the size, location and composition of the expected activities is listed in Table 16.

Table 16: Surface Infrastructure and Mining details of the proposed Mine Extension

Surface Infrastructure	Amount	Size (m^2)	Expected Contamination (Hydro or Geochemical)
Hards Dump	1	12865 m^2	Possible Ca, Mg, Cl and SO4
PCD	1	4389 m^2	Possible Ca, Mg, Cl and SO4, Possible Acid Generation
ROM Stockpile	1	14819 m^2	Possible Ca, Mg, Cl and SO4
Softs Dump	1	30109 m^2	Possible Ca, Mg, Cl and SO4
Topsoil Dump	1	4928.8 m^2	Possible Ca, Mg, Cl and SO4
Mining Details	**Amount**	**Size (m^2)**	**Expected Contamination (Hydro or Geochemical)**
Opencast	1	854046 sq m	Possible Ca, Mg, Cl and SO4, Possible Acid Generation

The proposed mine layout is seen in Figure 15. Most surface infrastructure will be position towards the north and north-east of the proposed mine.
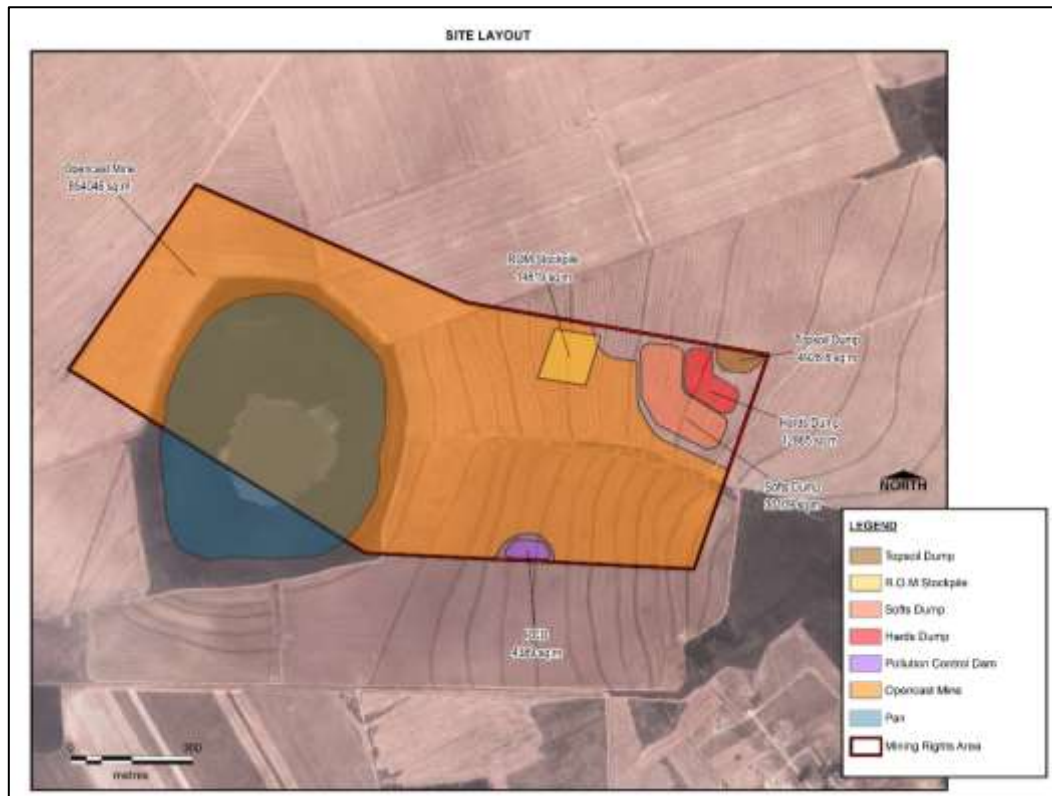
Figure 15: Planned mine Layout and Activity map.

4.7.2   Reverse Osmosis (RO) Treatment Plant

A RO plant is situated and is currently active at the fully functional mine. The plant is a multiphase reverse osmosis plant used for desalinization and metal removal. Like most RO plants, some type of pre-treatment is required. For this specific RO plant, minimum specification of a pH not lower than 5.2 was given. This means that feed water to the RO plant must be pre-treated to a pH of 5.2. Table 17 gives the RO plant specifications:

Table 17: RO Plant specifications

Plant Size (m^2)	±30
Maximum flow rate (ℓ/sec)	18
Current flow treated (ℓ/sec)	8
Recovery of treatment water (%)	Will treat 90 % of water
Minimum requirements	pH treatment to 5.2

5. RESULTS

5.1 Desktop Study

Before the commencement of the hydrocensus study, a desktop study was undertaken to identify and gather any known data of the area near and around the site. Information such as geological and hydrogeological data was evaluated and interpreted to gain a better understanding of the conditions pertained to the site area. A conceptual site model was developed to predict the hydrogeological conditions of the general region. The bulk of the desktop study was covered in Section 4 of this report labelled: Description of Study Site. All data discussed in Chapter 5 can be viewed in Appendix A: Data Dashboard.

5.2 Hydrocensus

The main objective of the hydrocensus was to familiarise the site and to obtain the necessary information to make informed proactive decisions regarding the management and protection of the groundwater resource in the area.

A hydrocensus was conducted on the 30 March 2017 within a 2 km radius of the site. This included the collection of groundwater related data from the study area and its surrounding environments (presence of boreholes, nearby springs, rivers, pans, water levels pumping rates etc.).

5.2.1   Boreholes

A total of ten boreholes were found during the hydrocensus. Of the ten boreholes identified, six are privately owned and used for various purposes such as domestic drinking water, irrigation of crops and for livestock. The remaining four boreholes are not in use. A summary of the groundwater information gathered during the hydrocensus is detailed in Table 18**Error! Reference source not found.**. The use of each borehole and position are depicted in Figure 16 and Figure 17.

Table 18: Hydrocensus Information

No.	ID	Latitude	Longitude	Elevation	Casing height	Static water level (mbgl)	Static water level (mamsl)	Sampled (Y/N)	Use				Comments
									Irrigation	Livestock	Domestic	Other	
Groundwater													
1	SBH6	-26.088278	28.799111	1578.112	0.000	8.01	1570.102	Yes		X	X		0
2	SBH8	-26.07525	28.793056	1566.492	0.000	8.59	1557.902	No					No sample, only water level
3	SBH9	-26.083444	28.796306	1572.475	0.017	-	-	No					Borehole closed, no access. Pump burned, not in use at the monent
4	SBH1	-26.093889	28.805722	1561.844	0.000	0.57	1561.274	Yes					Used to be a wind pump, borehole not in use anymore.
5	SBH10	-26.093083	28.777083	1580.524	0.030	-	-	Yes		X			Handpump, could not measure water level, only
6	SBH2	-26.091417	28.811722	1551.903	0.030	12.98	1538.923	Yes					Pump stolen, the farmer uses another two boreholes
7	SBH4	-26.102167	28.808111	1554.703	0.012	25.25	1529.453	Yes		X	X		Used for drinking and cooking at the Delmas Environmental
8	SBH5	-26.106222	28.797722	1570.883	0.000	-	-	Yes			X		Owner not at home, did not find the borehole, only took
9	SBH3	-26.098222	28.814472	1548	0.000	0	1548	Yes		X			Owner not at home at time of sampling, did not speak to owner
10	SBH13	-26.110139	28.760833	1588.239	0.000	3.59	1584.649	Yes			X		0
Surface water													
1	Pan	-26.097597	28.788045	1599.137		NA		Yes		NA			None
Newly drilled boreholes													
1	PBH1A	-26.101261	28.788834		0.950	1.82	-1.82	Yes				X	None
2	PBH2A	-26.09867	28.783038		0.930	1.31	-	Yes				X	None
3	PBH3A	-26.095217	28.783278		0.950	5.07	-	Yes				X	None
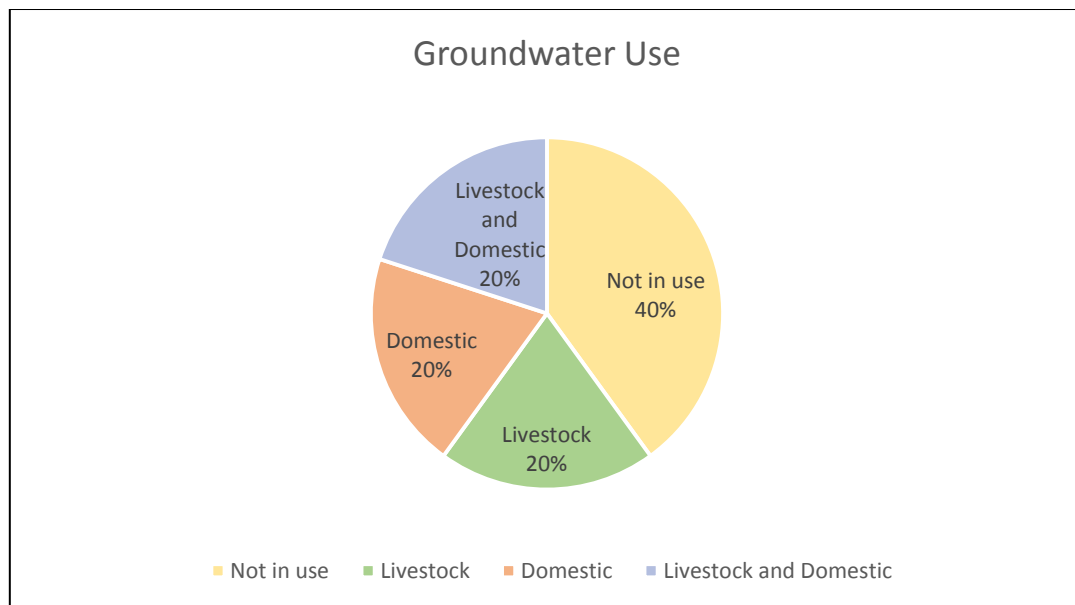
Figure 16: Groundwater distribution % use of the boreholes found during the hydrocensus.
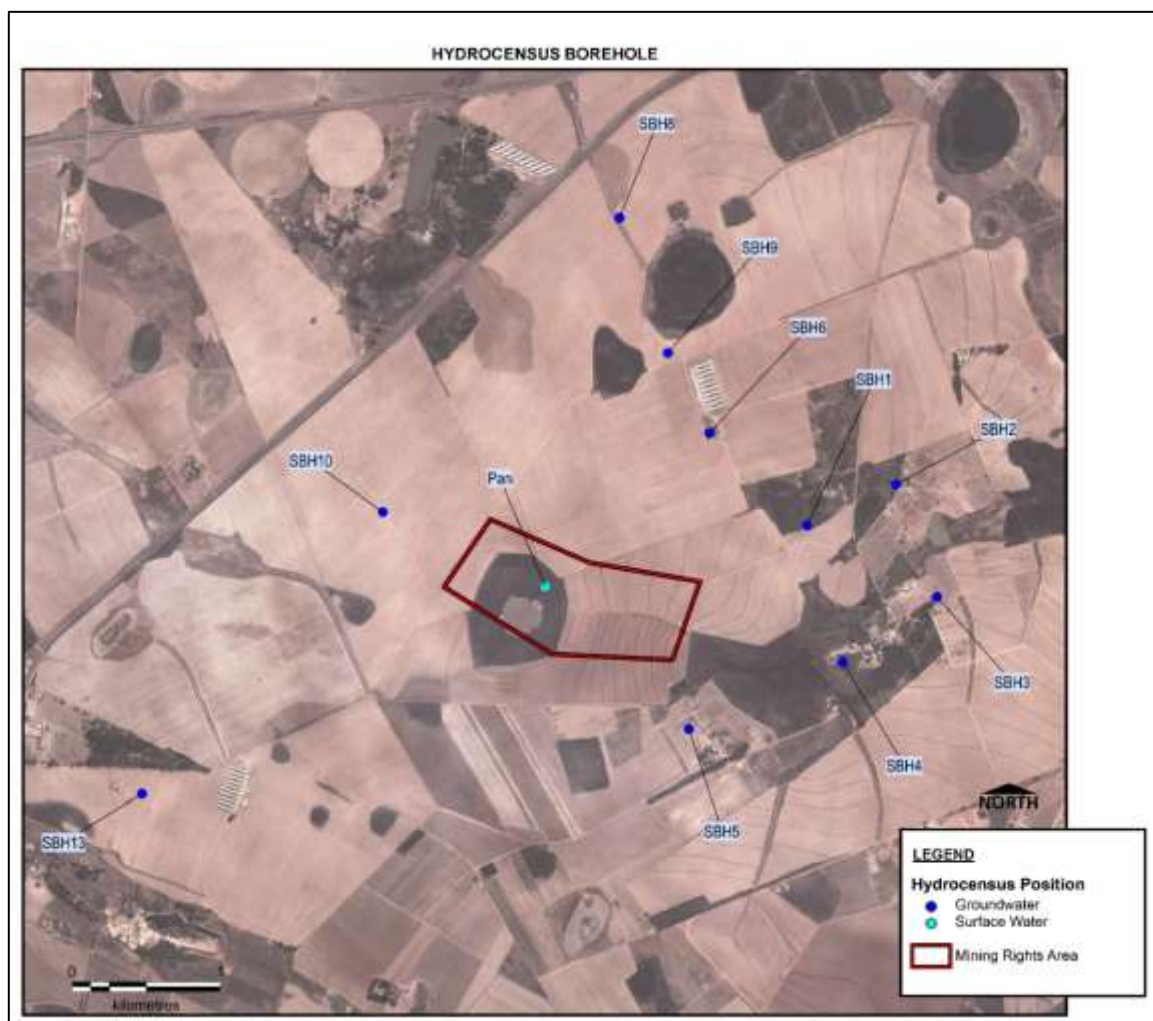
Figure 17: Location of hydrocensus boreholes.

5.2.2 Surface Water

One surface water features was found during the hydrocensus, namely a perennial pan situated in the centre of the proposed mine extension. In general the surface water flow follows the topography to the north-east of the site.

5.3 Drilling

Three (3) air percussion boreholes were drilled at strategic intervals around the pan (Figure 18) to depths of 30 m, 25 m, and 30 mbgl respectively. The borehole logs are presented in Figure 19. The boreholes can be described as follows:

- The lithology encountered is typical of Karoo sediments
- Borehole PBH1 encountered dolerite at 25 mbgl. This borehole also had a water strike at 26 mbgl.
- Boreholes PBH2 and PBH3 did not encounter any water strikes.
- PBH2 encountered weathered altered dolomite (WAD) at approximately 15 mbgl.
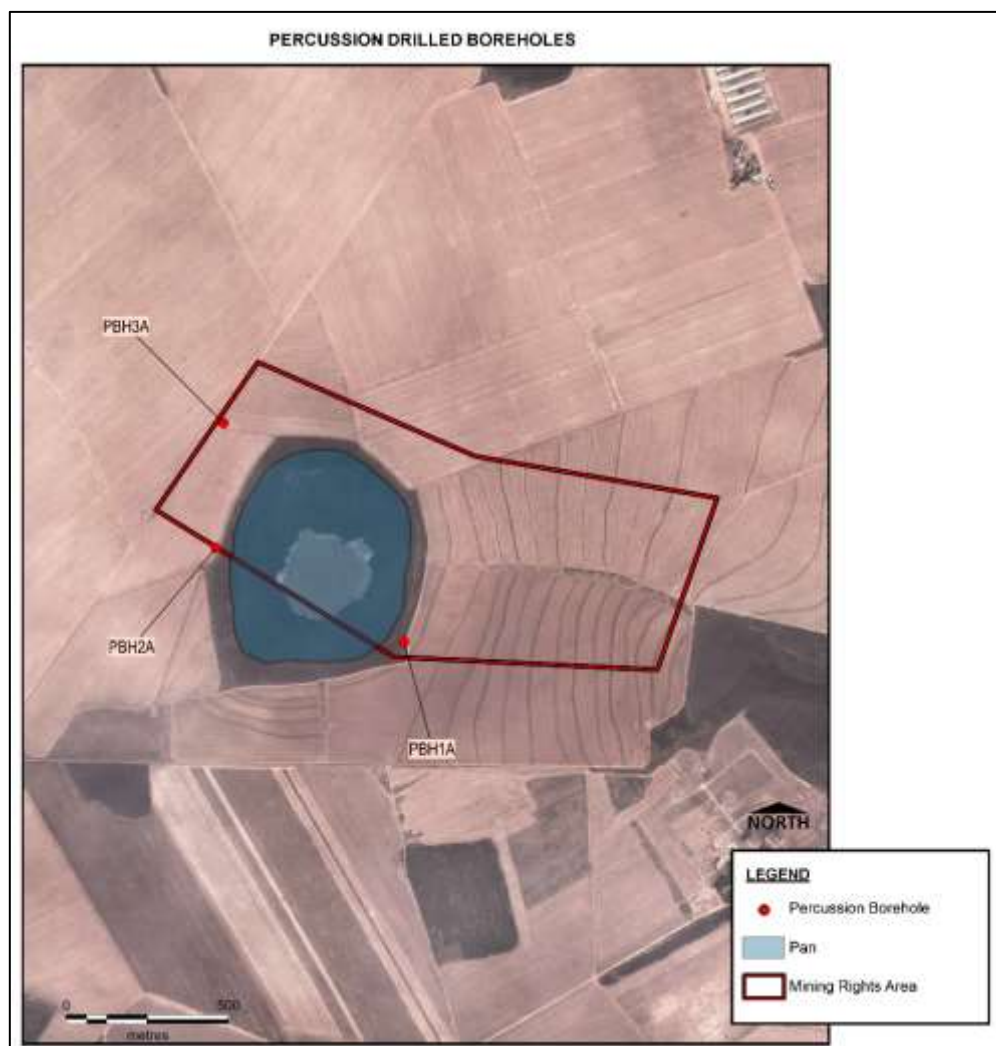
Figure 18: Percussion drilled boreholes positions
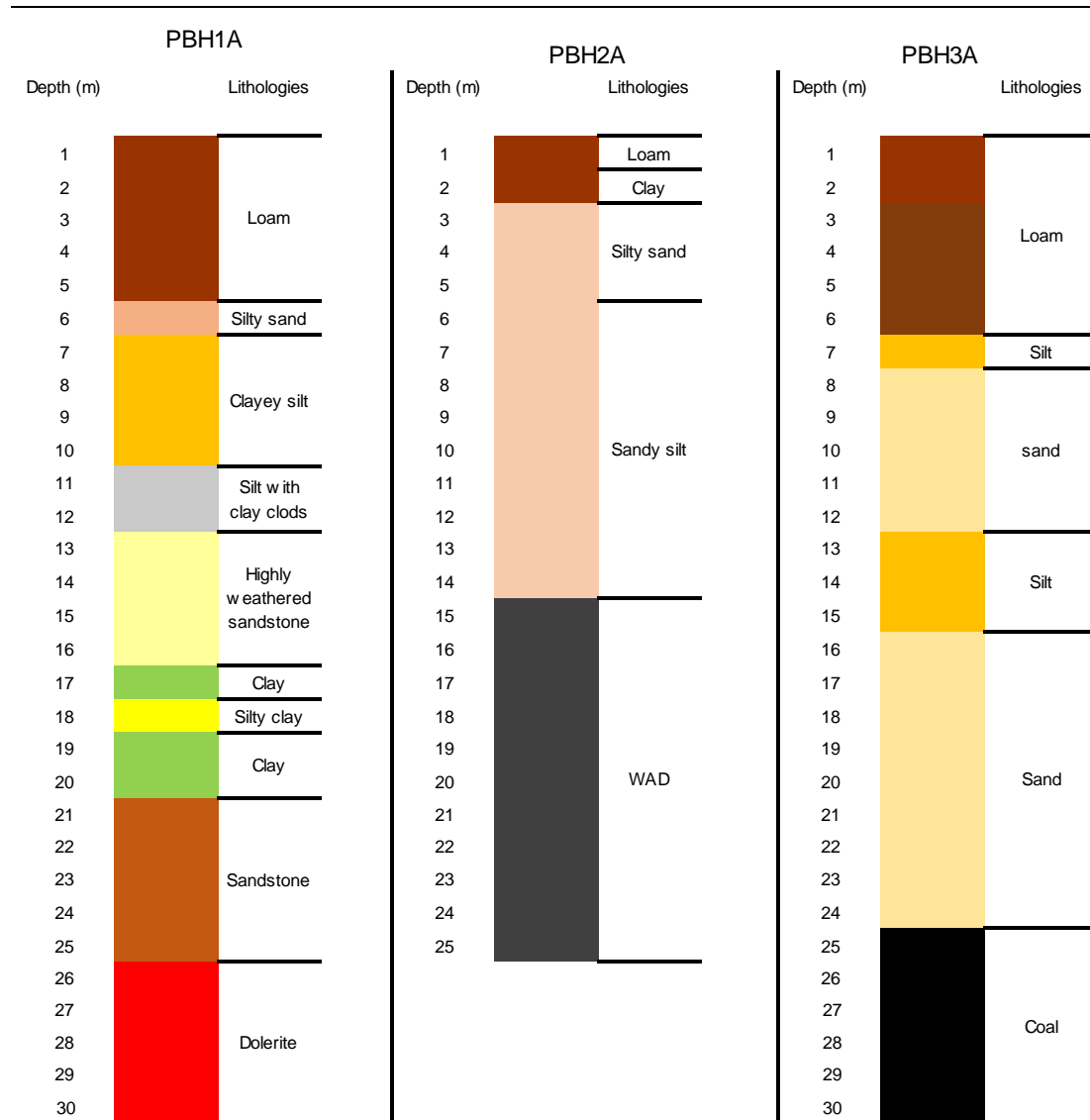
Figure 19: Stratigraphic columns for Mine X (drilling logs).

