LONG-TERM IMPACT OF INTERMINE FLOW FROM COLLIERIES IN THE MPUMALANGA COALFIELDS

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LONG-TERM IMPACT OF INTERMINE FLOW FROM COLLIERIES IN THE MPUMALANGA COALFIELDS

This project was motivated as a result of work initiated earlier by the mining industry, in which a smaller area south of Witbank has been investigated.

The Steering Committee responsible for this project consisted of the following persons:

Mr. H.M. du Plessis (Chairman)	Water Research Commission
Mrs. C.M. Smit (Secretary)	Water Research Commission
Mr. K. Pietersen	Water Research Commission
Mrs. B. Postma	Department of Water Affairs and Forestry
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Mr. T.H. Coleman	Wates, Meiring and Barnard
Mr. W. Pulles	Pulles, Howard and De Lange
Mr. D. du Plooy	Eskom

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The project was only possible with the co-operation of many individuals and institutions. The authors wish to record their sincere thanks to the numerous mines that have willingly contributed data and information to this project. Specific reference is also made in the different chapters to contributions by companies.

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EXECUTIVE SUMMARY

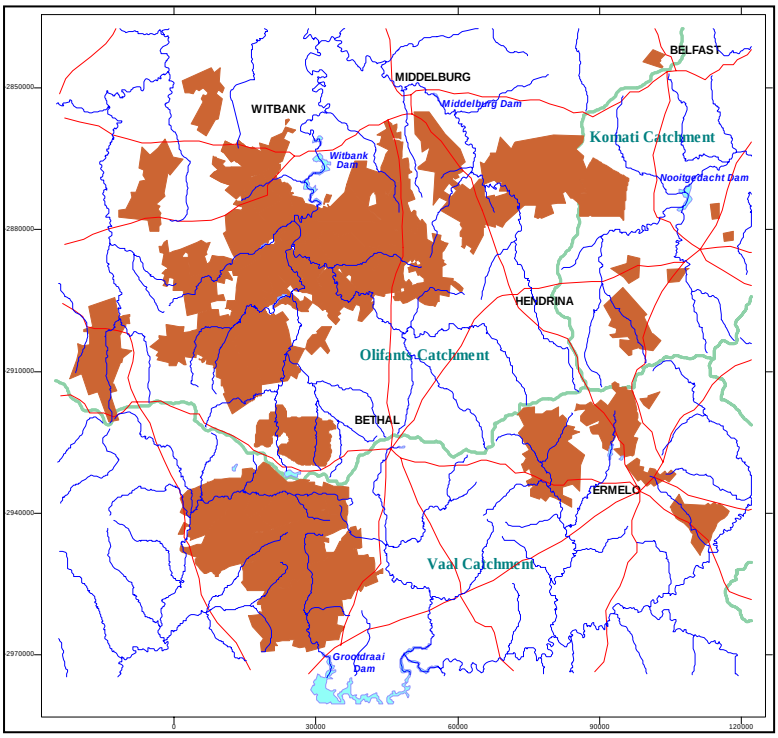
1 INTRODUCTION AND SCOPE OF INVESTIGATION

Mines fill up with water after closure. As a result, hydraulic gradients develop between them and different hydraulic water pressures are exerted onto peripheral areas or compartments within mines. This results in water flow between mines, or onto the surface. This flow is referred to as intermine flow. Intermine flow as a concept includes the quantity and quality of the water.

The significance of this project lies in delivering the following products:

- The establishment of a Geographic Information System (GIS), detailing the various aspects around intermine flow for the Mpumalanga Coalfields.
- The identification and discussion of management options to minimise the long-term impact of intermine flow on the environment.
- The provision of this information to the industry and government for the assessment of future liabilities and impacts.

A diagram showing the mine lease areas and extent of the area investigated is included below. The total area investigated constitutes 26 000 km^2. This covers all of the collieries in Mpumalanga from where information could be obtained.



1.1 AIMS OF THE INVESTIGATION

The aims of the investigation were:

- The establishment of a GIS for all the collieries in the Mpumalanga area, showing mined-out areas, future areas to be mined and mining methods.
- The identification of critical areas where intermine flow is likely and quantification of flows through field measurement and modelling.
- The identification of seepage and decant positions where water from mines will impact on groundwater and surface water, and the quantification of contributions through field investigation and hydrodynamic modelling.
- The identification and selection of management options to minimise the long-term impact of intermine flow on groundwater and surface water resources.

1.2 APPROACH TO THE RESEARCH

Information available in the South African coal-mining industry suggests that mines fill up with water and decant after closure. This usually occurs within 10 years. At the more isolated collieries, rebound of the water level may take up to 50 years. Apart from the fact that mine water is saline, low pH-values may also be encountered.

Detailed, site-specific investigations are necessary to facilitate predictions of long-term water- and salt balances. The following main issues required investigation:

- Natural groundwater flow paths.
- Interconnectivity between adjacent mining operations.
- Water migration routes.
- Decant and seepage points from the mines.
- The impact of intermine flow on surface water in the various catchments.

The following actions were required:

- Acquisition of cartographic information from all the mines.
- Consolidation of this information into a GIS.
- Identification of potential ground- and mine water flow paths.
- Identification of critical areas where intermine flow would occur.
- Investigation of these areas with assistance from the mines.
- Identification of decant and seepage points.
- Processing of existing water quality data, supplemented by additional sampling and chemical analyses.
- Prediction of flow rates and salt-load transfer using numerical models.
- Investigation of management options to minimise the long-term impact on unpolluted water systems.

2 DATA ACQUISITION

Relevant data and detailed mine plans were obtained from individual collieries, regional mine offices, mine head offices, the Department of Minerals and Energy (DME) and the Department of Water Affairs and Forestry (DWA&F). Some of these data are confidential and are not reproduced in this report. These constitute, in some instances, detailed future mine plans.

Information was requested from the mines in hard copy and digital format. This included:

- Mine lease areas.
- Surface topography.
- Surface features including watercourses, evaporation dams, dumps and areas of subsidence.
- Thickness of the weathered zone.
- Extent of existing and future mining.
- Floor contour plans of all major coal seams.
- Geological structures including dykes, sills and faults.
- Elevations of water levels in boreholes in mined and unmined areas.
- Quantities and qualities of water.
- Abstraction and disposal methods of mine water.

Despite most of this information being readily available at the mines, it took the best part of 18 months to acquire this. Reasons for this vary, but in general, two stand out:

- The provision of information was not a high priority at many of the mines.
- Clearance of information through head offices takes time.

3 GEOGRAPHIC INFORMATION SYSTEM

The WISH (Windows Interpretation System for Hydrogeologists) software package was selected for visualisation and interpretation of the data. Reasons for this are as follows:

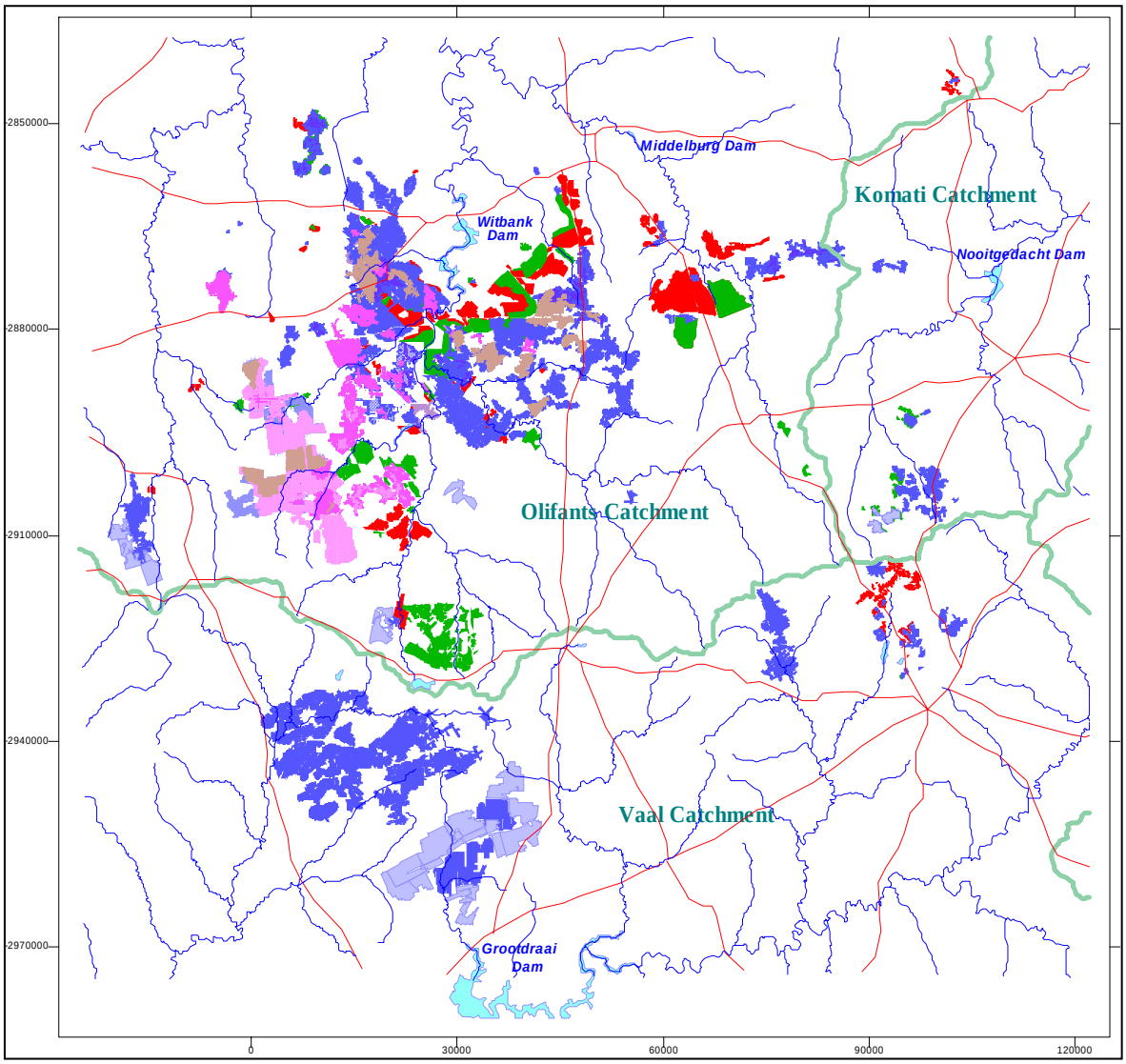
- WISH is easy to use, sponsored in part of its development by the Water Research Commission (WRC), and available from the Institute for Groundwater Studies (IGS).
- It consists of a map drafting and display facility. Maps may be integrated from other applications that are commonly in use at the collieries.
- Datasets from Microsoft Excel may be superimposed onto the maps. When it comes to relevant data processing, the processing power of WISH is unsurpassed by other software.

4 INTERMINE FLOW

Mining has been on all five coal seams in the area. Statistics on mined areas are provided below:

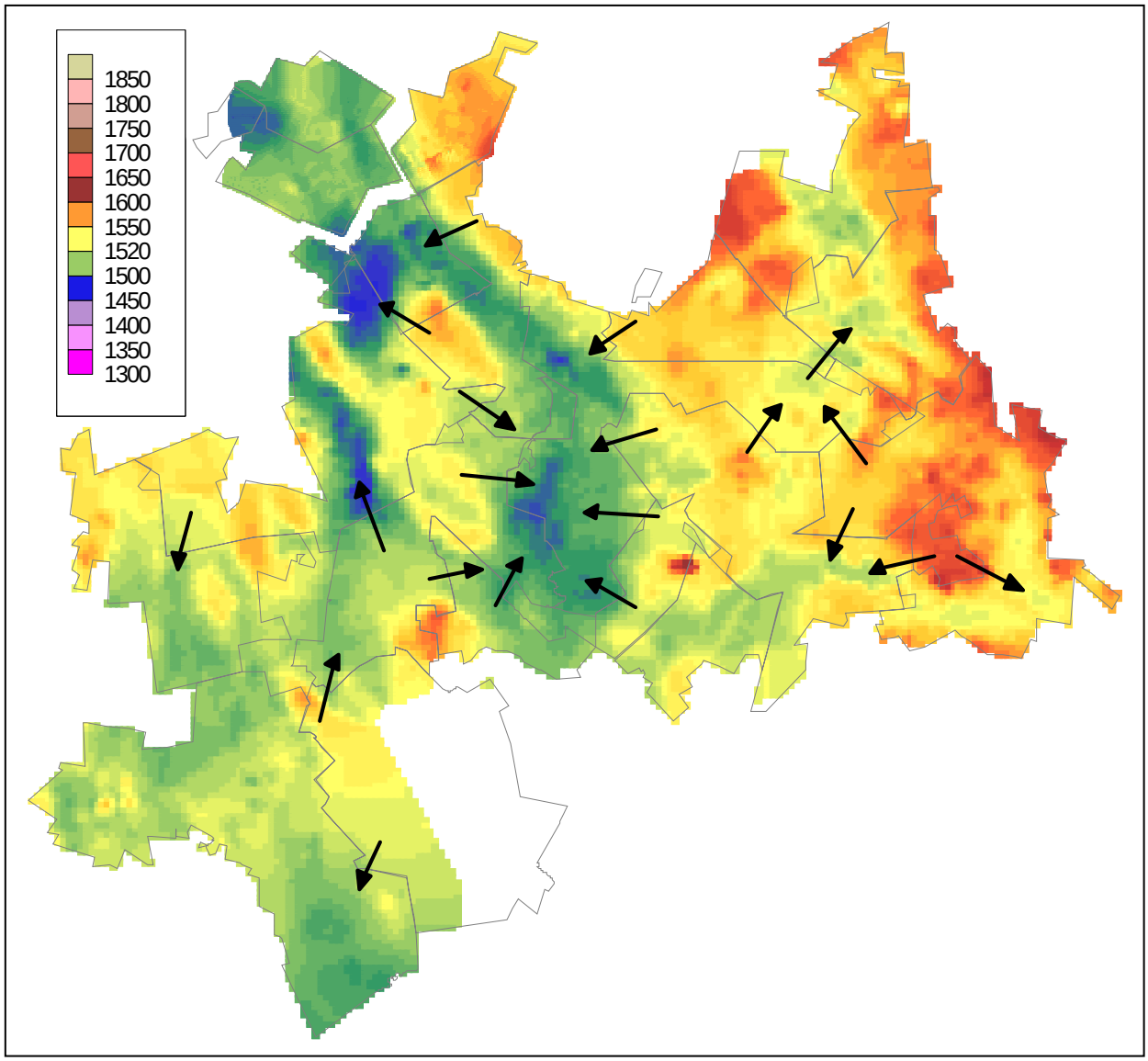
Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	coal mined (Mt)	Future volume coal to be mined (Mm3)	Future coal to be mined (Mt)	Total planned production (Mt)	% from each horison or method
U/G 5 Seam	7842	7051	14893	2.5	65.00%	127	85	115	76	161	5.4
U/G 4 Seam	13485	2833	16318	3	65.00%	263	175	55	37	212	7.1
U/G 2 Seam	98549	39695	138244	3	65.00%	1922	1281	774	516	1797	60.0
U/G 1 Seam	2525	0	2525	2.5	60.00%	38	25	0	0	25	0.8
Opencast	23557	14480	38037	3.5	90.00%	742	495	456	304	799	26.7
Totals	145958	64059	210017			3092	2061	1400	933	2995	