5.4 Aquifer Parameter Testing

5.4.1 Slug Tests

A slug test was conducted on one of the newly drilled percussion holes (PBH2A) to determine local aquifer parameters.

The borehole was subjected to a bailout test by removing approximately 15 litres of water from the borehole in a 1 min period. The borehole depth was measured at 25 mbgl with a static water level of 1.31 mbgl. The maximum negative displacement (lowering of the water level) was measured at 1.92 m below the static water level.

Using the AqteSOLV programme, the results of the slug tests are tabulated in Table 19 and Figure 20. The slug test calculated an average hydraulic conductivity (K) of 0.0142m/d.

Table 19: Slug test results summary

Name	Coordinates		Date	Water level (mbgl)	Maximum drawdown (m)	Transmissivity (m²/day)	Conductivity (m/d)
PBH2A	28.783038 E	26.09867 S	06/06/2017	1.31	1.92	0.348	0.0147
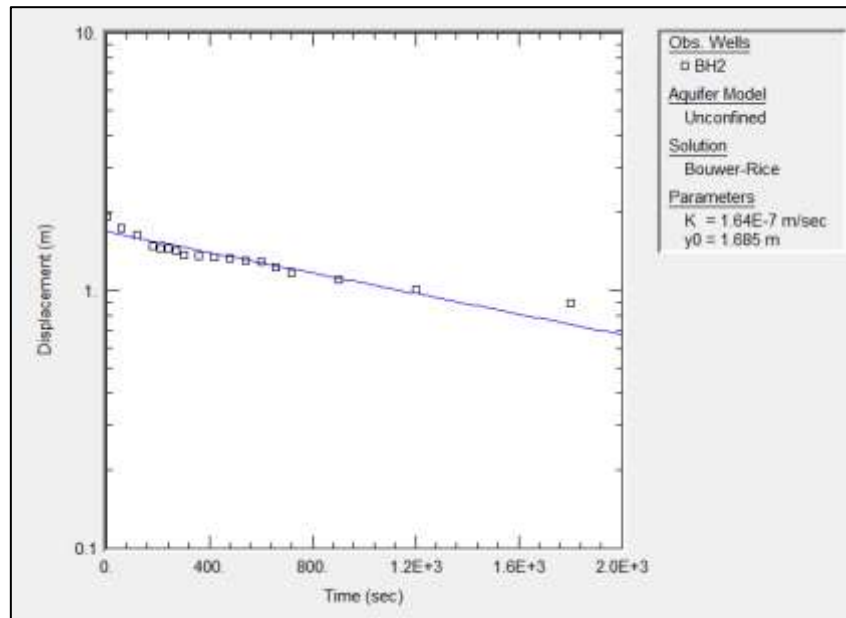
Figure 20: Slug test line fit (AqteSOLV).

5.4.2 Pump Test

Using the AqteSOLV programme, the results of the pumping test are summarised in Table 20 and Figure 21. The slug test calculated an average hydraulic conductivity (K) of 0.0226m/d.

Table 20: Pump Test result summary

Name	Coordinates		Date	Water level (mbgl)	Maximum drawdown (m)	Transmissivity (m²/day)	Conductivity (m/d)
PBH3A	28.783278 E	26.095217 S	06/06/2017	5.07	12.43	0.563	0.0226
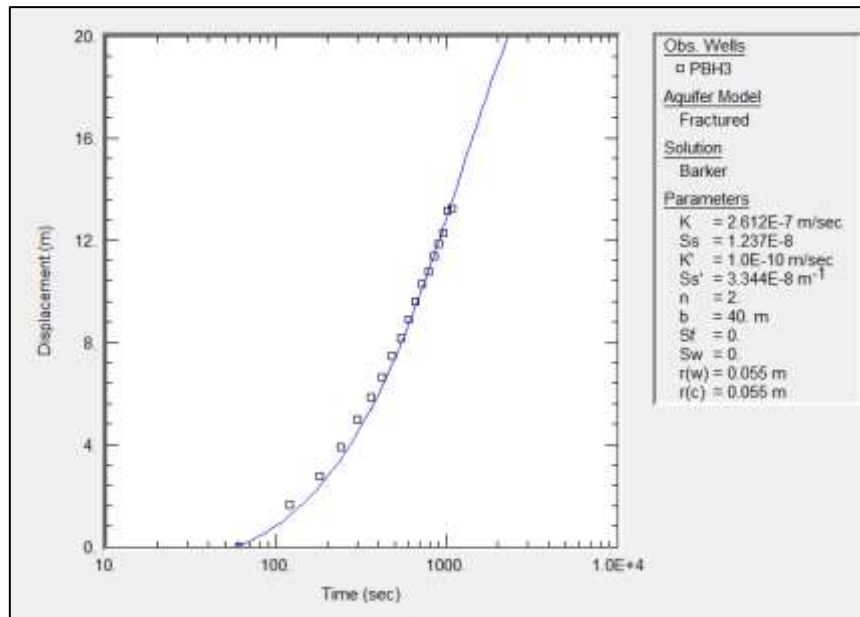
Figure 21: Pump test line fit (AqteSOLV).

5.4.3 Water Levels

During the hydrocensus, 10 boreholes were available for groundwater level measurement. The groundwater levels varied between a minimum of 0 m and a maximum of 25 mbgl. The water levels in the newly drilled boreholes were found to vary between a minimum of 1.31 m and a maximum of 5.07 mbgl. The relationship, using the boreholes from the hydrocensus, is represented in Figure 22.

This general relationship is useful to make a quick calculation of expected groundwater levels at selected elevations, or to calculate the depth of to the groundwater level (unsaturated zone):

Groundwater level   =   Elevation x gradient + intercept

Groundwater depth   =   Elevation – Calculated Groundwater Level

In unconfined and semi-confined aquifers, general a good relationship should hold between topography and static groundwater level. In this case the correlation is 95% which indicates that unconfined to semi-confined conditions exist in the vicinity of the study area. This relationship can be used to distinguish between boreholes with water levels at rest, and boreholes with anomalous groundwater levels due to disturbances such as pumping or local hydrogeological heterogeneities. Borehole SBH4 had such an anomaly due to pumping for domestic purposes.

Due to the heterogeneity of the subsurface, these relationships should not be expected to hold everywhere under all circumstances, and deviations should be expected. The calibrated static water levels as modelled have been contoured (Figure 23). Groundwater flow direction should

be perpendicular to these contours and inversely proportional to the distance between contours.

Table 21: Available groundwater level statistics (hydrocensus boreholes)

Groundwater level statistics	
Number of boreholes available	10
Number of boreholes with anomalous water levels	1
Min water level (mbgl)	0
Max water level (mbgl)	25.25
Mean water level (mbgl)	8.42
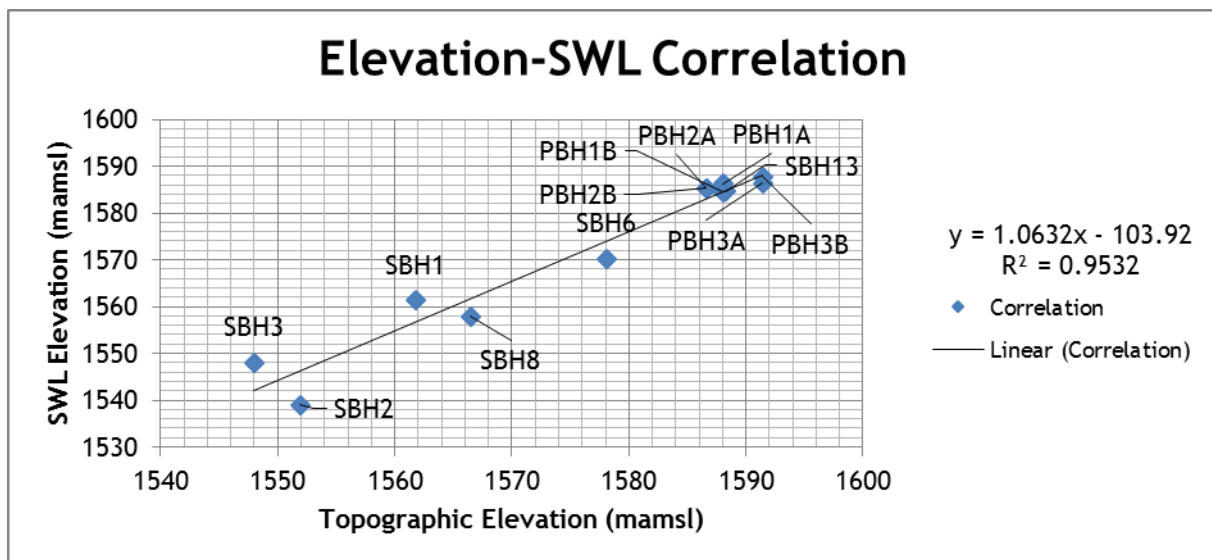
Figure 22: Correlation Graph of topography vs. available groundwater levels.

Figure 23: Contoured water levels of the water table aquifer using GMS.

5.5 Water Quality

A total of six water samples were collected. Five samples from five boreholes (SBH6, SBH10, SBH2, SBH4 and SBH5) and 1 surface water point (pan) around the study area during the hydrocensus. Three additional samples were collected from the newly installed boreholes (PBH1, PBH2 and PBH3). The results of the screening for groundwater and surface water are discussed below, and are presented in Table 22 and Table 23 .

5.5.1 Groundwater quality vs SANS standards

- Iron concentrations exceed the SANS aesthetic limit (lower limit) in SBH2, PBH1A, PBH2A and PBH3A which may affect the taste of water for human consumers.
- Manganese concentrations exceed the standard limit in SBH2, SBH10 and PBH2A.

5.5.2 Surface water quality vs SANS standards

- Iron and magnesium exceed the SANS standards in the Pan sample.

5.5.3 Spatial Analysis of Groundwater and Surface Water Quality

The pie, stiff and piper diagrams are represented in Figure 24, Figure 25, and Figure 26. These pie diagrams show both the individual ions present in a water sample and the total ion concentrations in meq/l or mg/l. The scale for the radius of the circle represents the total ion concentrations, while the subdivisions represent the individual ions. The following can be conducted from Table 22 and Table 23, and Figure 24, Figure 25, and Figure 26 that follow:

- The boreholes PBH1A, PBH3A, SBH2, SBH4 and SBH10 have higher proportions of HCO_3, Na, and Ca.

- The boreholes SBH5 and SBH6 have higher proportions of NO_3, Na and Ca and PBH2A has a high proportion of Na, Cl and SO_4.

- The pan sample has high proportions of Ca, Mg; and HCO_3 and SO_4. Due to the high evaporation rate, the pan sample is more mineralised than the groundwater samples.

The piper and stiff diagrams in Figure 24 and Figure 25 below indicate that the hydrochemistry at the site is dominated by Ca-Na-HCO_3 which is indicative of freshly recharged uncontaminated groundwater. However, samples PBH1A, PBH2B and SBH6 show some signs of SO_4 contamination.

Table 22:   Water qualities compared to SANS 241-1:2015 guidelines for human consumption (hydrocensus boreholes)

Parameter	Unit		SANS 241: 2015 Recommended Limits	Risk	Results					
					Pan	**SBH6**	**SBH10**	**SBH2**	**SBH4**	**SBH5**
Physical & Aesthetic determinants										
Electrical conductivity at 25^{0}C	EC	mS/m	≤ 170	Aesthetic	59.7	12.8	7.32	25.6	16.9	15.5
Total Dissolved Solids	TDS	mg/litre	≤ 1200	Aesthetic	418	89.9	51.2	179	119	109
pH at 25^{0}C		pH units	≥ 5 to ≤9.7	Aesthetic	7.47	6.35	7.13	7.47	7.06	6.65
Chemical Determinants - Macro determinants										
Nitrate as N	NO_3	mg/litre	≤ 11	Acute Health	<0.13	6.93	<0.13	<0.13	0.27	7.07
Nitrite as N	NO_2	mg/litre	≤ 9	Acute Health	<0.001	<0.001	0.004	0.001	0	0.058
Sulphate	SO_4	mg/litre	Acute Health ≤500; Aesthetic ≤250	Acute Health/Aesthetic	69.9	9.94	9.04	8.57	13.9	9.75
Fluoride	F	µg/litre	≤1500	Chronic Health	0.395	0.117	0.261	0.538	0.664	0.079
Ammonia as N	NH_3	mg/litre	≤ 1.5	Aesthetic	0.116	<0.01	0.363	0.282	<0.01	0.01
Chloride	Cl	mg/litre	≤ 300	Aesthetic	23.4	9.01	2	1.76	2.85	5.2
Sodium	Na	mg/litre	≤ 200	Aesthetic	87	13.2	6.68	24.6	26.3	10.5
Zinc	Zn	µg/litre	≤5000	Aesthetic	0.13	0.12	1.88	0.06	0.23	0.1
Chemical Determinants - Micro determinants										
Total Iron	Fe	mg/litre	Acute Health ≤ 2.0; Aesthetic ≤0.3	Acute/Aesthetic	0.36	0	0	0.48	0.05	0
Total manganese	Mn	mg/litre	Acute Health ≤0.4; Aesthetic ≤0.1	Acute/Aesthetic	0.14	0	0.3	0.56	0	0
Aluminium	Al	µg/litre	≤ 300	Operational	0	0	0	0	0	0
Concentration deemed to present an unacceptable health risk for lifetime consumption.										

Table 23: Water qualities compared to SANS 241-1:2015 guidelines for human consumption (new boreholes)

| Parameter		Unit	SANS 241: 2015 Recommended Limits	Risk	Results		
					PBH1A	PBH2A	PBH3A
Physical & Aesthetic determinants							
Electrical conductivity at 25⁰C	EC	mS/m	≤ 170	Aesthetic	23	17	6.16
Total Dissolved Solids	TDS	mg/liter	≤ 1200	Aesthetic	161	119	43.1
pH at 25⁰C		pH units	≥ 5 to ≤ 9.7	Aesthetic	7.84	6.58	6.24
Chemical Determinants - Macro determinants							
Nitrate as N	NO_3	mg/liter	≤ 11	Acute Health	1.47	1.45	1.73
Nitrite as N	NO_2	mg/liter	≤ 9	Acute Health	<0.001	<0.001	<0.001
Sulphate	SO_4	mg/liter	Acute Health ≤ 500; Aesthetic ≤ 250	Acute Health/Aesthetic	16.7	69.6	12.3
Fluoride	F	µg/liter	≤ 1500	Chronic Health	0.412	0.443	0.12
Ammonia as N	NH_3	mg/liter	≤ 1.5	Aesthetic	0.174	0.434	0.085
Chloride	Cl	mg/liter	≤ 300	Aesthetic	2.82	33.4	1.55
Sodium	Na	mg/liter	≤ 200	Aesthetic	26.5	28.8	8.47
Zinc	Zn	µg/liter	≤ 5000	Aesthetic	0	0.38	0.16
Chemical Determinants - Micro determinants							
Total Iron	Fe	mg/liter	Acute Health ≤ 2.0; Aesthetic ≤ 0.3	Acute/Aesthetic	0.55	7.57	1.49
Total manganese	Mn	mg/liter	Acute Health ≤ 0.4; Aesthetic ≤ 0.1	Acute/Aesthetic	0	0.24	0
Aluminium	Al	µg/liter	≤ 300	Operational	1.82	65.5	3.89
Concentration deemed to present an unacceptable health risk for lifetime consumption.							
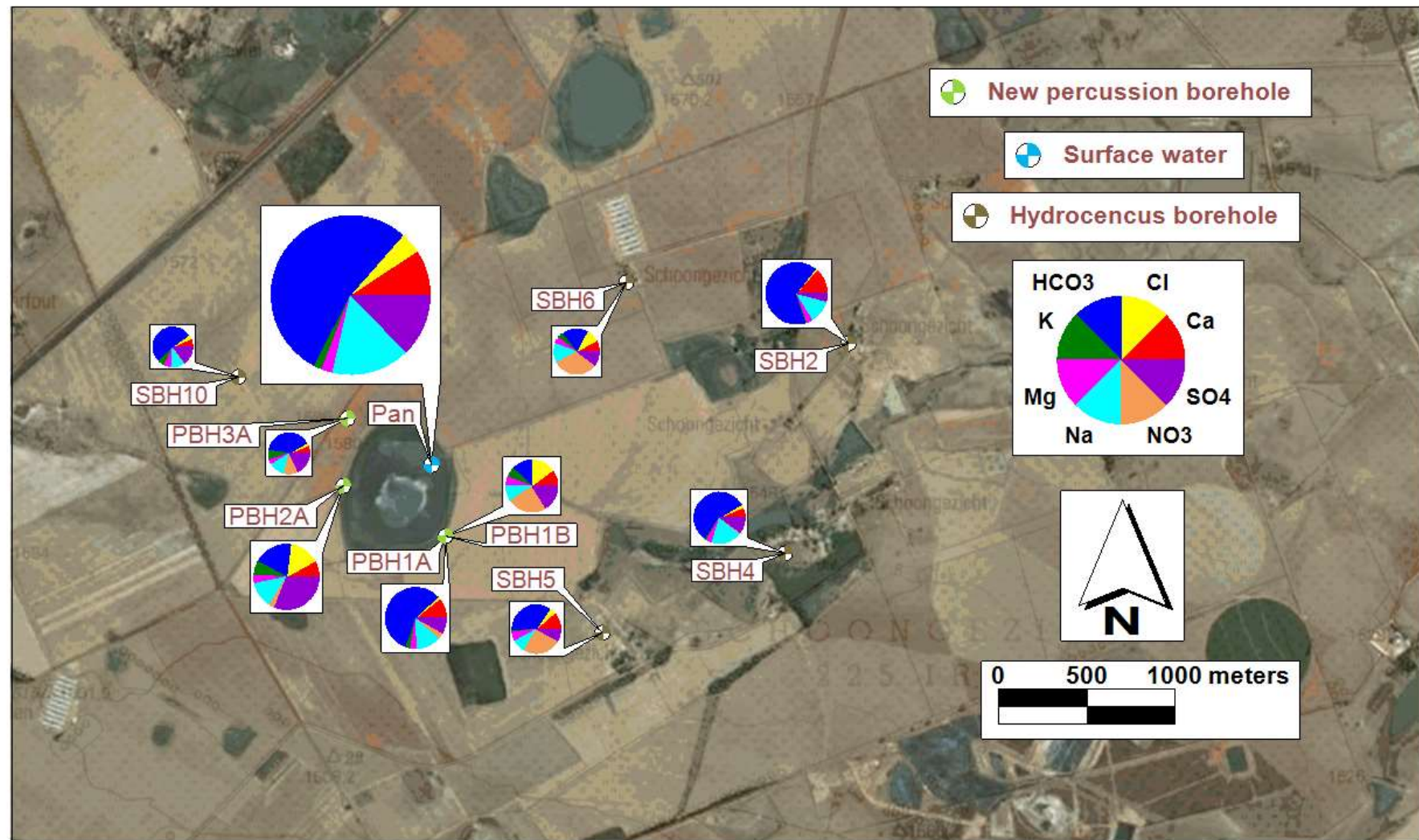
Figure 24:   Pie diagrams for groundwater and surface water samples
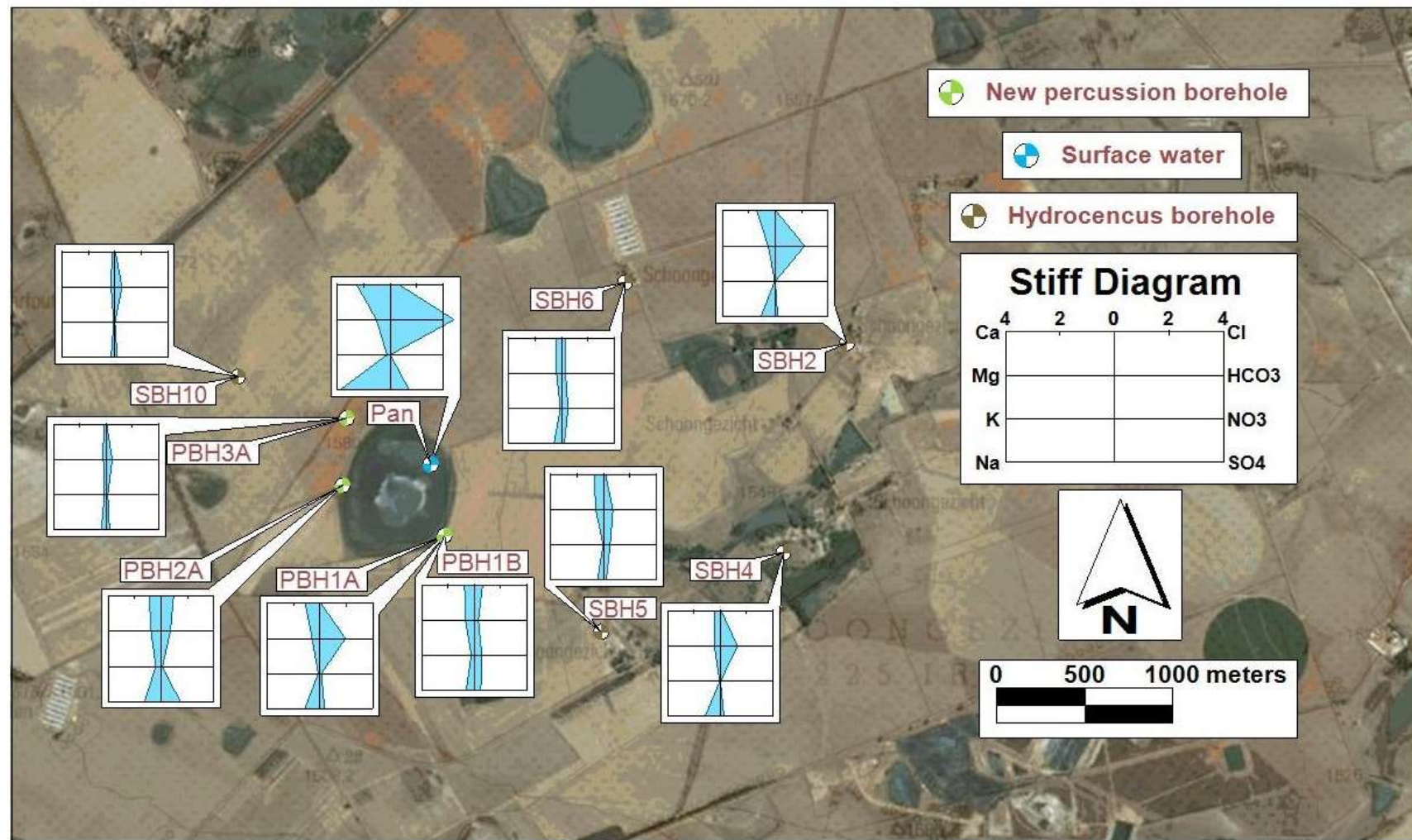
Figure 25: Stiff diagrams for groundwater and surface water samples
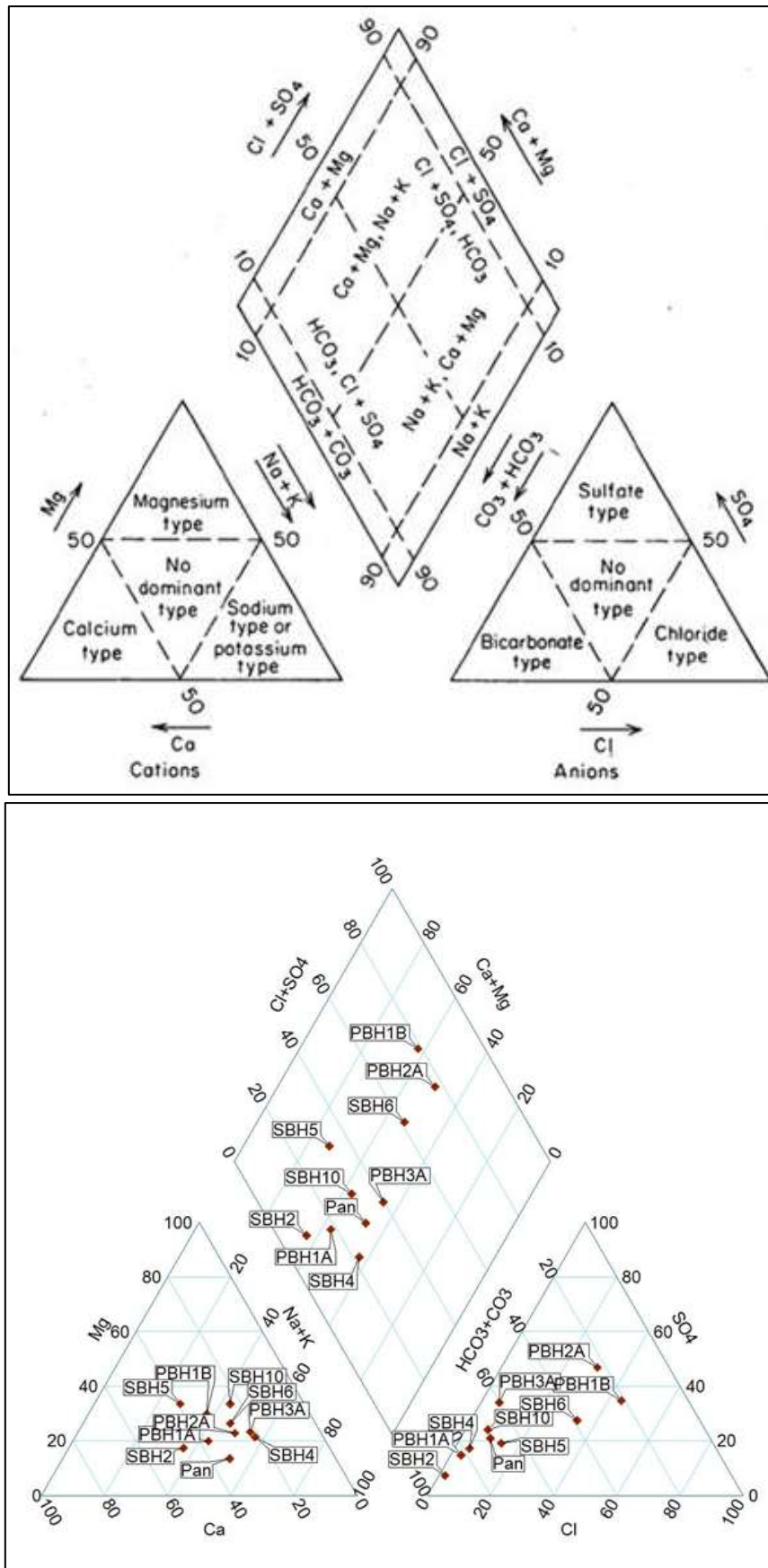
Figure 26:   ***Explanation of the hydro-chemical*** facies ***in the Piper Diagram***

5.6 Analytical Model

Analytical models are basic mathematic models used in hydrogeology to predict groundwater water flows. Analytical models are mathematical models that have a closed form solution (Marineli & Niccoli, 2000). A one-dimensional robust analytical model of groundwater flow, was used for estimating the amount of water influx into the opencast mine. The analytical model does not take into account the aquifer boundaries, nor for the spatial distribution of groundwater withdrawals from boreholes. The following assumptions were made:

- sustainable yield can be estimated without accounting for aquifer boundary conditions, aquifer geometry, and the spatial distribution of hydraulic properties of the system, and
- sustainable yield is a fundamental property of an aquifer independent of the locations of wells and rates of pumping from wells (Marineli & Niccoli, 2000).

Table 24 (Marineli & Niccoli Model) indicates an estimated 684 m^3/d flows into the opencast pit, using the parameters documented in the same table.

Table 24: Analytical Model Summary

Model Parameter		Value	Reason		
Initial saturated thickness (m)	h0	40	Estimated		
Saturate thickness above pit base (m)	hp	1	Estimated		
Recharge flux (m/d)	w	0.001	Calculated (5%)		
Horizontal hydraulic conductivity from pit centre (m)	Kh1	0.0226	Calculated		
Radius of influence from pit centre (m)	r0	684	Calsulated	**Total calculated discharge in to the Pit in m^3/d**	**683.89**
Effective pit radius (m)	rp	504	Calculated		
Horizontal hydraulic conductivity of zone 2 in m/d	Kh2	0.0015	Estimated		
Vertical hydraulic conductivity of zone 2 in m/d	Kv2	0.00015	Estimated		
Depth of water in the pit bottom (m)	d	0.01	Estimated		
Anisotropy of zone 2	m^2	10	Anderson et al. (2015)		

5.7 Numerical Model

Water level data obtained the hydrocensus was used to calibrate the steady state numerical groundwater flow model. The results obtained during the steady-state scenarios were used as initial conditions to simulate dewatering and contaminant transport impacts.

The optimal calibration aquifer parameters are presented in Table 25.

Table 25:   Optimal Calibrated Aquifer Parameters

Aquifer	Model layer	Layer thickness (m)	Porosity (%)	Hydraulic conductivity (m/d)
Soil Horizon	Layer 1	10	30	1.0
Shallow Weathered Aquifer	Layer 2	30	5	0.2
Weathered/Fractured Aquifer	Layer 3	30	4	0.01
Fractured Aquifer	Layer 4	30	3	0.001

During the operational phase, it is expected that the main impact on the groundwater environment will be the dewatering of the immediate surrounding aquifer. Water entering the mining pit will have to be pumped out of the void area to enable mining activities. This will cause a lowering in the surrounding groundwater table. To calculate the drop in water table, a worst-case scenario has been modelled, assuming that the entire opencast would be dewatered.

Base flow to the non-perennial tributary to the Wilge River is most likely to be affected by drawdown to the southeast of the mine. The computed total inflow into the mine, assuming that all areas in the mine are dewatered simultaneously, was calculated as 658 m^3/d (Table 27, Table 28, and Table 29). However, as the whole opencast will not be open at any one time, this is an overestimate. The calculated drawdowns for the worst case scenarios are depicted in Figure 27 and Figure 28. SBH10 of the identified hydrocensus points will be affected by a drawdown of about 6 m if the mine is entirely dewatered.

It must be cautioned that these calculations have been performed using simplified assumptions of homogeneous aquifer conditions, which may not represent a real world scenario (refer to assumptions made in Section 3.8.2: Assumptions and Limitation of Numerical Model of this report).

Table 26:   Calibration Statistics

	Model Parameter	Value	Unit	Reason
1	Recharge to the aquifer	0.0001	m/d	Calculated (5%)
2	Recharge to the backfilled opencast mine	0.0004	m/d	Hodgson and Krantz (1998) (20%)
3	Evapotranspiration	0.005	m/d	Calculated (DWAF-mm/year to m/d)
4	Boundaries	Topographic water divides	-	Existing boundary conditions present at the site that would potentially include modelled impacts
5	Refinement	20	m	Based on the scale of the mining area
6	Grid dimensions	130 x 140	Cell count	Product of the grid refinement
7	Hydraulic conductivity	0.02	m/d	Existing hydrogeological report (Du Toit, 2010)(Botha/ Kruseman)
8	Hydraulic anisotropy (vertical)	10	-	Anderson et al. (2015)
9	Effective porosity	(30% for regolith) 5 declining to 3 with depth in each layer	%	(Wang, et al., 2009) (Jiang, et al., 2009)
10	Layers	4	Count	Mining depth is 40m Wang et al. (2009)(Jiang)
11	Longitudinal dispersion	50	m	(Schulze-Makuch, 2005)
12	Mean residual head error	1.97	m	Head error statistics
13	Head error range	10	m	Calculated as 10 % of the difference between the maximum and minimum calculated head elevations

Table 27:   Summary of potential impacts during operation – dewatering

Mining Area	Area (m²)	Mining Seam	Maximum Drawdown (m)	Cone of depression from edge of pit (m)	Estimated Inflow for the Total Area (m³/day)	Evaporation (m³/day)	Potential Impacted Receptor	Expected Water Level Decline (m)
Opencast Mine during operations	851120	S2 Lower	30	880	658	3915	The Wilge River and SBH10	5-10

Table 28:   Summary of potential impacts post operations

Mining Area	Area (m²)	Potential impacted receptor	Rebound time (Years)	Potential decant (Yes/No)	Drain (decant volume) ℓ/s	Potential decant area
Backfilled Opencast Mine Post mining	851120	The tributary of the Wilge River	11	Yes	6 ℓ/s	To the south-east of the opencast

Table 29: Water Budget (Pit in vs out flows)

Source/ Sinks	Constant Head	Drains	ET	Recharge	Total Source/ Sink	Total Zone Flow	Total Flow
Flow in	657.4016965	0	0	4073.610948	4731.012584	0	4731.012584
Flow out	-300.8813282		0	0	-4731.031897	0	-4731.031897

5.7.1   Before Backfill
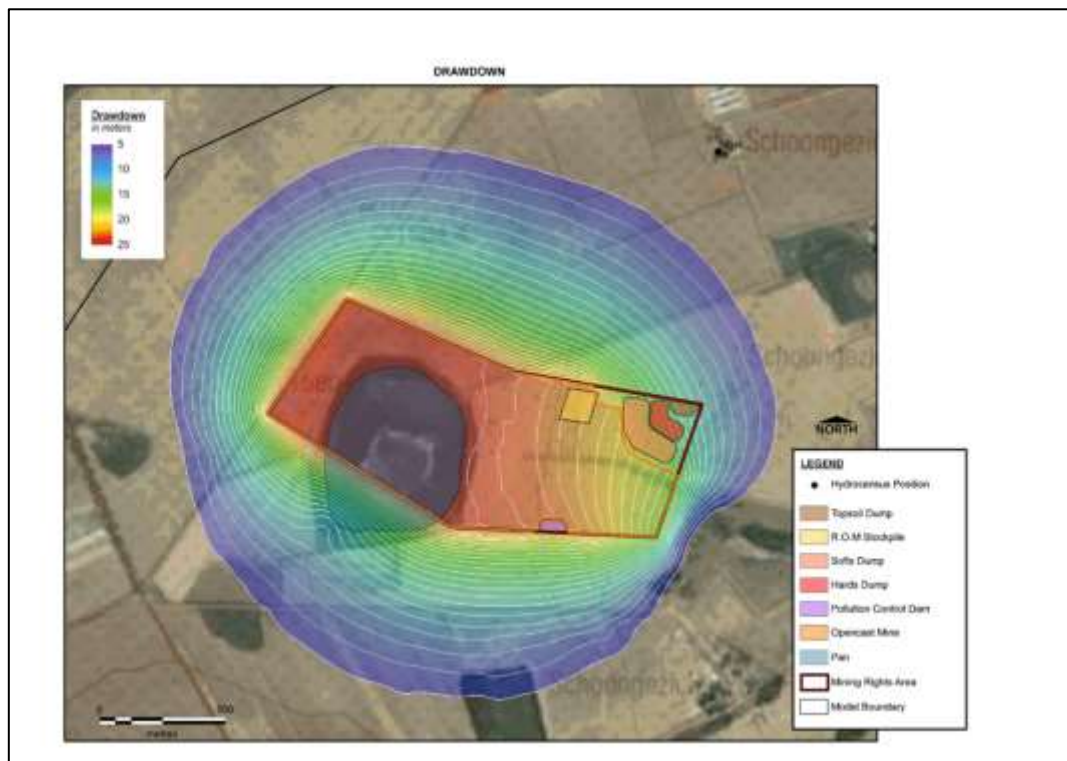
Figure 27: Drawdown before the mine is backfilled using a steady state simulation for the "during" mining scenario. It was assumed that the entire pit will be dewatered simultaneously to represent a worst case scenario and achieve the maximum drawdown for the simulation
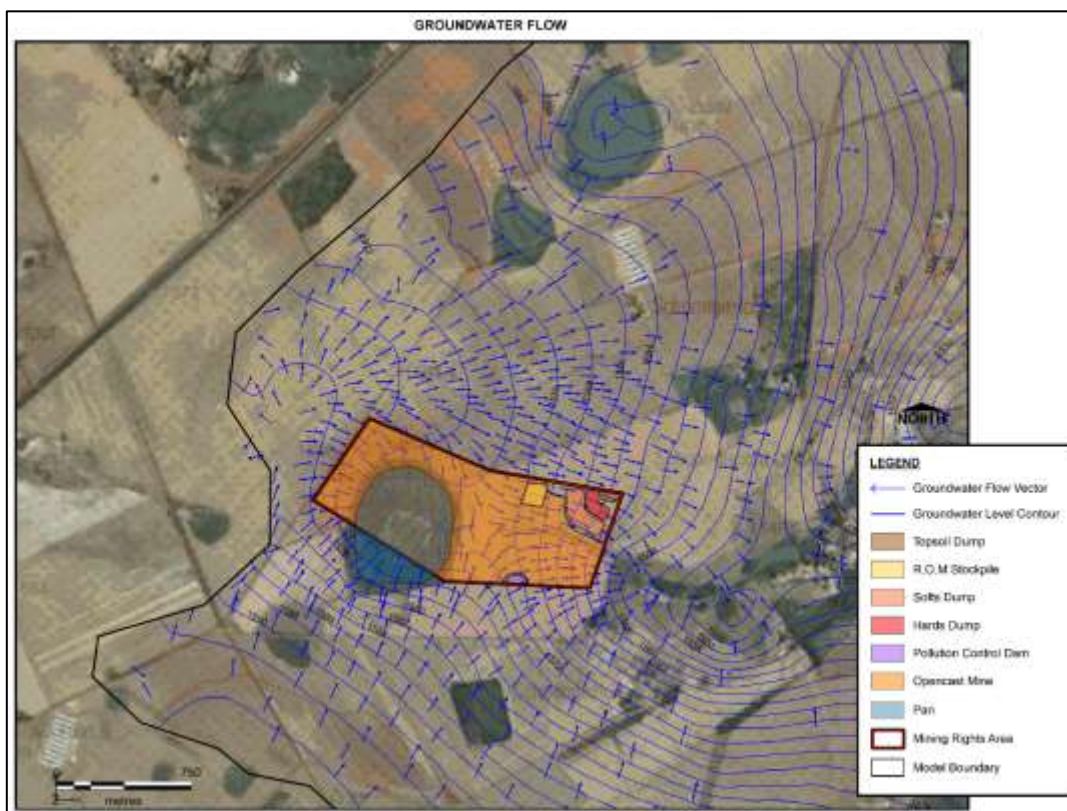
Figure 28: Flow vectors