As the statistics show, not all mining extends over the whole of the coalfields. An idea of the mining activities is presented in the map below.

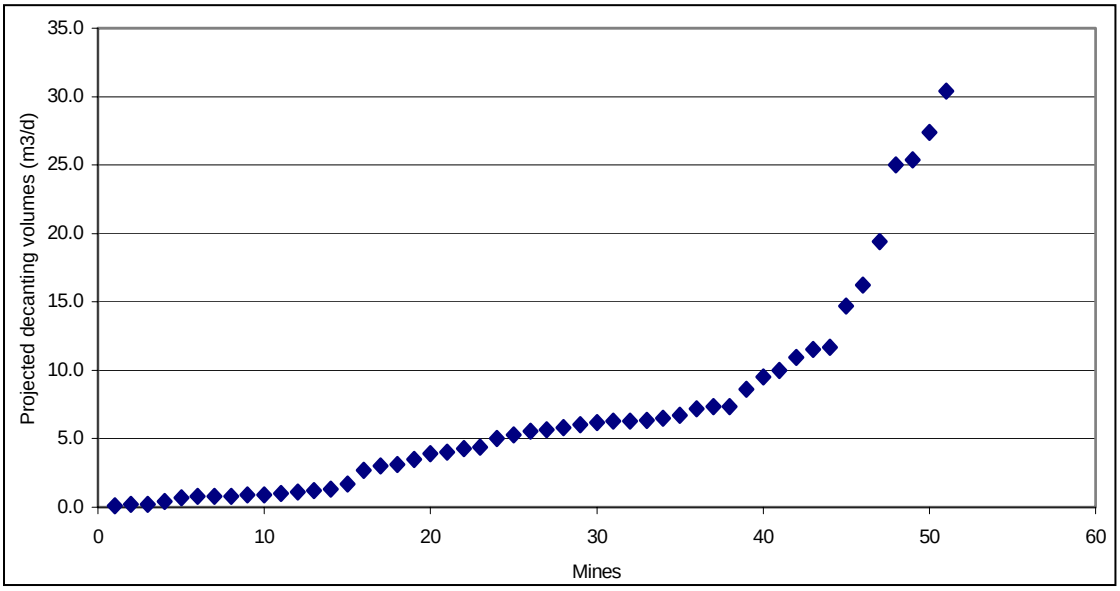


Sufficient connectivity (permeability and/or connections) exists between mines and the surface to allow interflow of mine water. These connections are in the form of opencast mining, shafts, prospect boreholes and subsidence structures.

Interflow of water on coal seam levels is also possible in most of the collieries due to the extensive nature of mining, particularly in the Witbank and Secunda areas. Pathways in mines dictate flow of underground water, rather than natural flow paths. Many critical areas of potential intermine flow (see diagram below) have been identified, where significant quantities of mine water will transfer from one mine to the other. Colour codes in this diagram refer to the No. 2 Coal Seam elevations.



The combined impact in terms of anticipated water that will be available for interflow or to decant from the various mines is in the order of 360 ML/d with a sulphate load of 660 t/d. The range of quantities and qualities for individual mines will vary significantly, depending mainly on the mining method employed. Many other site-specific factors also play a role. Typical quantities of water expected from the individual collieries, arranged from the least to the highest, are shown in the diagram below.



5 MANAGEMENT OPTIONS

Management options to cope with intermine flow are applicable on either a catchment or mine basis. Those on a catchment basis are:

- The reduction of the number of decanting points through the interconnection of mines.
- The control of decanting positions through interconnection of mine workings.
- The combined treatment of decanting mine water at convenient locations.

On a mine basis, the following actions will markedly reduce the volume of water or the amount of salt to be discharged:

- Design long-term water management schemes taking cognisance of neighbouring mining activities.
- Design the mine lay-out to retain as much of the mine water as possible in the underground workings whilst mining.
- Investigate coal barrier characteristics and design coal extraction accordingly.
- Minimise water ingress into mines to reduce water volumes, if required.
- Mix mine water of various qualities to achieve the best possible quality before flood discharge.
- Minimise salt loads by:
    - Flooding mine workings as soon as possible.
    - Flushing flooded mines.
    - Utilising the natural neutralisation potential of the coal and rock.

5.1 REGIONAL MANAGEMENT OPTIONS

Several major paleo-drainage channels are present on the No. 2 Coal Seam floor. If no barriers existed, much of the water on this coal seam horizon would drain towards the paleo-channels, where it would dam up. By providing for controlled release systems through barriers, much of the mine water could therefore be drained naturally towards the centrally located areas, from where abstraction and treatment can be done in a controlled fashion. An excellent example of an area where this will be possible, is the central Witbank Coalfield, where water from Tweefontein, Tavistock, Rietspruit, Douglas, Middelburg South, Goedehoop, New Clydesdale, TNC and SACE mines could centrally be controlled and, if required, treated before flood discharged. This would typically involve some 100 ML/d.