5.7.2 After Backfill and Decommission

After backfill and decommissioning of the opencast pit, it is assumed that the dewatering process will be discontinued. This will allow flooding of the backfilled pit area. The groundwater regime will then return to a state of equilibrium. Figure 29 depicts the conceptual site model for pre-mining vs. post mining.
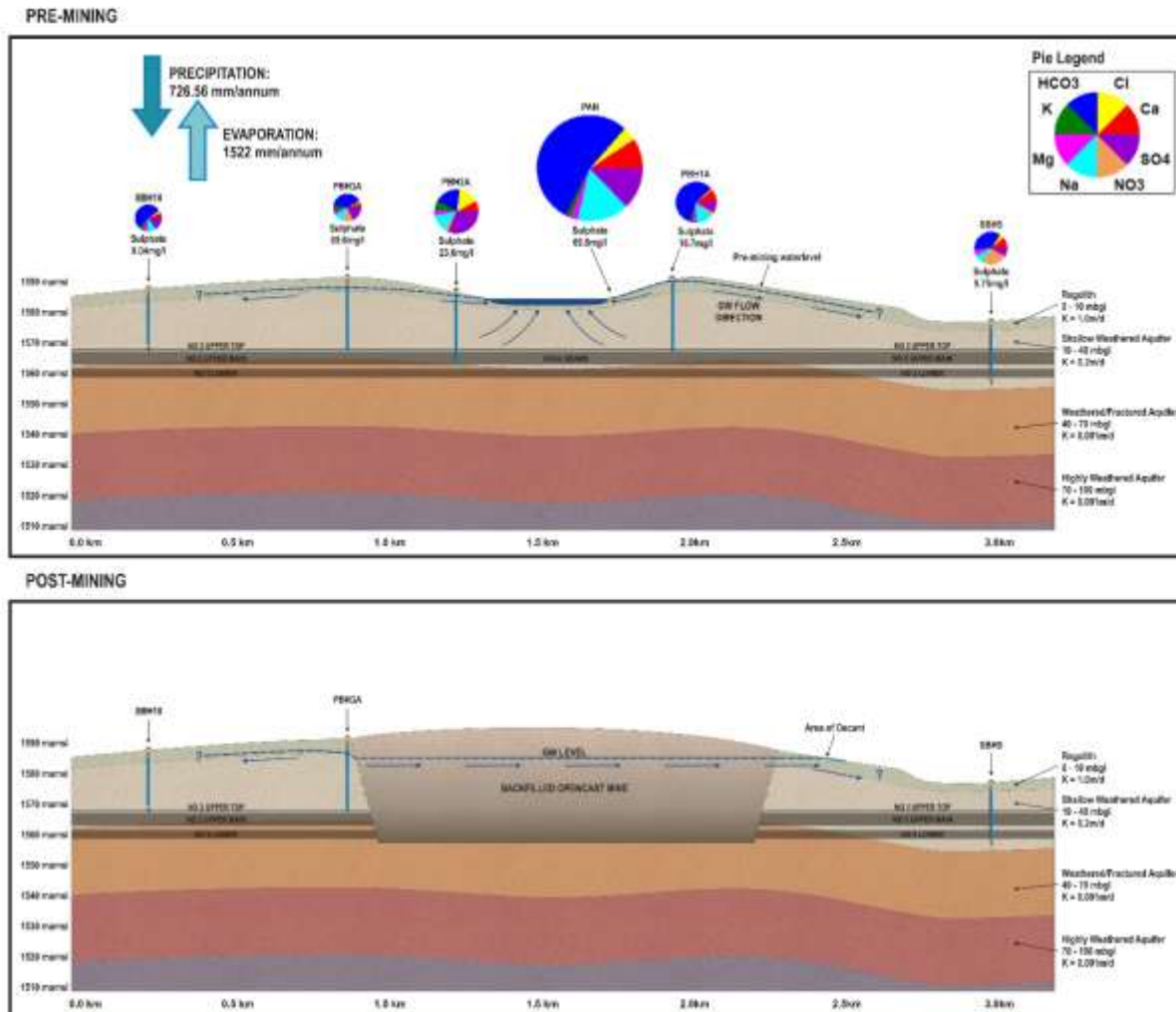
Figure 29: Conceptual site model

The rise in groundwater level is predicted to be relatively slow and the water levels are expected to recover in approximately 10 years. The slow recovery is contributed by the low hydraulic conductivity of the surrounding geology. In the state of the groundwater trying to reach equilibrium, it leads to groundwater rebound. A rebound curve is presented in Figure 30. This estimated rebound time is estimated at 11 years.
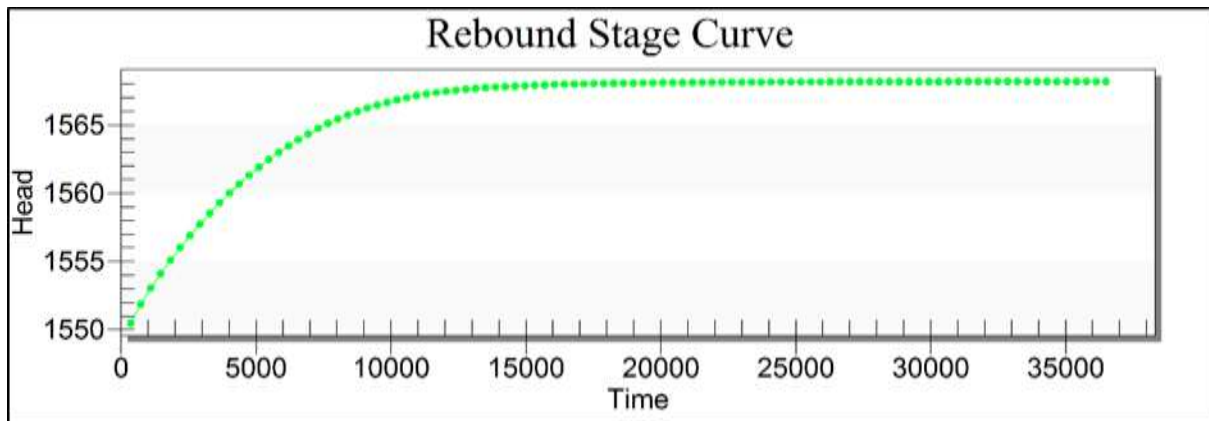
Figure 30: Rebound stage curve of groundwater level in the pit (water level in mamsal vs. time in days)

Groundwater within the mined areas is expected to deteriorate due to chemical interactions between the geological material and the groundwater. The resulting groundwater pollution plume is expected to commence with downstream movement. The predicted groundwater levels indicate that a rise in groundwater will occur and that decant will occur to the southeast of the opencast via sub-surface discharge to the surface after approximately 11 years. The predicted decant areas are shown in Figure 31.
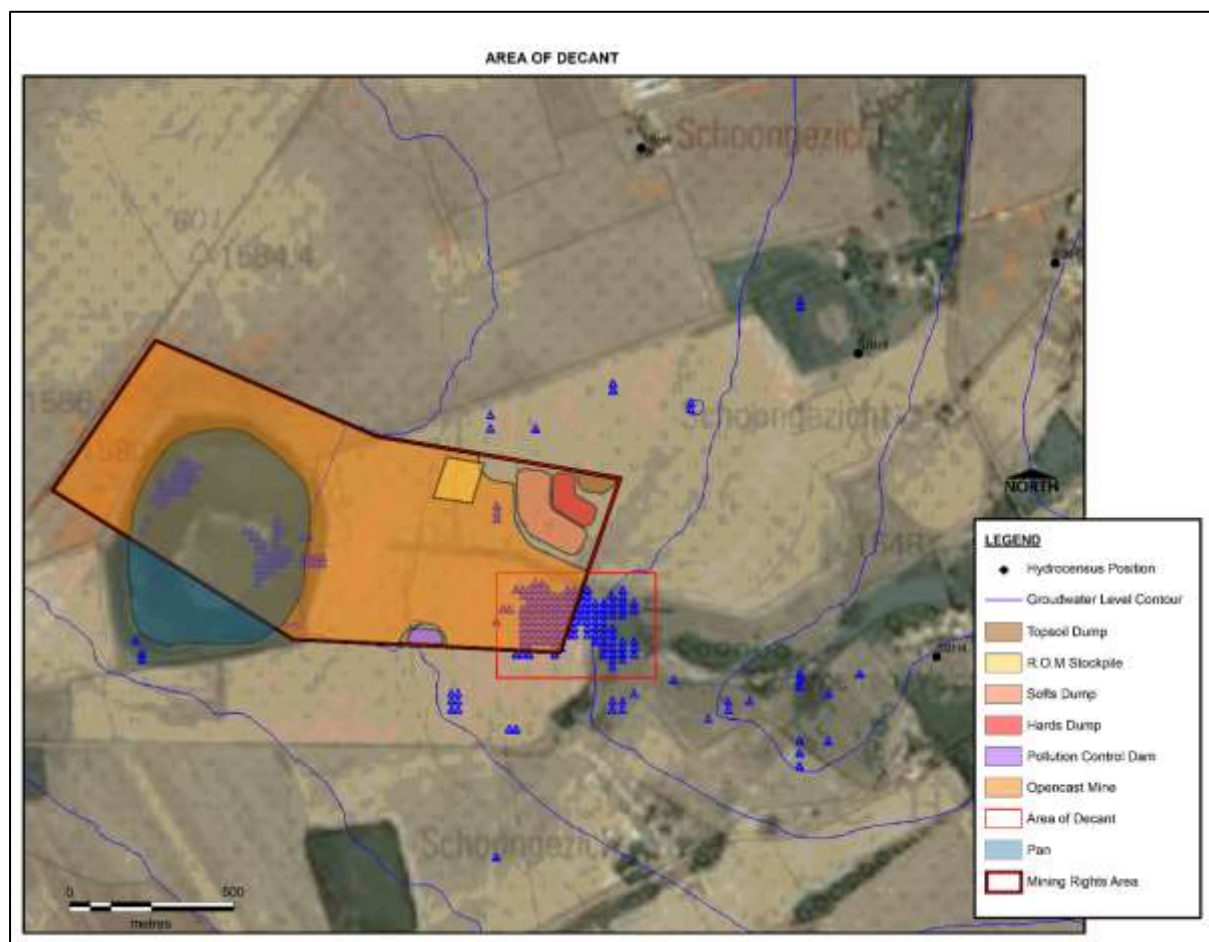
Figure 31: Decanting point of the proposed mine

Once the normal groundwater flow conditions have been re-instated, polluted water could potentially migrate away from the backfilled pit. As sulphate is one of the main pollutant constituents of coal mining, and dominant in mine water chemistry, the predicted transport of sulphate was calculated.

A starting concentration of 1 903 mg/litre of SO_4 was assumed as a worst case scenario. However, geological material is a transient contaminant source and decreases in the concentration of released contaminants are expected over time. A 5% (decay rate- (Mack & Skousen, 2008)) decrease in contaminant release concentrations from the mine were incorporated into the transport modelling. The migration of contaminated water from the backfilled pit was modelled for 5, 10, 25, and 50 years after operations have ceased, and the results are presented in Figure 32.

Figure 32: 5 years, 10years; 25 years; and 50 years (5% decay)

As stated previously, the results must be viewed with caution as a homogeneous aquifer has been assumed. Heterogeneities in the aquifer are unknown and the effect of this cannot be predicted. This can be overcome by installing a monitoring system to verify modelling results.

Within the limitations and assumptions in mind, it is estimated that the plume is unlikely to reach any of the identified hydrocensus boreholes in 50 years after mine. It is estimated that the plume will detach from the backfilled pit after 49 years.

5.8 Statistical Analysis

5.8.1 Pre-mining (Background Data)

A statistical analysis of 48 samples (Table 30) with IBM SPSS 20 was performed on the background hydrochemical data from Mine A, Mine B, Mine C and Mine X to determine if the hydrochemical signatures of the aquifers at these mining sites have similar chemical characteristics. Sample similarity was determined based on a minimum variance percentage and sample groupings with regards to major ion chemistry and physio-chemical parameters.

A dendrogram was constructed to identify which samples are likely to be related. This was performed to identify if the background hydrochemistry between the mines are similar to provide a basis for using similar decant chemistry in remediation calculations for Mine X after further post-mining investigations. (Figure 33).

All samples are grouped together with the exceptions of sample TNDB.TMD.22, which is a sample from Mine C. Sample TNDB.TMD.22 shows elevated EC, SO_4, Al and Ca, and lowered pH of 2.98, relative to other samples in the sample population and is an indication of acid generating conditions (acid already present). This sample can be classified as an outlier for the sample population. A 1% variance was observed between all samples with the exception of TNDB.TMD.22.

Table 30: Samples used in Statistical Analysis (background analysis)

Sample	TDS	NO3NO2	Cl	MALK	SO4	Ca	Mg	Na	K	Fe	Mn	EC	pH	Al
TNDB.ENGENDEPOT	136	0.11	53.1	32	3.52	5	2.1	40.5	0.68	1.63	0.58	20.6	6.29	0.11
TNDB.TMD.22	1808	0	17.2	0	1156	106	50.9	38.8	8.98	198	7.31	195.4	2.98	41.6
TNDB.10322.08.16	268	0.2	21.8	125	66	26.4	13.1	48.5	5.6	0.16	1.35	40.1	7.25	0.17
TNDB.BH05AFGRIGRIT	124	0.16	31.1	26	26.1	7.29	4.24	20.5	1.06	7.91	0.69	17.7	5.93	0.06
TNDB.10322.07	492	15.6	29.4	4	218	13.4	7.3	101	3.6	0.04	5.1	75.2	4.81	0.37
LP.BH4A	88	0.45	3.71	5	49.2	9.4	5.09	6.29	1.55	0.22	1.93	14.4	4.59	1.11
LP.BH4B	76	0.53	3.77	5	40.5	9.16	4.24	6.33	1.67	0.12	0.8	11.4	5.87	0.11
LP.BH5	176	0.1	5.13	131	13.5	39.3	7	15.4	3.57	0.14	0.01	28.1	8.11	0.11
LP.BH6	70	0.6	8.62	28	13.6	6.62	3.77	10	4.11	0.32	0.06	9.83	6.62	0.2
LP.BH7	78	0.98	26.4	20	5.76	9.08	5.29	8.42	6.62	0.21	0.03	13.4	6.61	0.1
LP.BH10	20	0.37	3.53	8	2.01	1.5	0.77	3.63	0.71	0.06	0.01	1.9	6.08	0.05
LP.BH11	22	0.32	3.44	10	1.95	1.84	0.94	3.86	1.1	0.15	0.02	2.06	6.15	0.12
LP.BH14A	24	1.12	4.26	6	1.56	1.42	0.81	5.05	0.62	0.08	0.02	2.24	5.66	0.07
LP.BH15	94	0.1	3.34	78	5.04	17.7	7.85	6.82	3.61	0.16	0.12	15.8	7.38	0.1
LP.BH16	116	0.1	3.23	92	7.79	25.2	6.24	9.02	3.44	0.11	0.03	19.3	7.92	0.1
LP.BH17	30	1.39	5.16	10	1.58	2.22	0.89	6.23	0.56	0.26	0.02	3.31	5.77	0.17
LP.BH18	26	1.04	3.84	8	1.5	1.7	0.81	5.07	0.72	0.15	0.02	2.24	6.01	0.1
KD.KRW024		0	4.42	20.01	4	2.27	1.39	3.88	0.9644	0	0.04	4.9	5.87	
KD.KRW026		0	4.47	2.49	6	1.28	0.33	3.82	0.2553	0.0044	0	2.13	5.97	0.0003
KD.KRW027		0	4.26	0.55	4.18	0.27	0.04	2.97	0.1606	0.0071	0	0.89	5.43	0.0082
KD.KRW029		0	4.84			6.87	3.43	4.6	1.5259		0.61	13.84		3.3455
KD.KRW030		0	4.92	0	35.65	5.09	1.95	4.03	1.658	0.1768	0.37	7.15	4.73	0.2781
KD.KRW031		0	3.94	39.84	32.18	12.6	3.48	4.78	8.931	0.8909	0.02	11.11	6.37	0.0781
KD.KRW034		0	4.48	0	176	25.72	12.63	9.55	14.212	0.5649	2.03	57.42	3.1	0.1737
KD.KRW035		0	4.36	6.19	5.5	0.92	0.23	5.11	0.8527	0.0018	0.01	1.74	5.65	0.0021
KD.KRW036		0	3.39	0	190	32.23	23.54	6.34	4.2533	0	1.14	36.89	4.76	0.9077

| Sample	TDS	NO3NO2	Cl	MALK	SO4	Ca	Mg	Na	K	Fe	Mn	EC	pH	Al
KD.KRW038		0	3.85	2.4	41.12	7.29	1.23	4.6	3.7852	0.0007	0	6.8	5.99	0
KD.KRW039		0	5.46	0	76.95	8.46	5.99	5.46	3.3109	0.1081	0.66	17.65	3.95	1.8298
KD.KRW040		0	4.13	4.25	8.11	0.59	0.17	4.19	1.343	0.0172	0.01	2.34	5.2	0
KD.KRW041		0	4.63	6.3	6.92	0.61	0.23	4.26	3.0558	0.0324	0.01	3	5.7	0.001
KD.KRW043		0	6.41	3.78	13.82	3.19	0.98	5.02	1.3175	0.013	0	4.26	5.99	0.0028
KD.KRW049		0	8.14	95.84	189	59.35	22.75	23.67	17.0862	0.0215	1.42	55.5	6.52	0.0193
PDW.SHBH1	86	0.1	3	63	8.7	12.9	5.49	7.19	2.07	0.01	0.29	14.07	7.66	0.01
PDW.SHBH2	118	0.17	3	90	8.2	17.8	5.6	13.1	1.5	0.01	0.12	18.03	7.63	0.01
PDW.SHBH3	116	0.12	4	71	18.8	13.8	5.61	15.9	1.88	0.31	0.05	18.1	7.71	0.01
PDW.SHBH4	160	0.38	6	126	11.7	22.6	16.8	7.77	3.17	0.01	0.01	7.22	7.22	0.01
PDW.SHBH5	80	6.4	6	48	8.8	11.3	4.95	8.81	0.98	0.01	0.01	6.87	6.87	0.01
PDW.SHBH6	56	7.6	9	25	6.9	5.23	2.98	8.2	3.44	0.01	0.01	6.84	6.84	0.01
PDW.SHBH7	154	0.13	6	113	6.5	7.72	2.27	45.7	2.04	0.01	0.01	23.55	7.72	0.03
PDW.SHBH8	60	0.12	5	35	13	6.01	1.12	5.12	2.46	0.18	0.01	5.24	6.5	4.37
PDW.SHBH9	196	0.72	5	164	6.9	24.1	10.6	26.5	7.23	0.01	0.01	32	7.9	0.01
PDW.SHBH10	54	0.12	3	40	3	4.05	3.68	5.1	3.34	1.43	0.28	8.64	7.14	0.01
PDW.SHBH11	98	3.6	4	73	7.2	15.6	8.84	8.27	1.4	0.01	0.01	18.37	7.81	0.01
PDW.SHBH13	60	2.1	4	38	6.7	6.53	2.99	7.4	1.52	0.01	0.01	10.15	7.32	0.02
PDW.SHBH14	64	2.2	5	42	4.5	8.39	4.23	7.79	1.89	0.01	0.01	11.84	7.29	0.01
PDW.PBH1A	149	6.53	2.82	118	16.7	22.1	7.18	26.5	4.86	0.55	0.05	23	7.84	1.82
PDW.PBH2A	111	6.44	33.4	43	69.6	18.2	9.22	28.8	15.9	7.57	0.24	17	6.58	65.5
PDW.PBH3A	40	7.64	1.55	27.7	12.3	4.06	2.57	8.47	5.05	1.49	0.05	6.16	6.24	3.89
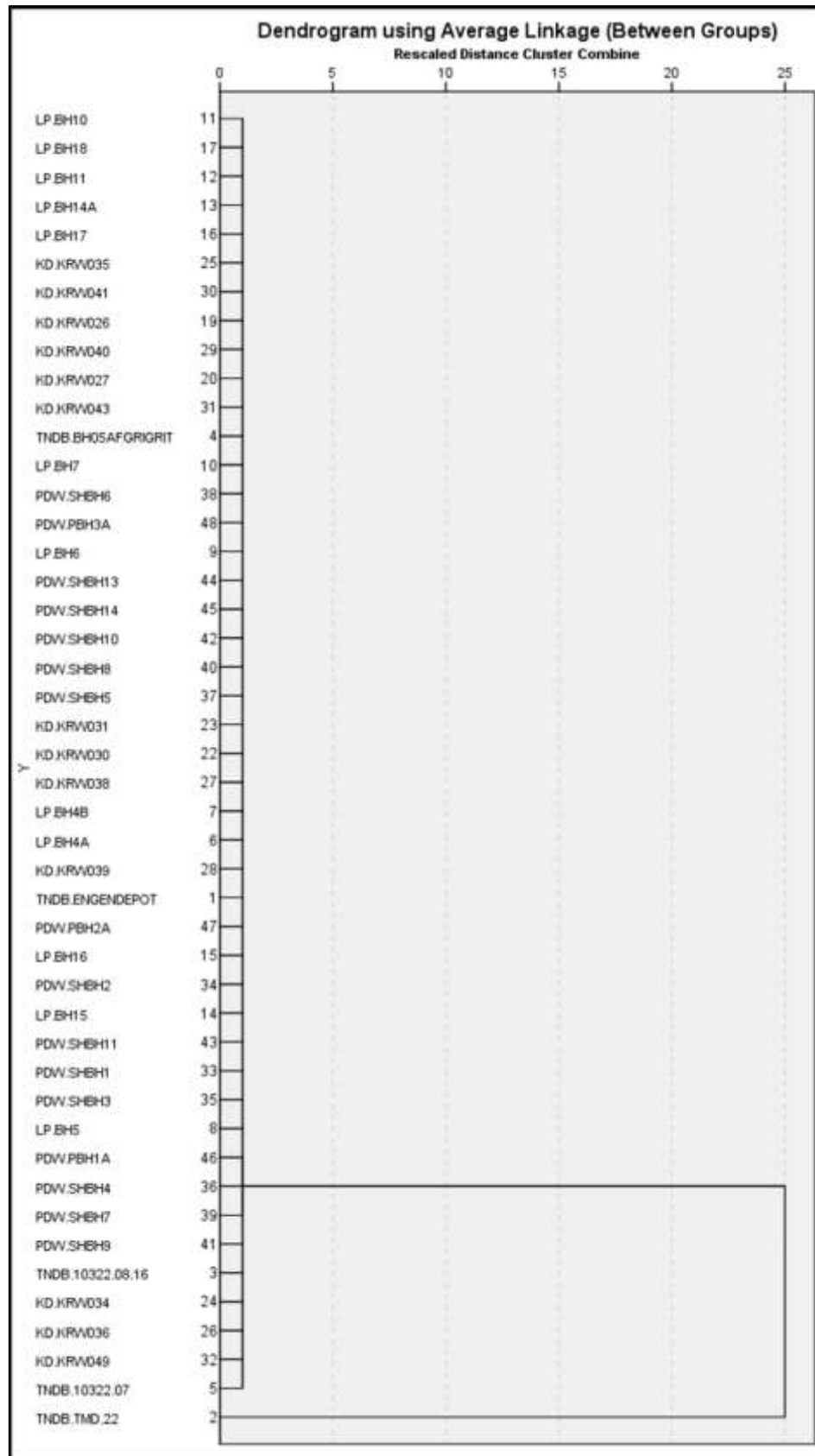
Figure 33: Dendrogram relating all analysis data between the collected background sample data from Mine A. Mine B, Mine C and Mine X.

5.8.2 Post-mining (Decant Data)

A statistical analysis of 10 decant samples (Table 31) with IBM SPSS 20 was performed on the discharge (decant) hydrochemical data from Mine A, Mine B and Mine C. A dendrogram was constructed (Figure 34) for the analysis data to determine which samples are likely to be related and to determine the reason for this relation or difference, if any.

The dendrogram identified four distinct groups with each representing 1% variance between samples. Group 1 consists of samples (LP.SW1), (LP.SW2), (LP.SW3), and (TNDB.MineWater). Similarly, group 2 consisting of samples (KD.KRW012) and (KD.KRW014) also show a 1 % variance; group 3 consisting of samples (TNDB.TDB.2) and (TNDB.TDBD.1) show a 1 % variance; and group 4 consisting of samples (LP.SWS) and (KD.WP054), also a show 1% variance.

When comparing group 1 and group 2, a variance of 8% is evident. When comparing group 3 and group 4, a variance of 4 % is evident. However, a 25 % variance is evident when comparing group 1 and group 2 to group 3 and group 4. Based on the constructed dendrogram a variance of >25% between samples was considered to show a significant variability between sample groups for the purposes of this study. Therefore, samples from group 1 and group 2 mentioned above can be interpreted to be less affected by acid rock drainage when compared to group 3 and group 4, which show a more advanced stage of AMD development. Group 3 and group 4 have a low pH (<3.16) and are high in Fe (21.8 mg/L) and SO4 (>4198 mg/L).

Table 31: Deterministic Mine Water Decant Quality (decant quality)

Sample	TDS	NO3NO2	Cl	MALK	SO4	Ca	Mg	Na	K	Fe	Mn	EC	pH	Al
TNDB.TDB.2	8002	0	171	0	5679	480	229	1477	40.9	60.5	13.8	945	3.2	32.4
TNDB.TDBD.1	8402	0	200	0	5983	467	223	1701	38.5	21.8	12.7	1007	3.16	13
TNDB.MineWater	4276	0.2	11	0	2835	425	270	25.3	6.9	2.42	49.9	396	3.46	83.8
LP.SW1	3808	0.31	17.6	0	2697	333	179	41.4	10.7	161	32.8	385	2.79	119
LP.SW2	3812	0.39	17.1	0	2674	329	177	40.9	10.4	160	32.5	386	2.79	117
LP.SW3	3904	0.42	17.1	0	2739	346	183	41.1	9.36	119	34	405	2.68	119
LP.SW5	6002	0.1	23.4	0	4198	482	267	55.2	3.15	259	61.6	559	2.51	141
KD.KRW012		0	8.62	13.65	271	52.38	21.65	9.79	21.328	0	5.41	50.44	5.75	0.112
KD.KRW014		0	10.5	0	206	32.35	11.25	8.32	7.679	0.009	4.09	38.08	4.12	7.2626
KD.WP054		0.09	5.11	0	5484	268.48	166.07	7.88	11.514	552.249	64.09	469.19	2.64	260.626

Table 32: Predicted mine water decant from Mine X.

| Average [ ]														
Sample	TDS	NO3/NO2	Cl	MALK	SO4	Ca	Mg	Na	K	Fe	Mn	EC	pH	Al
TNDB.Minewater	4276	0.2	11	0	2835	425	270	25.3	6.9	2.42	49.9	396	3.46	83.8
LP.SW1	3808	0.31	17.6	0	2697	333	179	41.4	10.7	161	32.8	385	2.79	119
LP.SW2	3812	0.39	17.1	0	2674	329	177	40.9	10.4	160	32.5	386	2.79	117
LP.SW3	3904	0.42	17.1	0	2739	346	183	41.1	9.36	119	34	405	2.68	119
KD.KRW012		0	8.62	13.65	271	52.38	21.65	9.79	21.328	0	5.41	50.44	5.75	0.112
KD.KRW014		0	10.5	0	206	32.35	11.25	8.32	7.678	0.0089	4.09	38.08	4.12	7.263
AVERAGES:	**2633**	**0.22**	**13.65**	**2.275**	**1903**	**252.96**	**140.32**	**27.80**	**11.06**	**73.72**	**26.45**	**276.75**	**3.598**	**74.36**
The above average concentrations were used as input parameters for the Geochemical analysis.														
KD.WP054		0.09	5.11	0	5484	268.5	166.07	7.88	11.51	552.249	64.09	469.19	2.64	260.63
LP.SW5	6002	0.1	23.4	0	4198	482	267	55.2	3.15	259	61.6	559	2.51	141
TNDB.TDB.2	8002	0	171	0	5679	480	229	1477	40.9	60.5	13.8	945	3.2	32.4
TNDB.TDBD.1	8402	0	200	0	5983	467	223	1701	38.5	21.8	12.7	1007	3.16	13
AVERAGES:	5602	0.048	99.88	0	5336	424.37	221.27	810.3	23.5	223.4	38.05	745.05	2.878	111.76
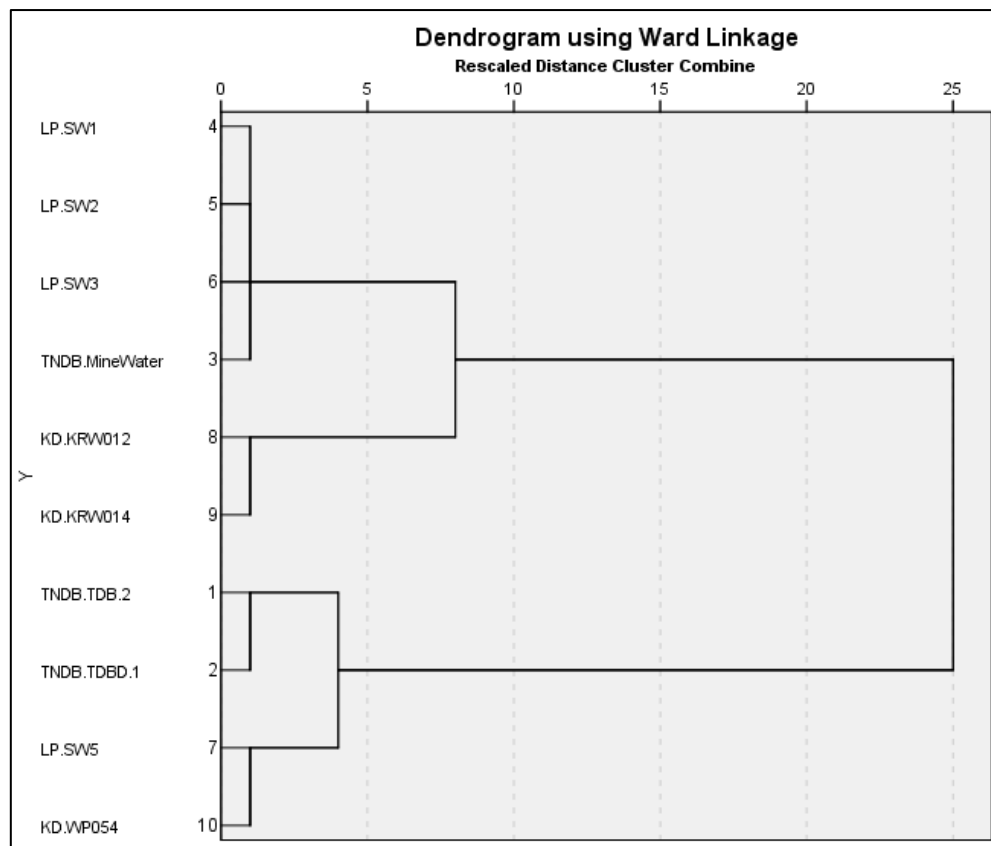
Figure 34: Dendrogram relating all analysis data between the collected decant data from Mine A, Mine B and Mine C.

Table 33 below summarises the different operational processes that could possible influence decant (AMD) concentrations at various mines.

Table 33: AMD Prediction ((EPA), 1994)

Operations	Acid Generating factors effected	Variables
Blasting Removal Storage	• Sulphide source • Particle size • pH variation • Rapid O2 release • O2 entry • AMD stage	• Variable concentrations in source • Mixing of material • Aerations • Preferential flow paths • Infiltration • Hydraulic conductivity

Table 34: Comparison of Acid Rock Drainage Factors in Waste Rock Stockpiles and Tailings Impoundments (Brodie et al., 1991).

Acid Generating Factors	Waste Rock Pile	Tailings Impounded
Sulphide Source	• Variable in concentration and location. • Conditions may vary from sulphide rich to basic over short distances.	• Conditions uniform, often with very high sulphide content.
Particle Size	• Average rock size typically greater than 20 cm (but highly variable).	• Tailings may be 100 % less than 0.2 mm.
pH Variation	• Highly variable conditions over short distances.	• Fairly uniform conditions with a few major horizontal zones.
Initiation of Rapid Oxidation	• Usually starts immediately after first rock is placed (in "trigger" spots).	• Usually starts after tailings placement ceases at the end of mine life.
Oxygen Entry	• Pit along preferential flow paths. • Seasonal variation in flow path "flushes" out stored products resulting in concentration peaks.	• Seepage slow and uniform. • Reduced flow paths variation and stored product "flushing".
ARD Release	• Large infiltration resulting in large seepage from toe and to groundwater. • Rapid release following generation, sometimes with both neutralized and acid ARD seeps.	• Large early top surface ARM runoff. • Lower infiltration. • Gradual transition in seeps from process water to neutralized ARM to low pH ARD.

5.9 Geochemical Model

The mine water hydrochemistry concentrations predicted in Table 32.
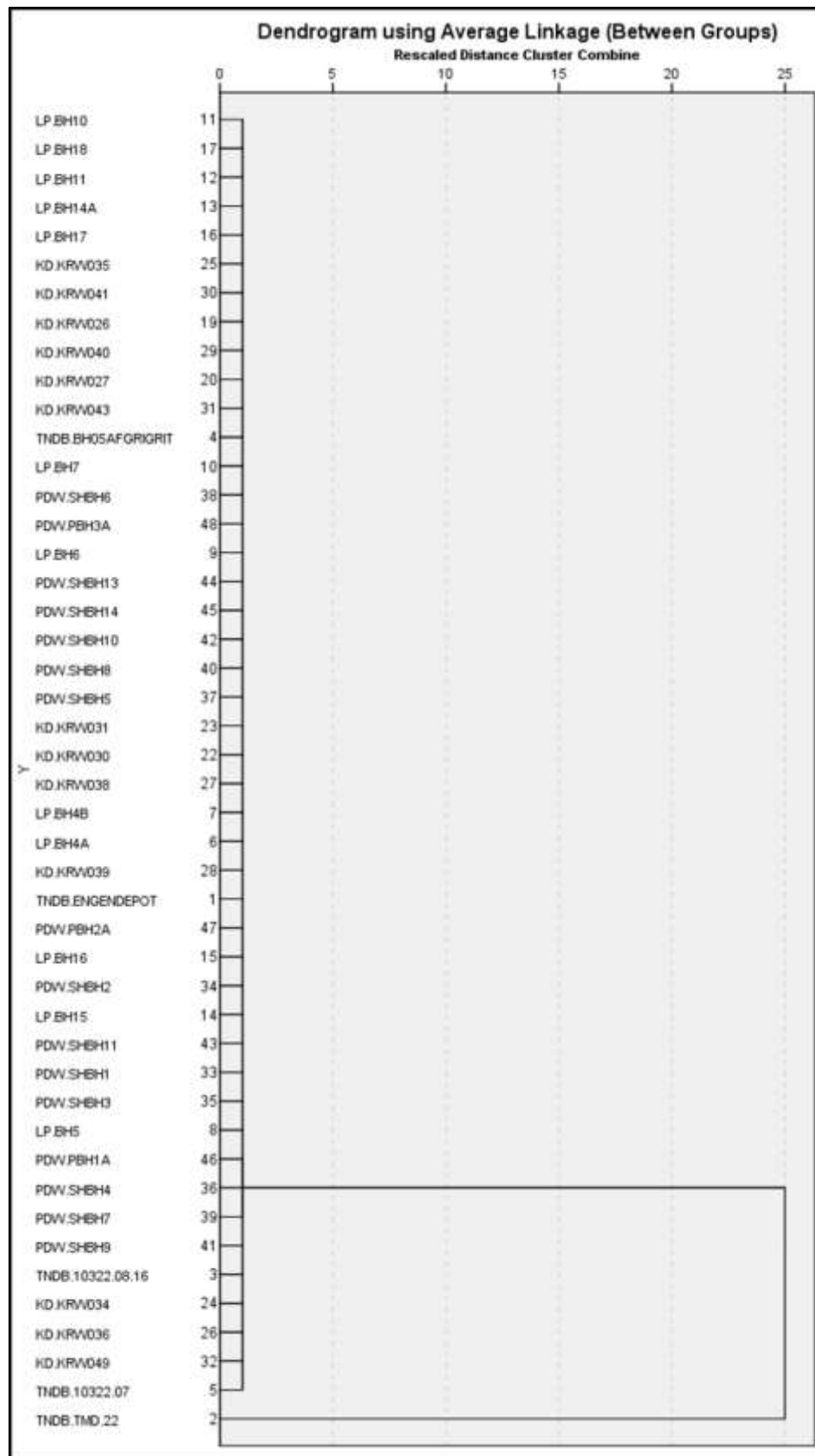

Dendrogram using Average Linkage (Between Groups)

*Figure 33: Dendrogram relating all analysis data between the collected background sample data from Mine A.
Mine B, Mine C and Mine X.*

Post-mining (Decant Data)' was used as input data to the geochemical model.

It must be cautioned that these calculations have been performed using simplified assumptions which may not always represent a real world scenario (refer to assumptions made in "3.10.1 Assumptions of Geochemical Model" of this report).

5.9.1 Relationship between pH and SO4 (measured data)

To determine the statistical relationship between pH and SO_4, a total of 1790 samples was analysed from a predetermined radius around the study area.

Firstly, pH was converted to a hydronium ion (H^+). The H^+ ions concentrations were then compared to concentrations of SO_4 measurement. The following formula will be used for the conversion:

$$pH = 10^{-x}$$

Secondly, a mathematical expression of the relationship between the two parameters was represented by a straight line formula. This straight trendline equation was then used to obtain calculated H^+ concentrations based on measured SO_4 concentrations. The calculated H^+ concentrations were then converted back to pH. The calculated pH was then compared to the measured pH to determine variation from measured values. A variance in pH of 0.8 was calculated.

To determine the correlation fit of the linear trend line, a fit (R^2) greater than 0.7 was deemed significant.

Additionally, using SPSS, all available groundwater SO_4 and H^+ concentrations were used for correlation by bivariate regression analysis using Pearson's method. A two-tailed significance test was used with a significance level of 99% ($\alpha=0.99$). The result shows a positive correlation at a 99% confidence interval with a coefficient of variation of 0.846.