Another example is the diversion of much of the decant water from New Denmark Colliery into the Waterval River. This would contribute an additional 15 ML/d to the 90 ML/d anticipated from the Secunda mines. Many other such examples, though not on the same scale, can be found on a local basis. It is obvious that significant advances in water quality management can be made by planning the possible transfer of salt through the underground workings, to centralised treatment facilities. Here acid water, for instance, could be blended with alkaline water. Settlement of iron from the mine water could be allowed, before release of the water during flood events.

5.2 LOCALISED WATER MANAGEMENT OPTIONS

5.2.1 Neighbouring mining activities

None of the mines visited during this investigation had sufficiently knowledge of mining activities in neighbouring mines. This is a serious shortcoming in the overall planning of intermine flow management. Much more emphasis should be placed on information exchange with adjacent mines. Joint water management committees should be established. This should be extended to the regional level, where corporate houses and the government should contemplate management strategies for catchments.

5.2.2 Containment of mine water to flood areas and utilise the neutralising potential

Mine water has historically been pumped from active mine workings to allow unhindered coal production. Almost no consideration has been given to the best management strategy for water while mining. Yet, this is simple: Mine from deep areas to shallow areas and leave water behind in the mined-out workings. This strategy has, for the past few years, been applied in several of the larger collieries with significant success. The advantage of this mining sequence does not only lie in managing water volumes, but also in water quality management. Mined-out areas are flooded, thus excluding oxygen. Furthermore, the natural alkalinity of water is not flushed from the rock. This counteracts acidification in these mines.

5.2.3 Design of barrier pillars

Barrier pillars between mines or compartments should be designed for the specific purpose of keeping water from transgressing through it; or to release water at a specific rate through controlled flow mechanisms. Only one of the mines has ever done permeability testing in a barrier pillar. It is essential that barrier pillars be tested and that monitoring systems be installed before intermine flow takes place in an uncontrolled fashion. Issues such as liability and accountability can only be addressed once this information is available.

5.2.4 Minimising water volumes

The minimisation of water volumes in mines leads to high salt concentrations, but smaller volumes of water that need to be handled. Up to now, minimisation strategies have not been applied to a significant extent as a management tool. This is largely due to the vast scale of coal mining operations. It is almost impossible to change a mining strategy, for instance, from underground high extraction to bord-and-pillar, once mining has commenced. Other water minimisation strategies include the artificial flooding of closed mines improved rehabilitation of surface and the planting of trees. The conclusion is that water influx minimisation should be considered carefully against the option of active flushing.

5.2.5 Mixing of mine water

At most of the larger mines, the opportunity exists for mine water of different qualities to be mixed, thus improving the overall water quality. Typical benefits of doing this would lie in pH adjustment and iron precipitation. For the latter, retention of the mine water in a surface holding facility where aeration is possible, is necessary. Such a facility could also be used for quick release of the water during flood discharge. Very few other chemical benefits would be forthcoming from mine water mixing, because most of the constituents are undersaturated in this water.

5.2.6 Minimising salt loads

Several options for minimising salt loads from mines need to be exploited for future use. These typically are: (i) Flooding of mines as soon as possible after closure; (ii) Active flushing of flooded mines and (iii) Greater utilisation of the natural base potential in the coal and rock for acid water neutralisation. These are all concepts with great potential, but will necessitate a change in the direction of thinking by the controlling authorities. Active flushing would imply the controlled release of salt into a catchment, with the specific purpose of improving mine water quality to the extent that it will become a useable resource. Computer simulations have suggested that through active flushing, the sulphate concentration in a mine could be dropped by 50% every 20 – 40 years, depending on the rate of flushing. Considering that many of the mines will have holding capacities far exceeding that of the Witbank Dam, this is

a management option that, at the very least, should be pursued in a couple of the smaller mines, thus establishing a number of field trials.

6 AIMS AND ACCOMPLISHMENTS

The following is a discussion of the degree to which the aims of the project have been met.

The establishment of a Geographical Information System for all the collieries in the Mpumalanga area, showing mined-out areas, future areas to be mined and mining methods.

A geographic information system, using the WISH software package, was established. It was used throughout this investigation for the processing of the data. The complete dataset is available on CD-ROM as part of this report.

The identification of critical areas where intermine flow is likely and quantification of flows through field measurement and modelling.

The critical areas where intermine flow is likely and other areas not so critical, have been identified in this report. Modelling of flux across selected boundaries has been done; the results of which have been compared with actual field observations. The mines have been advised of these areas with the recommendation that further site-specific investigations are required.

The identification of seepage and decant positions where water from mines will impact on groundwater and surface water, and the quantification of contributions through field investigation and hydrodynamic modelling.

After intensive investigations, decanting positions have been identified for each of the mines from where information was obtained. This information is provided in the report. Field investigations and hydrodynamic modelling were carried out for a selection of sites.

The identification and selection of management options to minimise the long-term impact of intermine flow on groundwater and surface water resources.

Management options to minimise the long-term impact of intermine flow have been subdivided into two categories, namely regional and local. This selection of management options is backed by more than 20 years personal experience in the Mpumalanga coal industry. Far-fetched and futuristic ideas are not subscribed to, because of the requirement to find urgent solutions, applicable to real-life situations, on a regional scale.

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1 INTRODUCTION

Mines fill up with water after closure. As a result, hydraulic gradients develop between mines and different hydraulic water pressures are exerted onto peripheral areas of mines or compartments within mines. This results in water flow between mines or onto the surface. This flow is referred to as intermine flow. The concept of intermine flow includes both water quantity and quality.

Figure 1 shows the location of the study area in South Africa. The total area under investigation is in the order of 26 000 km^2. Figure 2 shows all the mines included in the study. All collieries where information was available, including both opencast and underground mining, are included in the investigation.