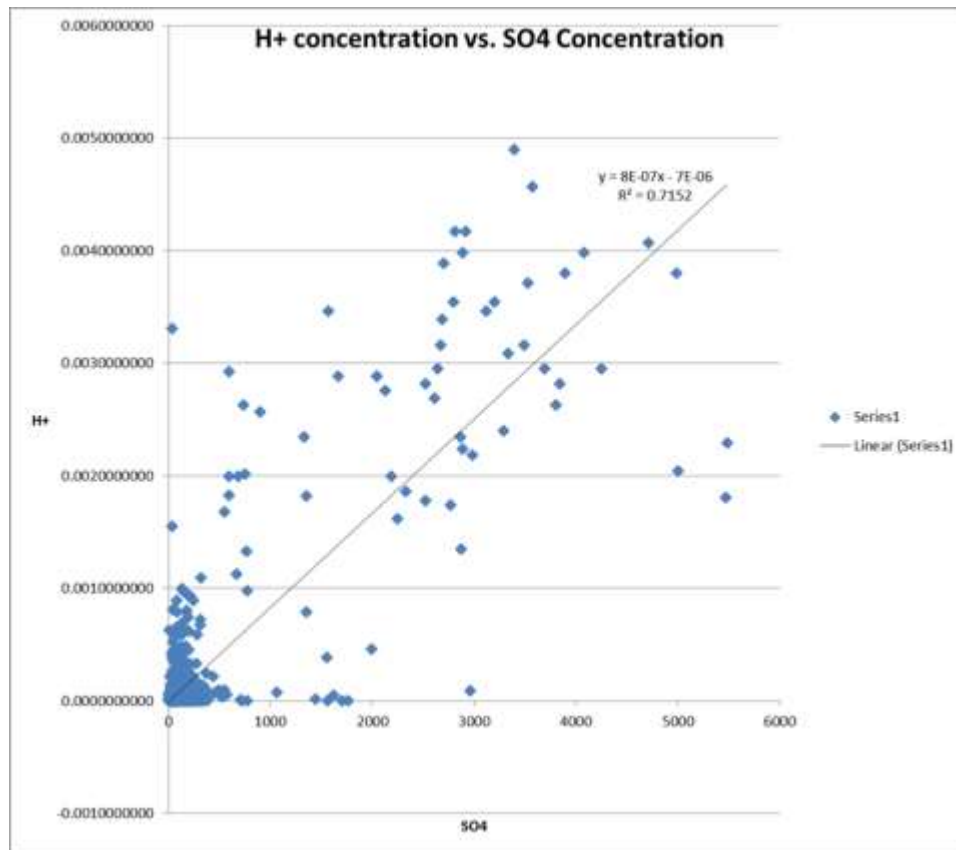*Figure 35: Mathematical relationship between measured pH and SO4*

The general straight line trend equation is represented as y= mx+c, where m is the gradient, and y=c is the value where the line cuts the y axis was used to present the relationship between measure pH and SO_4 concentrations. The following equation was read from the graph:

$$y= 8E\text{-}07x - 7E\text{-}06$$

The coefficient of determination (R^2) measures how well the regression line approximates the real data points. An R^2 of 1 indicates that the regression line perfectly fits the data. From the above graph, R^2 was calculated as:

$$R^2 = 0.7152$$

As described in section 3.10.2.1, an R^2 of greater than 0.7 is deemed significant. $R^2$is greater than 0.7 and is regarded as a significant fit.

5.9.2 The relationship between pH and the volume of pre-treatment product pH vs. SO4 (Model)

The mathematical relationship using the above equation was used to calculate pH for a given SO$_4$ concentration. The straight line equation determined in 5.9.1 was used to calculate the H$^+$ concentration relative to SO$_4$ concentration from the numerical transport model. The mean variance of 0.8 calculated in 6.9.1 for the modelled data was added to obtain a more accurate correspondence to measured data and for the worst case scenario. Figure 36 shows the relationship of calculated SO4 from the numerical model to the calculated pH from the geostatistical model. pH concentrations ranges between 3.6 (year 1) to 4.7 (year 50), while SO$_4$ concentration range between 1950 mg/l (year 1) to 157.94 mg/l (year 50).
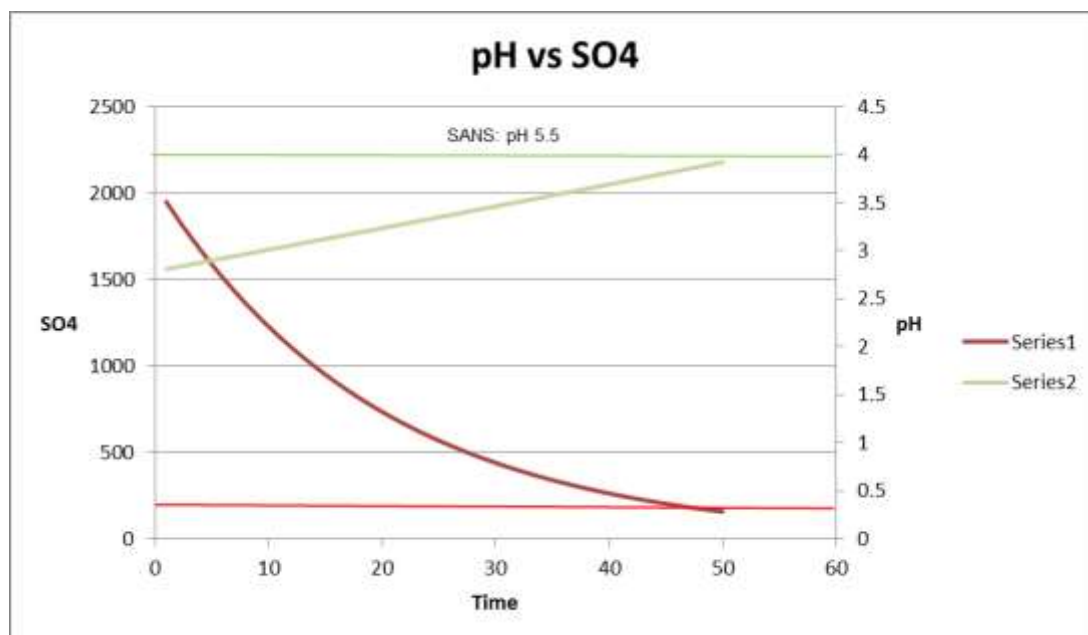

Figure 36: Calculated pH (geo-statistical model) vs. calculated SO$_4$ (numerical model)

5.9.3 Determine the dynamic volumes of treatment product to be added.

The dynamic volumes of treatment product required was calculated by determining the required raise in pH (pH jump required) from initial concentration (discharge quality of start of mine decant) to the required pre-treatment of pH 5.2.

Three pre-treatment products were identified: Lime, Limestone and Sodium Hydroxide (NaOH).

5.9.3.1 Pre-treatment with Lime

To determine the dynamic amount of product needed, various pH values relative to reaction progress were examined. The fraction of lime reacted versus pH (Figure 37) shows that larger volumes of lime is required to treat a lower pH value as a larger change in pH value is requires.
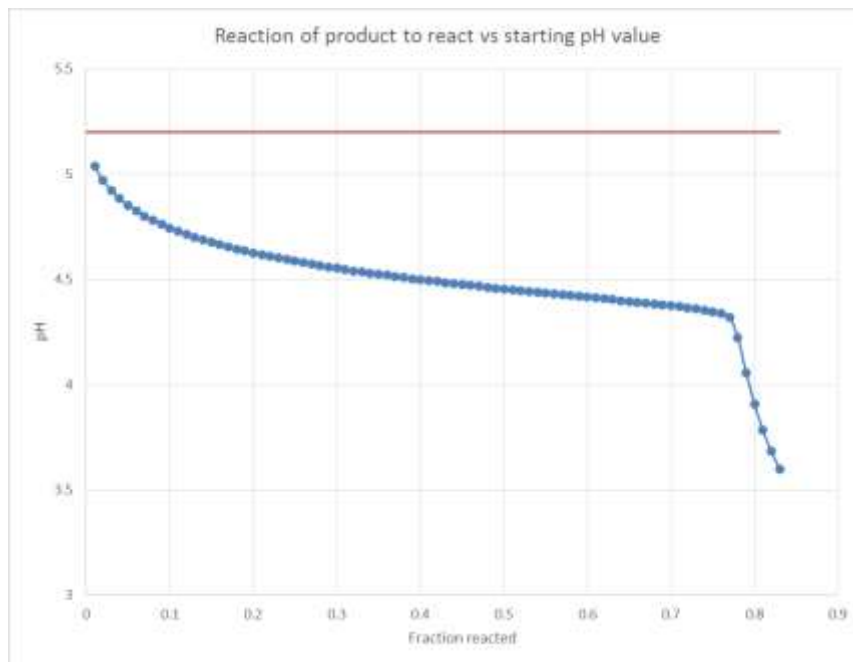
Figure 37: Reaction of product to react vs. starting pH.

5.9.3.1.1   Geochemical Analysis of dominant constituents in reaction (Lime)

The ions causing non-linear cumulative raises in pH during the reaction are Al^+ and K^+ (Figure 38). As pH drastically decreases so too does K^+ concentration. Al concentration decreases gradually until a reaction fraction of 0.87 (pH ±5), where K^+ concentrations increase slightly.
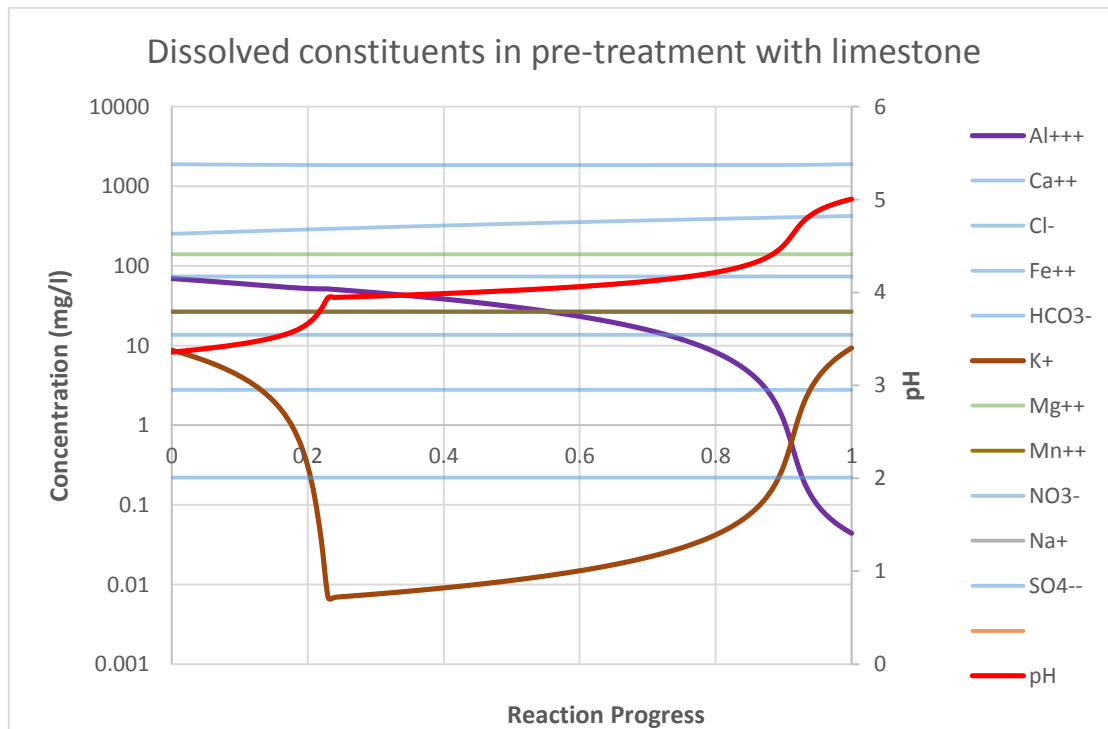

Figure 38: Identification of constituents causing non-linear cumulative raise in pH during product reaction

Eh-pH diagrams also known as Pourbaix diagrams are often used to indicate the range of redox potentials and pH values in which a chemical species is stable. This could be in an aqueous, solid or gaseous phase. The main purpose is to indicate the predominance areas of pH and redox conditions in which one would expect to find a particular species.

When raising the pH in the pre-treatment with lime, under low pH conditions, aluminium sulphate concentrations are found in solution. As pH increase to pH 3.8, Al^+ precipitates out and Alunite is formed (Figure 39). Alunite consists of a hydrated aluminium potassium sulphate mineral. At pH 5.1, alunite becomes thermodynamically unstable and gibbsite $(Al(OH)_3$ becomes stable. Therefore alunite breaks down and releases K^+ in the process to change to a more stable form. The blue line indicates the progress of the reaction.

Additionally, in Figure 40 potassium precipitates out of solution at pH 3.8, confirming the possible formation of alunite, and appears again at pH 4.6 in solution when gibbsite is formed.

Pourbaix diagrams are a function of both pH and redox potential, therefore the Pourbaix diagrams for all three pre-treatment products will be the same taking into account that that the starting mine water chemistry is used, and pre-treating within the same pH range (pH 3 – 5.2).
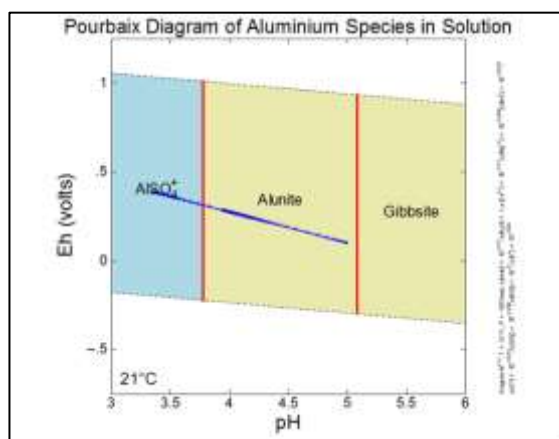

Figure 39: Pourbaix diagram of for SO4 (Eh-pH)
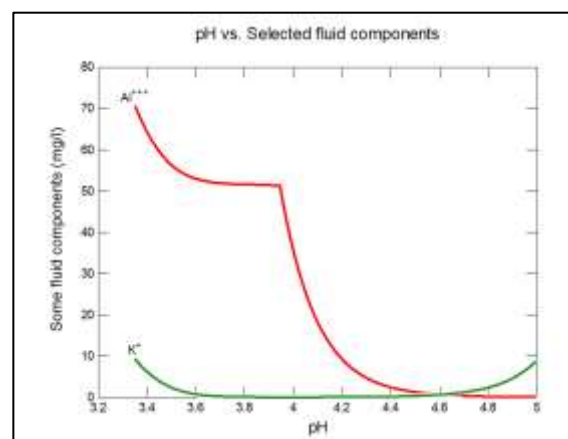
Figure 40: Stability Diagram and pH vs. selected Fluid Components (lime)

5.9.3.1.2   Lime volumes and Costs

To estimate the volume of pre-treatment product needed to treat pH, the weight of lime required for each pH jump was calculated using Geochemist's Workbench.

The total volume of lime required to raise pH from initial concentration (discharge quality at the start of mine decant) to required concentration (pH 5.2) was calculated by examining various pH values relative to the fraction of lime to react, assuming 100% reaction of lime product. The relationship between the fraction of lime reacted and raise in pH required can then be graphically displayed on a scatter plot diagram. This representation aided in determining the relationship between pH change and product mass. A trendline (polynomial)

was then fit to the graph and a mathematical expression of the relationship between the lime mass and raise in pH could be determined (Figure 41). The polynomial trendline equation was then used to calculate the required product for various calculated pH values based on modelled SO_4 concentrations.

An early stage and a late stage curve (Figure 42) trendline equation was developed to factor in the influencing hydrochemistry constituents Al^+ and K^+. These separate equations allowed for a more accurate calculation on the amount of product needed per year.
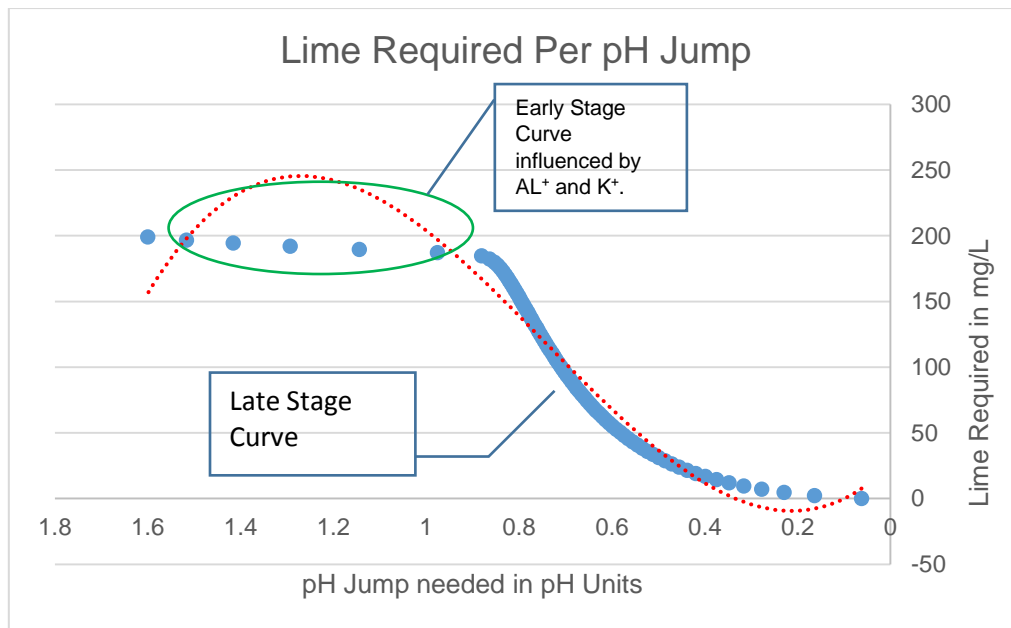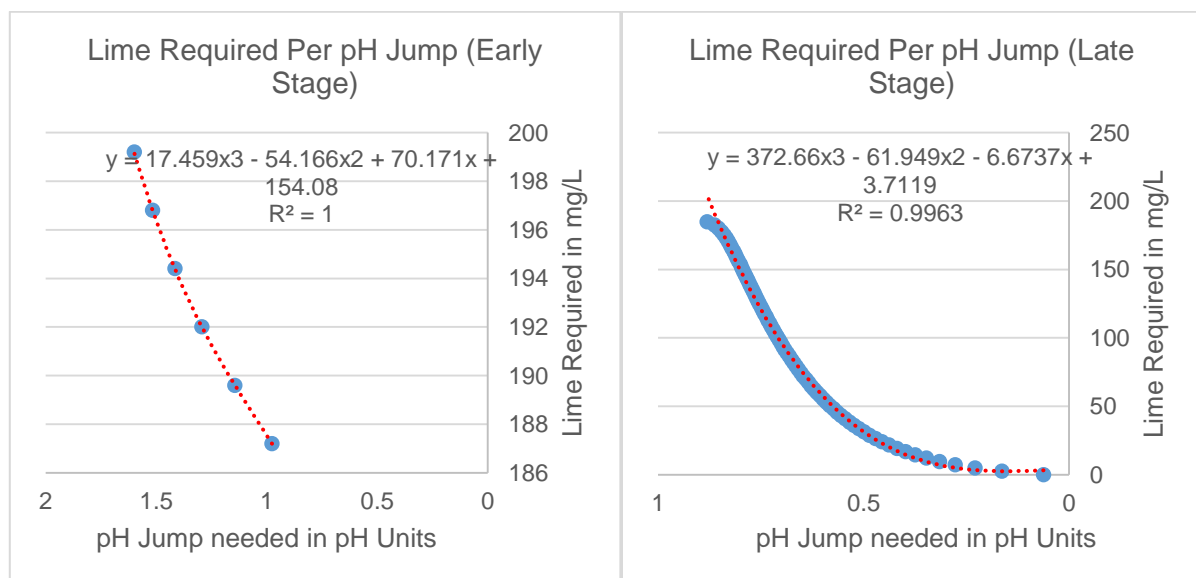

Figure 41: Lime required per pH jump.

Figure 42: Early and Late Stage Curve-lime required per pH jump

The calculated results show an estimated total cost of R 9 770 686-00 (assuming a limestone price of R3285 -00/ ton) with a maximum treatment cost in year one of approximately R689163-00 and a minimum approximate cost in year 50 of R63021-00, with an annual decrease of approximately a R12 315-00/year for the first 30 years.
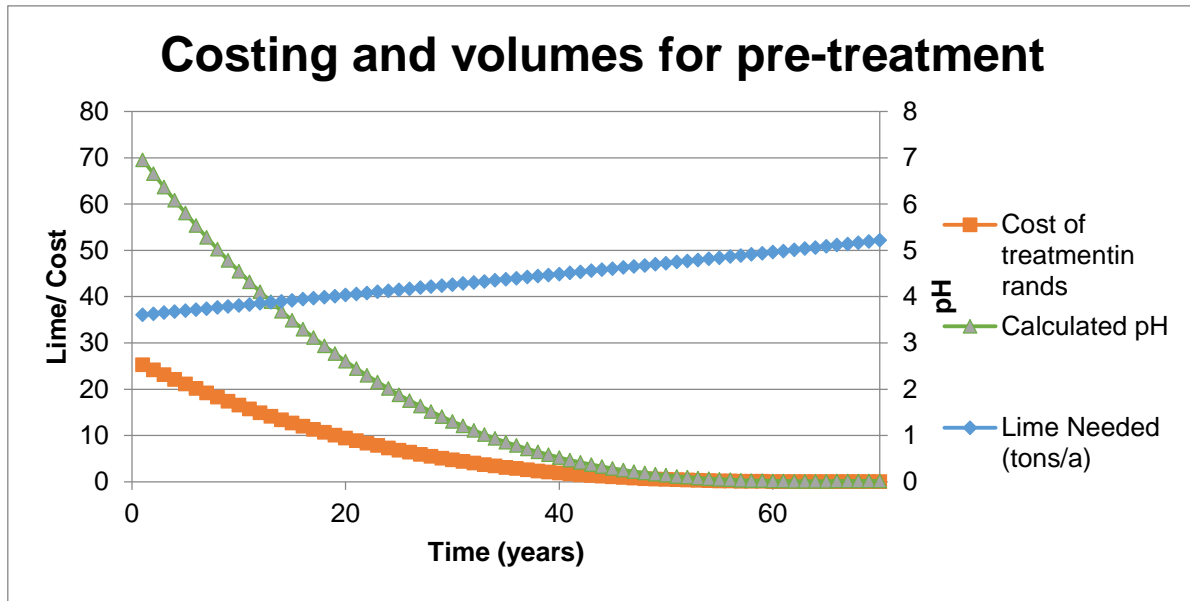

Figure 43: Relationship between pH, Lime needed, Lime Cost, and number of Years.

5.9.3.1.3   Summary of results: Lime volumes and Costs

Influencing Chemistry	Mineral Precipitation	pH Jump	Formula	Total Lime Required in tones (50 year period)	Cost of Lime per tonne (May 2018)	Total Lime Cost
K	Alunite to Gibbsite	3.6 to 5.2	$y=372.66(x^3)-61.949(x^2)-6.6737(x)+3.7119$ $R2 = 0.9963$	4398	R2433-00	R10 700 537-00

5.9.3.2 Pre-treatment with Limestone

As with lime, to determine the dynamic amount of limestone needed, various pH values relative to reaction progress were examined. The fraction of limestone reacted versus pH (Figure 44) shows that a larger volume of limestone is required to treat a lower pH value.
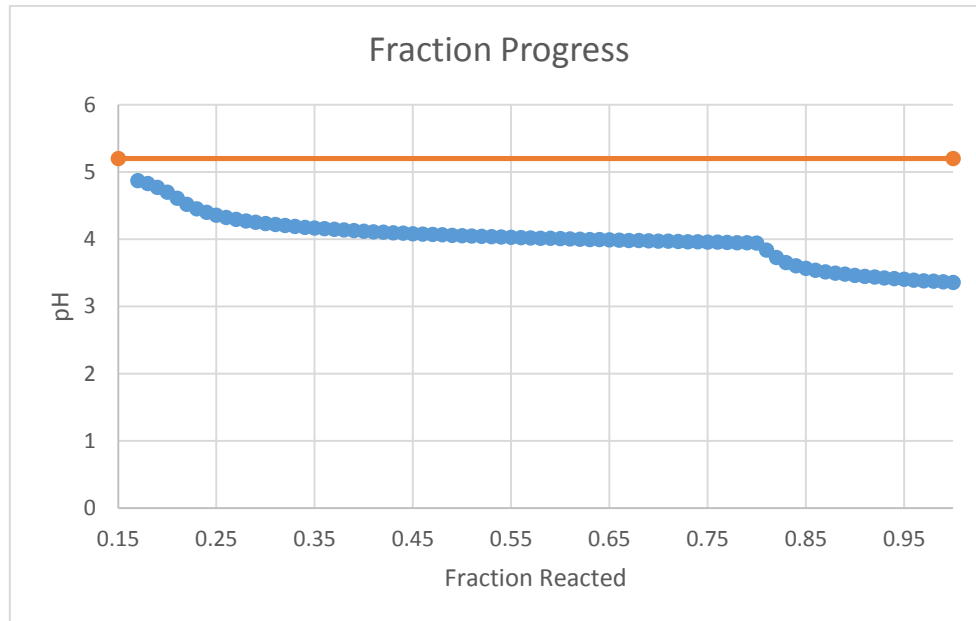

Figure 44: Reaction of product to react vs. starting pH (Limestone)

5.9.3.2.1   Geochemical Analysis of dominant constituent in reaction (Limestone)

With the use of limestone as a pre-treatment chemical, the major ions causing non-linear cumulative raises in pH during the reaction are Al^+ and K+ (Figure 45). With the addition of limestone ($CaCO_3$), the dissolved HCO_3 concentrations increase, causing pH to drastically increase. The concentrations of K^+ and Al^+ decrease until a reaction fraction of 0.35 (pH ±4), where K+ concentrations increase slightly.
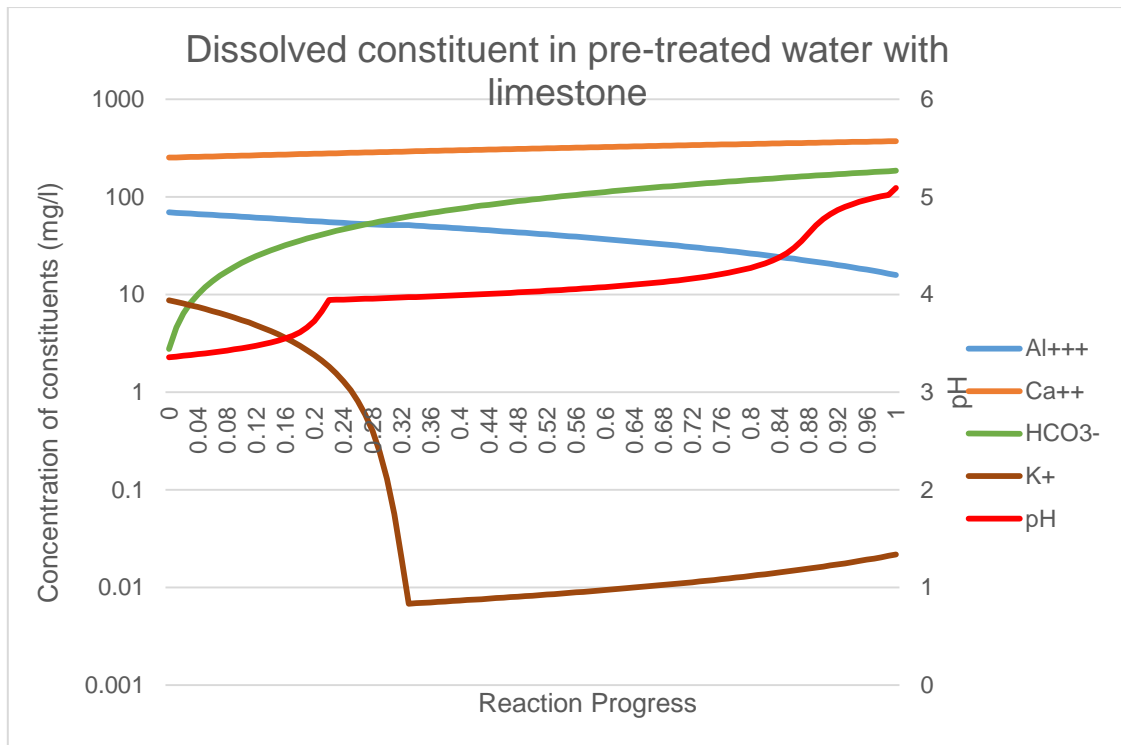
Figure 45: Identification of constituents causing non-linear cumulative raise in pH during product reaction

5.9.3.2.2 Limestone volumes and Cost

To estimate the volume of pre-treatment product needed to treat pH, the weight of limestone required for each pH jump was calculated.

An early stage and a late stage curve (Figure 46) trendline equation was developed to factor in the influencing hydrochemistry constituents Al^+ and K^+. These separate equations allowed for a more accurate calculation on the amount of product needed per year.
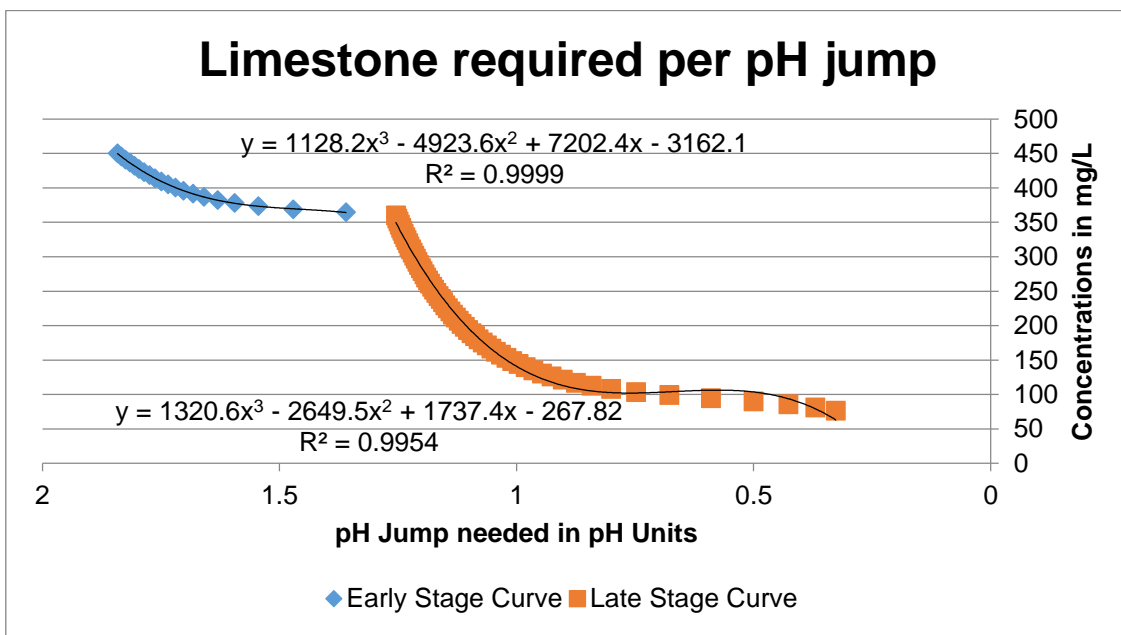
Figure 46: Limestone required per pH jump

The calculated results show an estimated total cost of R9 770 656 -00 (assuming a limestone price of R3285 -00/ ton) with a maximum treatment cost in year one of approximately R 689 163-00 and a minimum approximate cost in year 50 of R63 021-00, with an annual decrease of approximately a R12887-00/year, for the first 24 years.
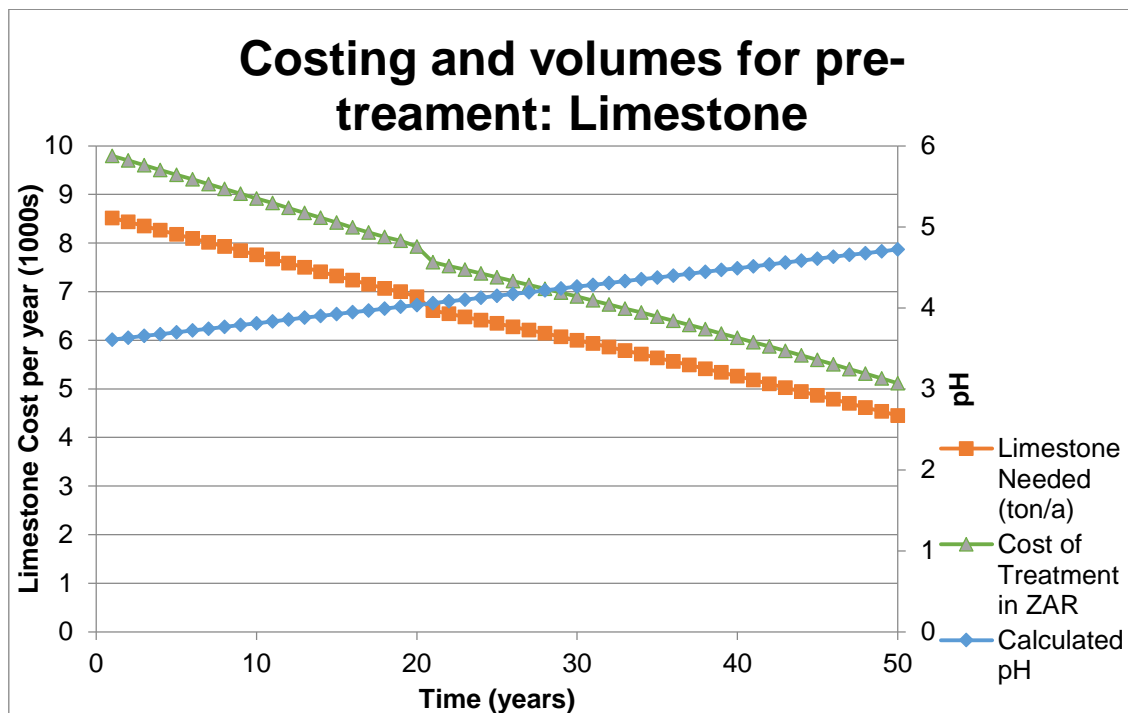
Figure 47: Relationship between pH, Limestone needed, Lime Cost, and number of Years

5.9.3.2.3   Summary of Results: Limestone volumes and Cost

Influencing Chemistry	Mineral Precipitation	pH Jump	Total Limestone Required in tones	Formula	Cost of Limestone per tonne (May 2018)	Total Lime Cost
Al, K, HCO₃	Alunite to Gibbsite	3.6 to 5.2	2975	$Y = 1320.6\,(x^3) - 2649.5\,(x^2) + 1737.4\,(x) - 267.82$	R3285 - 00	R9 770 686-00

5.9.3.3 Pre-treatment with NaOH

To determine the dynamic amount of NaOH needed, various pH values relative to reaction progress were examined. The fraction of lime reacted versus pH (Figure 48) shows that larger volume of NaOH is required to treat a lower pH value as a larger change in pH value is requires.
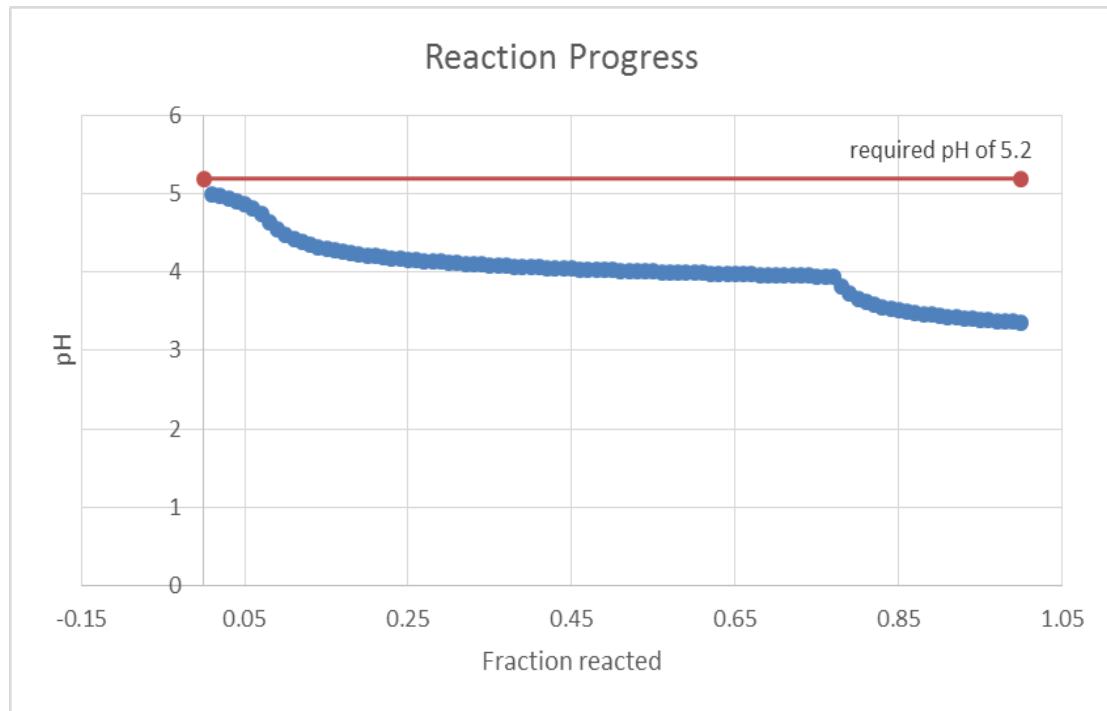

Figure 48: Reaction of product to react vs starting pH (NaOH)

5.9.3.3.1 Geochemical Analysis of dominant constituent in reaction (NaOH)

The major ions causing non-linear cumulative raises in pH during the reaction with NaOH are Al+ and K+ (Figure 46). With the addition of limestone (NaOH), the dissolved Na concentrations increase slightly throughout the reaction. The concentrations of K^+ decrease until a reaction fraction of 0.25 (pH ±4), where K+ concentrations increase slightly until a spike at reaction fraction 0.85 (pH ±4.3). The Al^+ concentration in solution decreases slightly until reaction fraction of 0.80 (pH ±4), after which it drastically decreases.
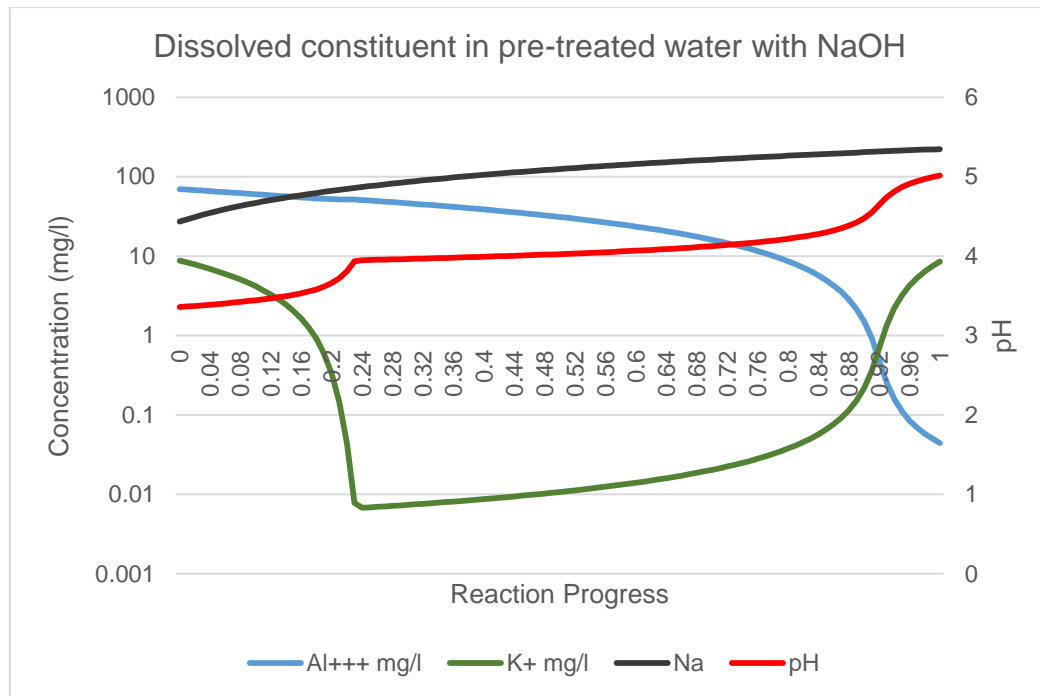
Figure 49: Identification of constituents causing non-linear cumulative raise in pH during product reaction.