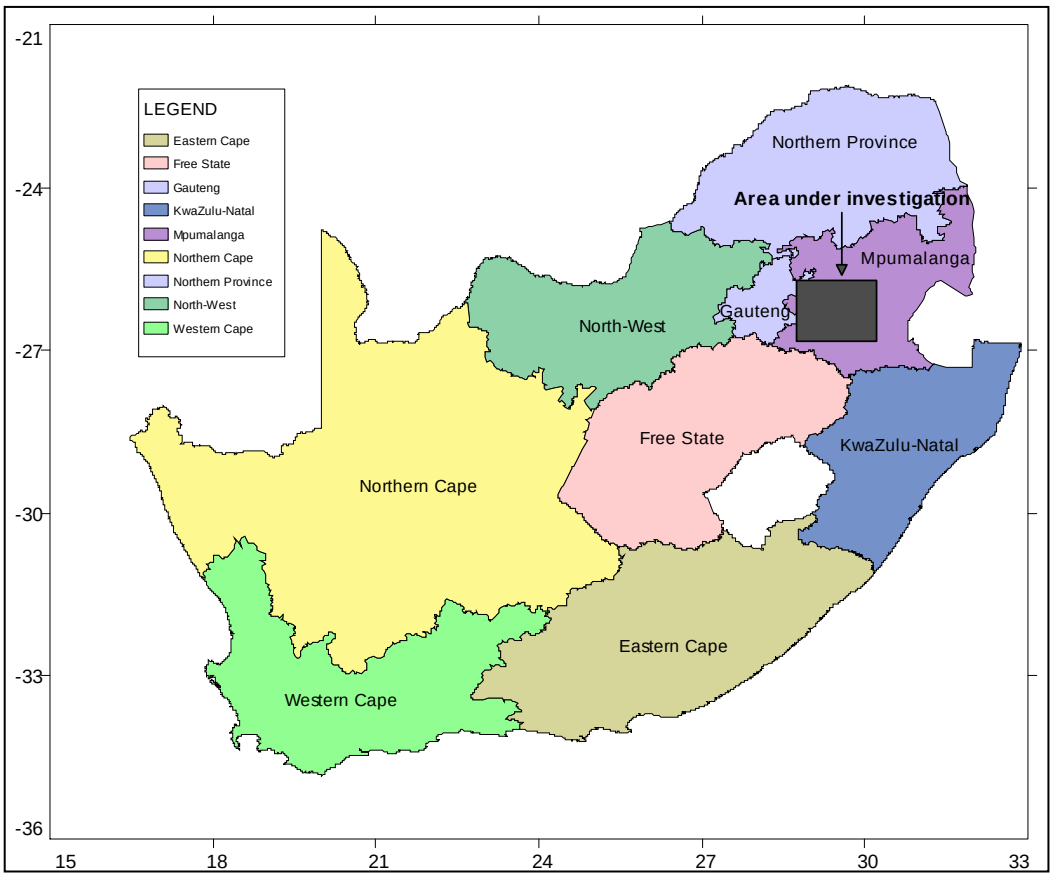


Figure 1. Locality plan of the area under investigation.

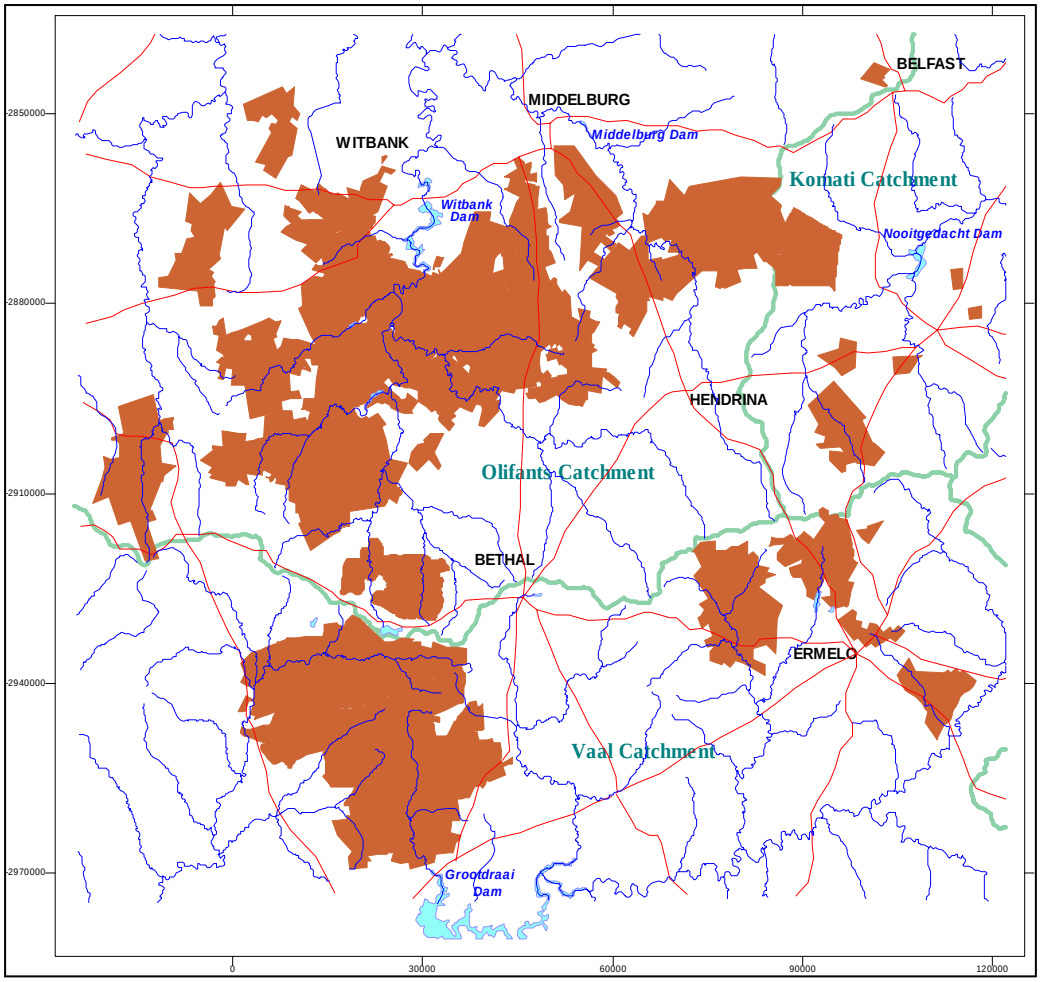


Figure 2. Plan showing mines included in the study.

1.1 SCOPE OF THE INVESTIGATION

This investigation was preceded by a similar investigation, initiated by the mining industry in a smaller area south of Witbank. It was concluded from the initial investigation that the problem for intermine water flow is real and that planning on a catchment level rather that for individual mines was necessary. Following from this, the scope of the current investigation was defined as follows:

- The establishment of a Geographical Information System (GIS) for all the collieries in the Mpumalanga area, showing mined-out areas, future areas to be mined and mining methods.
- The identification of critical areas where intermine flow is likely, and quantification of flows through field measurement and modelling.
- The identification of seepage and decant positions where water from mines will impact on groundwater and surface water, and the quantification of contributions through field investigation and hydrodynamic modelling.
- The identification and selection of management options to minimise the long-term impact of intermine flow on groundwater and surface water resources.

1.2 APPROACH TO THE RESEARCH

Information available in the South African coal-mining industry suggests that mines fill up with water and decant after closure. This usually occurs within 10 years. At the more isolated collieries, rebound of the water level may take up to 50 years. Apart from the fact that mine water is saline, low pH-values may also be encountered (Hodgson and Krantz, 1998).

Detailed, site-specific investigations are necessary to facilitate predictions of long-term water- and salt balances. The following main issues need to be investigated:

- Natural groundwater flow paths.
- Interconnectivity between adjacent mining operations.
- Groundwater migration routes.
- Decant and seepage points from the mines.
- The impact of these issues on surface water in the various catchments.

The investigation for the Mpumalanga Coalfield required the following actions:

GIS - Chapter 5	Acquisition of cartographic information from all the mines.
	Consolidation of this information into a GIS.
Intermine flow - Chapters 6 - 8	Identification of potential ground- and mine water flow paths.
	Identification of critical areas where intermine flow could occur.
	Investigation of these areas with assistance from the mines.
Decant and flow rate - Chapters 6 - 7	Identification of decant and seepage points.
	Processing of existing water quality data, supplemented by additional sampling and chemical analyses.
	Prediction of flow rates and salt-load transfer using numerical models.
Management options - Chapter 9	Investigation of management options to minimise the long-term impact on unpolluted water systems.

2 BACKGROUND INFORMATION

The concern for mine-water interflow and its potential long-term impact on surface water within the Olifants Catchment can be anticipated from existing information. This exists in many forms, of which the following items probably stand out:

- The extent of mining.
- Topography and surface drainage.
- Historic relationship between mining and stream water quality.
- Historic trends in mine-water quality and quantity.
- Geology and geohydrology.
- Coal-floor contours.

2.1 EXTENT OF MINING

Figure 2 shows the extent of the mine lease areas and that of the area investigated. Although the map shows an area of 26 000 km^2, the mine lease areas cover only 4 250 km^2. The older mines lie in the area south and west of Witbank, where extensive mining has, and is still taking place. Outwards from this area are many newer mines that commenced mining during the past 30 years. Both underground and opencast mining methods are used.

2.2 TOPOGRAPHY AND SURFACE DRAINAGE

Surface drainage occurs to the north, east and south (Figure 3).

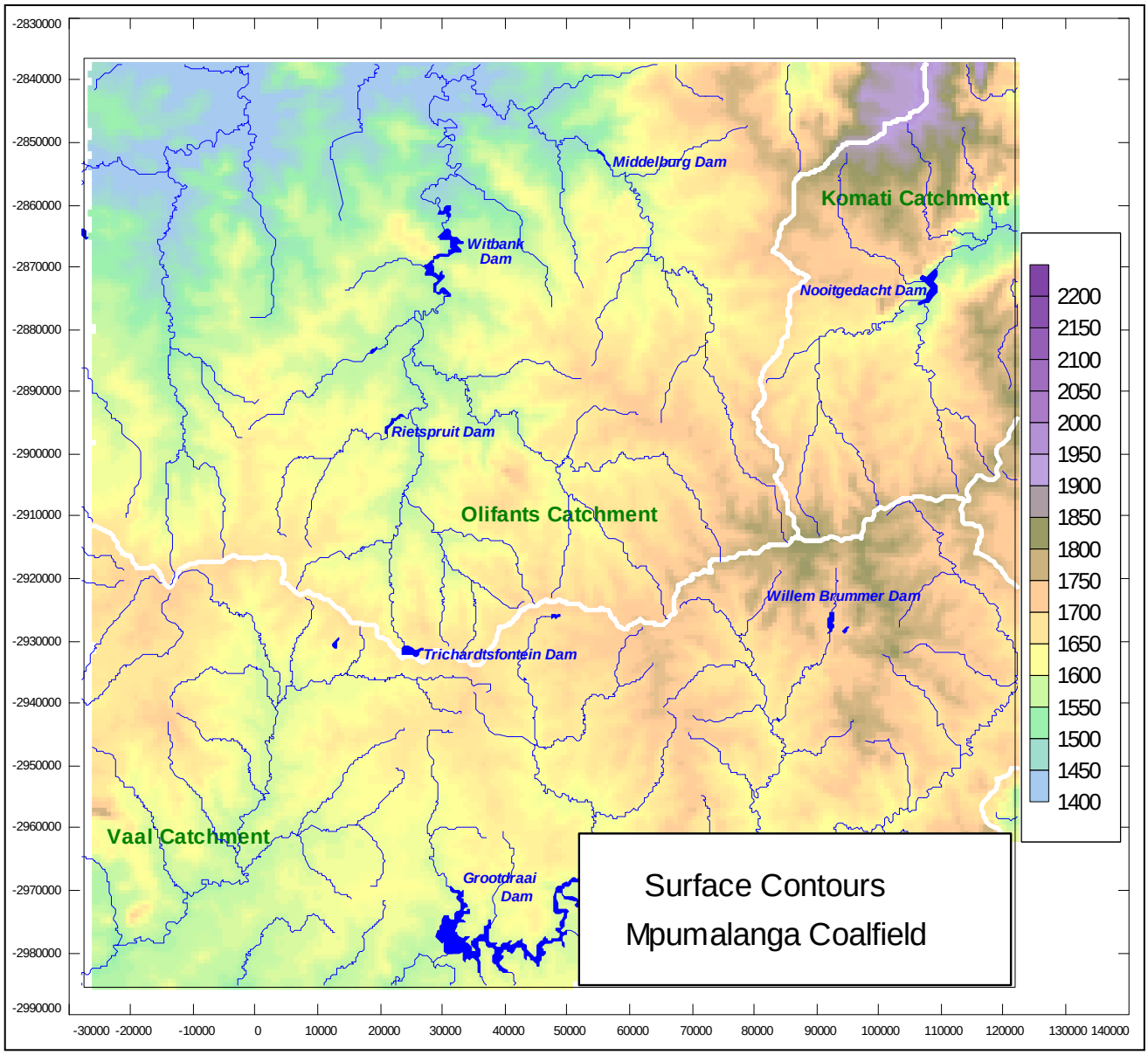


Figure 3. Surface topography, drainage systems and catchments of the Mpumalanga Coalfields.

The topography is of a gentle rolling nature. Steeper slopes are present at sandstone outcrops. Studies by the mining industry indicate that surface run-off for this area is in the order of 6 -10% of the annual rainfall of 710 mm, with 8% as an average. In terms of this study, the main concern is the proximity of the Olifants River and Witbank Dam to the mine workings. In the event of mine water spilling into the river, this could have a significant impact on the dam water quality, particularly in the dry season.

The southern area drains to the south and southwest, into the Grootdraai Dam followed by the Vaal Dam further downstream. The eastern catchment is that of the Komati River.