5.9.3.3.2   NaOH Volumes and Costs

To estimate the volume of pre-treatment NaOH needed to treat pH, the weight of NaOH required for each pH jump was calculated. An early stage and a late stage curve (Figure 50) trendline equation was developed to factor in the influencing hydrochemistry constituents Al^+ and K^+. These separate equations allowed for a more accurate calculation on the amount of product needed per year.
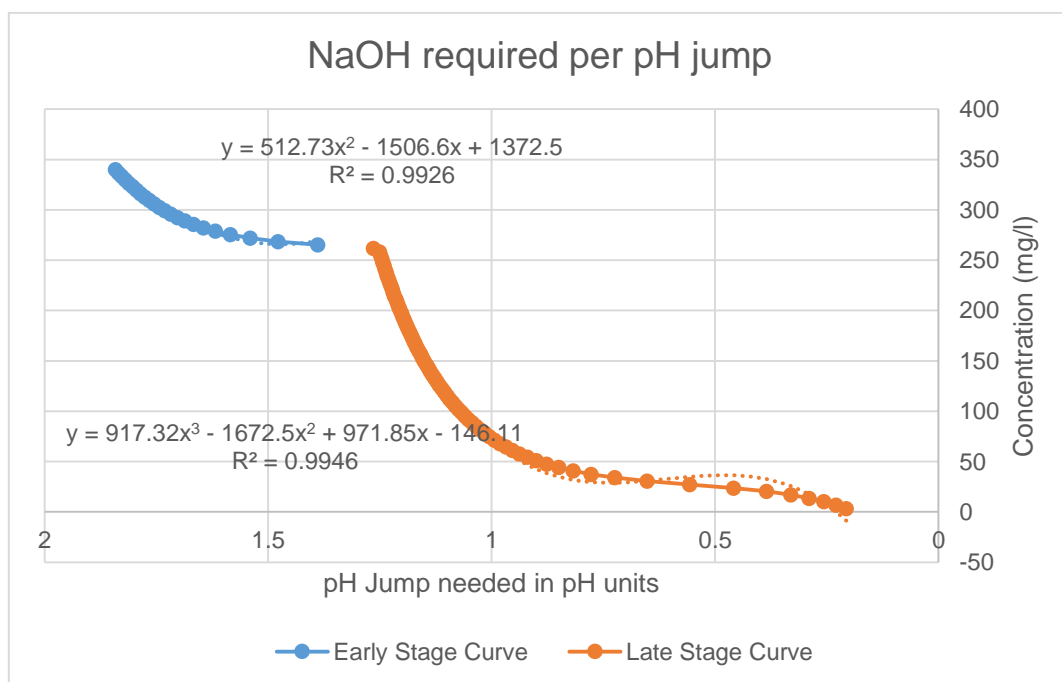
Figure 50: NaOH required per pH Jump

The calculated results show an estimated total cost of R 431 048 919-00 (assuming a limestone price of R13 500-00/ ton) with a maximum treatment cost in year one of approximately R 25 076 882-00 and a minimum approximate cost in year 50 of R661 487-00, with an annual decrease of approximately a R49 000/year for the first 50 years.
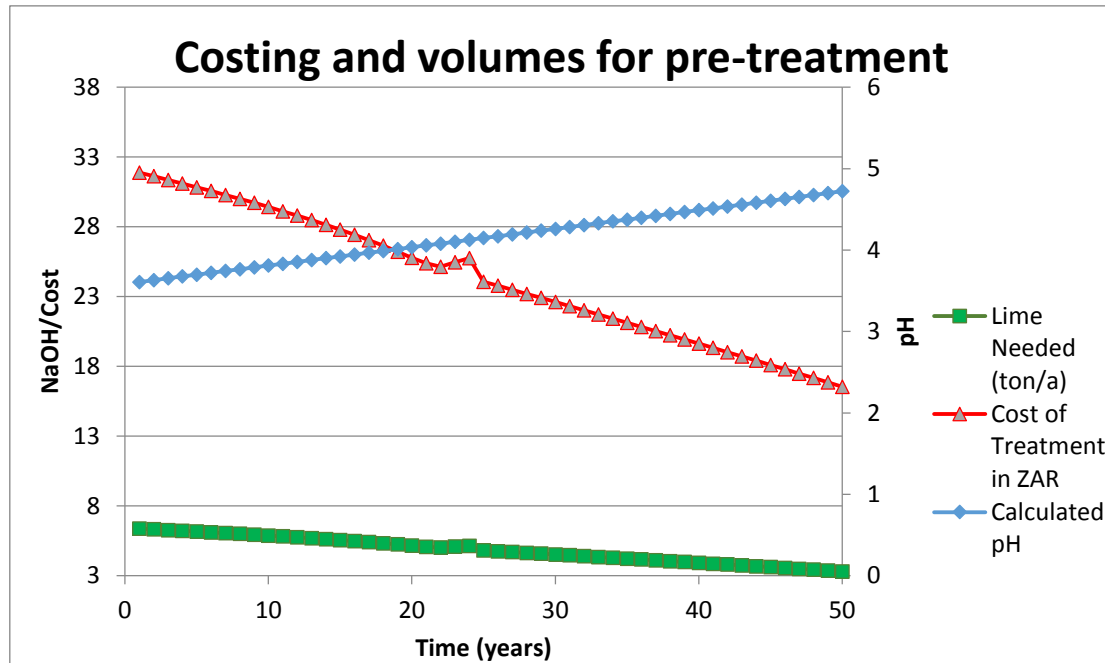
Figure 51: Relationship between pH, NaOH needed, NaOH costs, and number of years

5.9.3.3.3 Summary of Results: NaOH volumes and Cost

Influencing Chemistry	Mineral Precipitation	pH Jump	Total NaOH Required in tones	Formula	Cost of NaOH per tonne (May 2018)	Total Lime Cost (50 years)
Al, K, Na	Alunite	3.6 to 5.2	31 929	$Y = 995.53\,(x^4) - 2151.2(x^3) + 1603.4\,(x^2) - 418.98\,(x) + 42716$ $R^2 = 0.9946$	R13 500 - 00	R431 048 919-00

5.10   Selection of Mine Water Treatment Method Based on Cost

Chemical	Application	Neutralisation Efficiency (% of Applied Alkali Used)	Quality (tons) for pre-treatment	Relative cost as of May 2018 quotations (R/t)	Cost per year (R)
Lime, Ca(OH)₂	Powders	90	4425	R 2433-00	R 10 700 537 -00
Limestone, CaCO₃	Powders	30 – 50	3901	R 3285-00	R 9 770 686 -00
Sodium Hydroxide, NaOH	Flakes	100	2953	R 13 500-00	R 431 048 919 -00

6. DISCUSSION

The characterisation of mine water discharge quantity and quality is essential in the selection of a specific treatment method. Most treatment methods, however, require a form of pre-treatment.

Mine water Quantity Assessment

When models are used to calculate estimated predictions for the future, regardless the number of years, it must be understood that the processes underlying the model characteristics are not likely to represent accurate real life conditions due to changes in climate, erosion or sedimentation, anthropogenic factors, subsistence, or any other factors that are not identified (Nordstrom & Nicholson, 2017). To limit these influencing factors, all models (numerical and geochemical) must be updated with time, as new monitoring data is made available, and new information is described in the hydrogeological and hydrochemical systems.

The key factors in developing the numerical and geochemical models were to match the models with the current objective, understanding and capturing the site-specific hydrogeological conditions, model calibration, and identifying the assumptions and limitations for each model.

The numerical model was developed to evaluate the groundwater resource and aquifer system and to assess the impacts of Mine X on the groundwater environment (Qui, et al., 2015). The geological and hydrogeological input factors were calculated by reviewing drilling logs and aquifer parameter data. This allowed for a better representation of reality. Flow vectors and estimated flow volumes could be calculated and the impact of the open cast mine on the aquifer system could be determined. The numerical model using GMS developed for the open cast mine was compared to an analytical model developed by Marineli & Niccoli, 2000 to see how the results of these two models compare to one another. The numerical model is representative of the conditions on site taking data availability limitations into account as the results it calculated are reproducible using two different modelling methods. This is a quality check that can be implemented on all sites. An analytical model is a mathematical expression, taking limited site characteristics into account. Therefore, the further apart the results of these two models are from one another, the wider range in error is considered.

The analytical model results calculated an estimated water inflow of 684 m³/d into the pit while the numerical model calculated an inflow of water 658 m³/d into the pit. This deference in inflow volumes are considered minute in groundwater terms with a mere difference of 26 m³/d. Therefore these models are said to correspond well with one another.

The numerical model indicated that borehole SBH10 situated up-gradient and west of the opencast pit, will be affected by a drawdown of about 6 m if the mine is entirely dewatered. Base flow to the non-perennial tributary to the Wilge River is most likely to be affected by drawdown to the southeast of the mine. These groundwater calculations are based on the fact that the all areas in the mine are dewatered simultaneously. However, as the whole opencast will not be open at any one time, this is likely to be an overestimate.

The constructed numerical groundwater flow model was not only to estimate flow volumes,, but also to calculate groundwater flow directions, potential future decant points, discharge volumes and plume migration. The numerical model estimated a decant volume of approximately 6ℓ/sec which requires active treatment (Geller & Schultze). It is calculated that the decanting will occur approximately 11 years post-mining (after mining operations have ceased and the open cast pit has been backfilled).

To reduce the impacts of decant, mitigation measure such as cap and cover can be put in place. However, with set advantage, disadvantages such as changes in pit hydrochemistry may occur, resulting in influences in the time of sources depletion, and ultimately resulting in extending treatment time and potential greater costs. Therefore, a trade of will have to be made between treatment time and rehabilitation method.

The aim of rehabilitation during and post-mining is promote decant treatment to an acceptable level after which rehabilitation measures to minimize decant quantity can be implemented. Therefore if protocol mitigation measures and rehabilitations methods are put in place and adhered to, decanting volume may be drastically decrease or even avoided altogether. This could have huge cost saving implications in the future.

Mine water Quality

A decant of approximately 6ℓ/s has the potential to occur post-mining at Mine X, resulting in the discharge of contaminated mine water to the environment. This can lead to various negative environmental effects if left untreated. As mine water chemistry is a dynamic system, a dynamic quantification for its pre-treatment may be required for the site.

Additionally to decant volumes, specified dynamic concentrations were incorporated into the numerical model. The model calculated that the contamination plume will detach from the pit after approximately 49 years post-mining. Following plume detachment and the initial flush, groundwater from the surrounding aquifer will continue to enter the backfilled pit. As the source term is depleted at this stage no further notable contamination is expected and the entire hydrogeological system will reach a new hydrochemical equilibrium. Therefore, the detached contamination plume will shrink over time and the contamination concentration in the aquifer will reduce over time.

Post mining, the backfilled opencast mine will have a higher porosity and hydraulic conductivity that generally exceeds that of the surrounding hydrogeological aquifer system. This results in faster flow velocities and could result in flooding. This process results in rapid weathering of secondary minerals and the formation of AMD, until the backfilled mine is fully flooded. Once fully flooded the AMD process de-accelerates until steady state conditions are reached. The addition of water to the opencast mine through recharge and groundwater inflows, will cause the potential contaminants to dilute. This process will continue until the sources of contamination are depleted or chemical equilibrium is reached. Other processes such as dispersion and advection into the surrounding aquifer will occur which can generate a contamination plume migration which may impact sensitive receptors.

An RO plant at the current mine has capacity for additional treatment volumes. However, a pre-treatment pH of 5.2 is required. At this pH, neither ferric iron nor aluminium as with many other metals is soluble in large quantities (Geller, et al., 2013). From literature and from numerical model concentrations change over time.

Sulphate is one of the main controlling factors in mine water chemistry for coal mines in Mpumalanga (Huisamen, 2017). The statistical analysis and geochemical model positively identifies that there is a relationship between SO_4 concentrations and a corresponding pH. pH and SO_4 are dependent variables in the opencast pit and are influenced by the amount of sulphide materials present in the backfilled opencast mine. Other influencing parameters could include carbonate mineral phases. However, due to the absence of alkalinity in the mine water samples, it was assumed that carbonates are not present or depleted. Therefore, pH and SO_4

in decant water remain independent due to the absence of sulphides and the fact that sulphate is not a pH dependant species.

The results of the statistical model indicated a positive relationship between pH and SO_4, and identified that possible statistical relationships between other constituents, are also likely. Therefore metals concentrations can also be calculated using these relationships, and effectively targeted and treated by passive and/or active methods.

Because SO_4 concentrations are dynamic and the associated pH follows a similar trend (statistical model), dynamic addition of pre-treatment chemical volumes are required. This implies that the annual review of pre-treatment cost is imperative.

Based on the evaluation of pre-treatment products in the results presented, limestone is most effective based on cost for pre-treatment of mine water decant from Mine X that will be processed by an RO plant.

Limestone is one of the most effective ways to treat mine water decant. Limestone has a low material cost and is the safest and easiest to handle for the treatment of mine water. However, the successful application thereof is sometimes limited due to the low solubility and tendency to form an external coating that limits the reactive chemical potential of limestone. For the purpose of this investigation, we assumed that all product will react in solution. Therefore, a higher grade quality of sandstone should be obtained (150um fine). The pricing in this dissertation relates to such high graded fine sandstone. The applications methods used include diversion wells and trenching, and can be applied by man or machine, with no intense equipment needed.

The selection of the most efficient pre-treatment product for the neutralization of pH is site specific and project related, with costing, time, and end goal in mind.

7. CONCLUSION

Water is a critical resource in South Africa and may pose huge risks to mining projects. Therefore the management and protection of the water resources are imperative.

This study focused on an opencast Colliery called Mine X situated in the Mpumalanga coal field. Mine X wishes to extend their mining operations which will include an additional opencast pit that will extend approximately 80 hectares. Part of the mine water management strategy was to conduct a hydrogeological impact study and to determine suitable mitigation measures to address future calculated impacts.

Groundwater quantity impacts are expected during the operation of the mine. Additionally, decant of approximately 6ℓ/s has the potential to occur post-mining, resulting in the discharge of contaminated mine water towards the east of the proposed site. This can lead to various negative environmental effects if left untreated. The operational Colliery current treats contaminated groundwater through the use of a Reverse Osmosis (RO) Plant on site. High concentrations of contaminant feed waters cause rapid scaling and fouling of membranes, and are highly corrosive which leads to higher anticipated operational and capital cost.

The RO plant was designed to treat a feed water flow of 15ℓ/s, and currently runs at a maximum 8ℓ/s. High concentrations of contaminant feed waters cause rapid scaling and fouling of membranes, and are highly corrosive which leads to higher anticipated operational and capital cost. The recommendation put forward by the current operational Colliery was to pre-treat feed water to a minimum pH of 5.2.

Therefore, currently there is a surplus by the RO Plant to receive excess feed water. The predicted 6ℓ/s decant from Mine X will therefore be re-routed to the RO Plant were water will be treated.

However, before the decant water could be transferred to the Ro plant, it had to go through a pH adjustment to the required pH of 5.2. The geochemical-statistical model allowed for the prediction of water chemistry over time, and was applied to different pre-treatment scenarios to predict the water quality pH along with a cost-benefit analysis of each of the pre-treatment methods.

Three treatment products were selected based on industry uses and chemical availability. Lime, Limestone and NaOH was selected. Generally, the powder dispersion application is preferred to the slurry in these instances as it does not require complicated design and maintenance.

Based on the evaluation of pre-treatment products in the results presented, limestone is most effective based on cost for pre-treatment of mine water decant from Mine X that will be processed by an R.O. plant. One should note that this selection is based on a specific choice in product which will lead to specific costing that is dynamic over time.

Concentrations of contaminants in drainage water are controlled by site specific parameters. Therefore, the application of these formulas are site specific, and will have to be recalculated for each unique site. However, the same method can be applied to predict a dynamic cost.

8. RECOMMENDATIONS

Following the above discussion, the following is recommended:

During Mining:

- Water quantity and quality data should be collected on an ongoing and regular basis during mining operations to recalibrate and update the mine water numerical and geochemical models.

- Identify where possible, areas of the mine that will result in non-contaminated water and ensure that it routed away from any contaminate water.

- Reduce where possible, the size of un-rehabilitated areas (pit, waste dumps) that produce contaminated runoff. Rehabilitation should be designed to minimise/ eliminate water ponding. On-going rehabilitation as mining operations progress will be an advantage.

- Monitoring of contaminated effluent storage facilities, to manage the risk of leakages and spillages.

- The capacity to pumping water out of the pit into storage dams during mining operations is essential. This will minimise the contact that inflow water will have with potential contamination sources (rock slopes, rock dumps), and prevents the possible deterioration of water quality.

- Where possible, separate acid generating material and non-acid generating material, as characterised by geochemical sampling and analyses.

- The presence of WAD in borehole PBH2A must be investigated and possible impacts thereof quantified.

Post Mining:

Following is recommended upon the closure of the mine

- Water quantity and quality data should be collected on an ongoing and regular basis once mine has stopped. These data sets will be used to recalibrate and update the mine water numerical and geochemical models.

- During the backfill process, the acid producing material should be placed as low in the pits as possible, followed by the non-acid generating material. Once mine has been decommissioned, rapid flooding should take place into the pit to reduce the initial flush time period, and to expose the acid generating minerals too as little oxygen as possible. Thereafter, the recycling and treatment of mine water decant can commence.

- Contaminated mine water discharge can be captured using a set of lime trenches (pre-treatment) which should be constructed downstream of the mine to the southeast and should be large enough to capture inflows of approximately 6 ℓ/s flows.

- The pre-treatment of approximately 240 mg/l of lime in a lime trench before the treatment of reverse osmosis.

- When updating the numerical and geochemical models, update the volume of lime calculations needed to predict dynamic costing.

Suggestions for further research:

- Kinetic geochemical modelling can be applied for a more representative definition of the hydrochemical system in the pre-treatment process.

- Kinetic geochemical modelling can be performed to provide further insights to process happening within the backfilled opencast pit to obtain sight specific mine water evolution (This data can be used for the calibration of the hydrochemistry observations).

- Define relationships between other contaminants of concern in different mine settings.

- Further investigate the Influences of hydrogeological structures with relation to the mine water chemistry.

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