2.3 SURFACE WATER QUALITY

Stream water quality in the coalfields deteriorated over the past 20 years, due to seepage and discharge of mine water. The DWA&F monitors surface run-off water quality and quantity at many localities within the catchment (Figure 4).

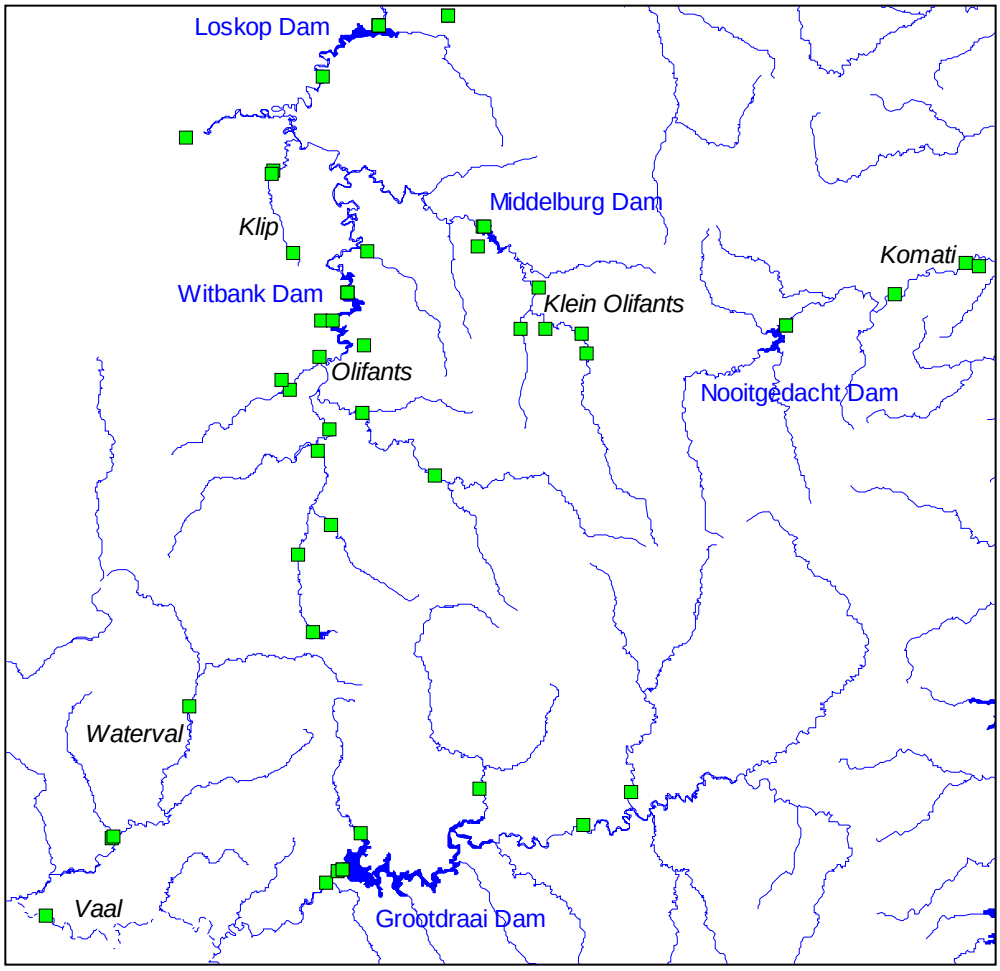


Figure 4. Regional surface water monitoring points of the DWA&F in the study area. Scale from left to right is 150 km (Names in black at streams indicate stream names).

Datasets of interest are those in the streams that lead away from mining area. These are typically to the north, i.e. the Klip Spruit, Witbank Dam in the Olifants River,

Spook Spruit and the Klein Olifants River at the Middelburg Dam. To the south are the Waterval River, Leeu Spruit and the Vaal River before it enters into the Grootdraai Dam (Figure 5).

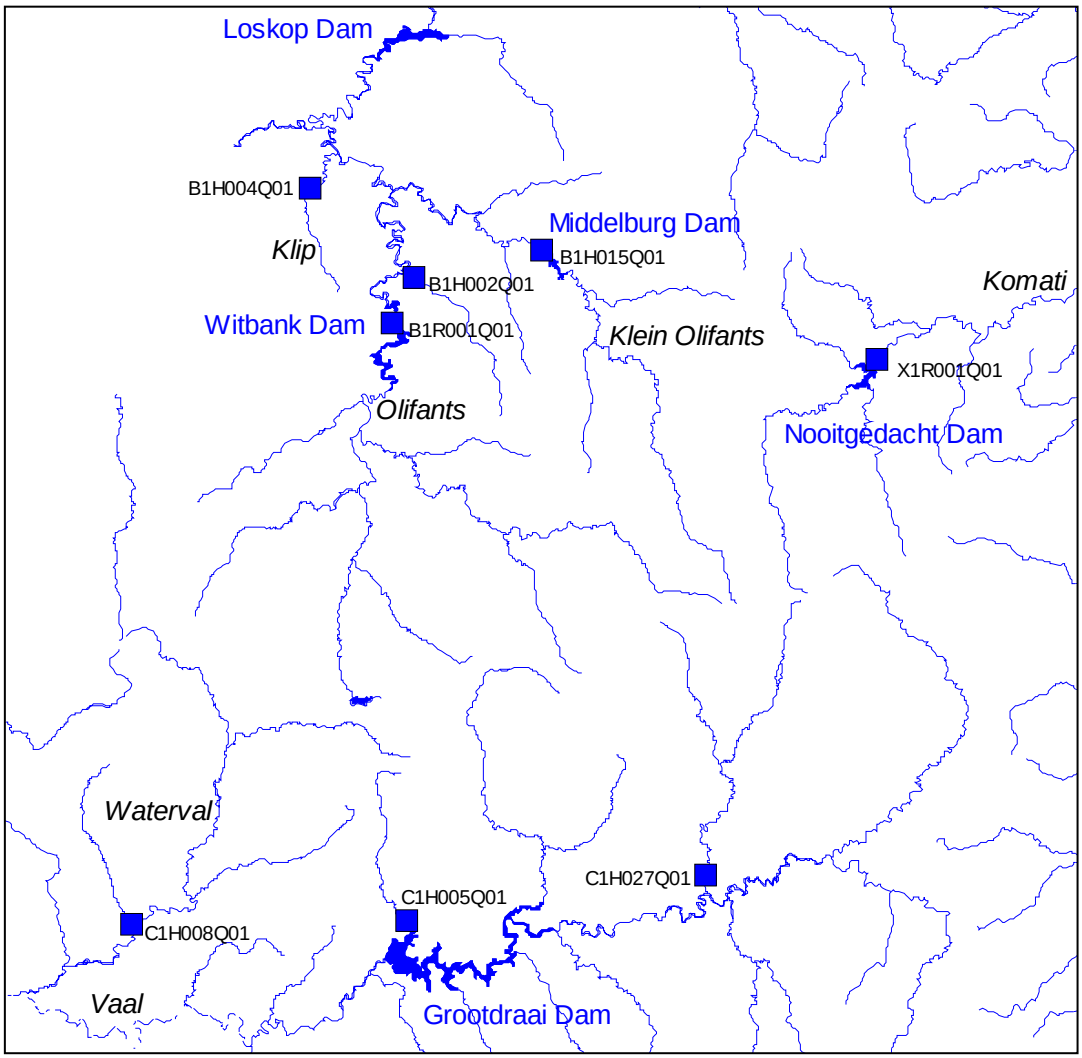


Figure 5. DWA&F water quality and quantity monitoring positions leading away from mining areas. Scale from left to right is 150 km.

Details on sulphate and sodium concentrations at these sites are shown in Figure 6 – 8. Conclusions from these displays are:

- The data from the DWA&F are generally of a high quality and should more often be used by mines for the regional evaluation of surface water quality, as it relates to the mines. These data are available from the DWA&F upon request.
- Significant is the degree of sulphate contamination in the streams. The Klip Spruit and Spook Spruit are heavily contaminated. Similarly, high spot values of sulphate may be found along some of the other streams, though in the dams, such as Witbank, Middelburg and Grootdraai, these values average out. Hence the significantly lower and less fluctuating sulphate concentrations in the dams.

- Sodium is a constituent that varies significantly in concentration over the coalfields. Many of the collieries, such as those in the central northern and northeastern portions, are practically devoid of high sodium concentrations. Further south, sodium is increasingly present in the mine water. This is particularly true for mines south of the -2 893 000 line, which transects the study area from west to east just north of Kriel and Matla Collieries. The Secunda and New Denmark Mines are known for their high sodium levels in the mine water. This is then also reflected in the water of the Waterval River and Leeu Spruit, which periodically receive water from these mines.

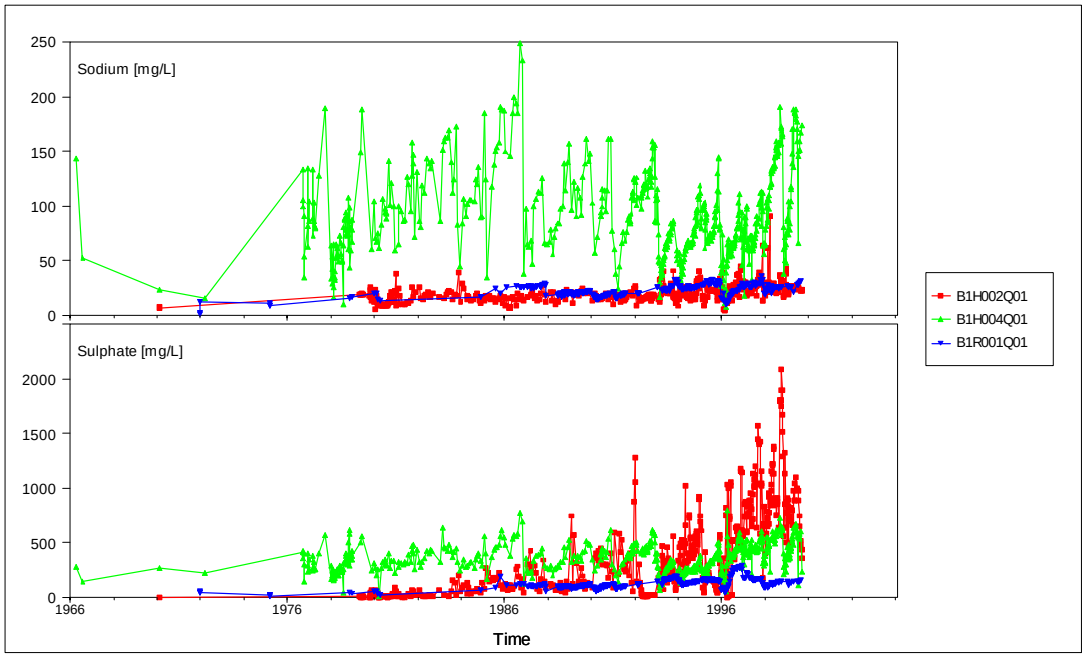


Figure 6. Water qualities in the Klip and Olifants Rivers and the Spook Spruit.

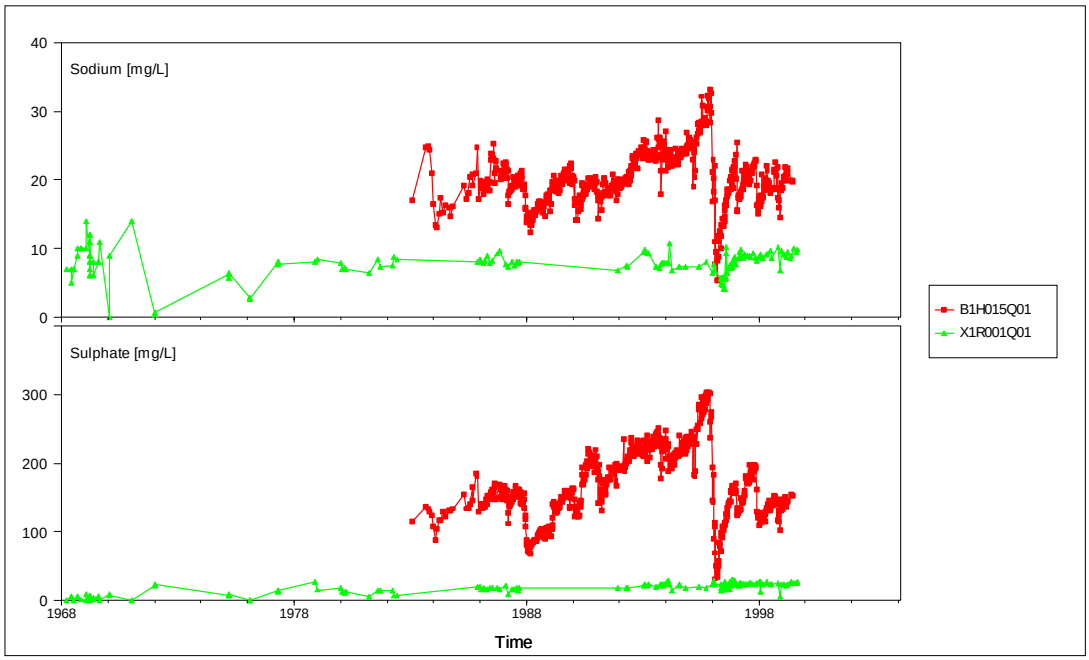


Figure 7. Water qualities in the Klein Olifants and Komati Rivers.

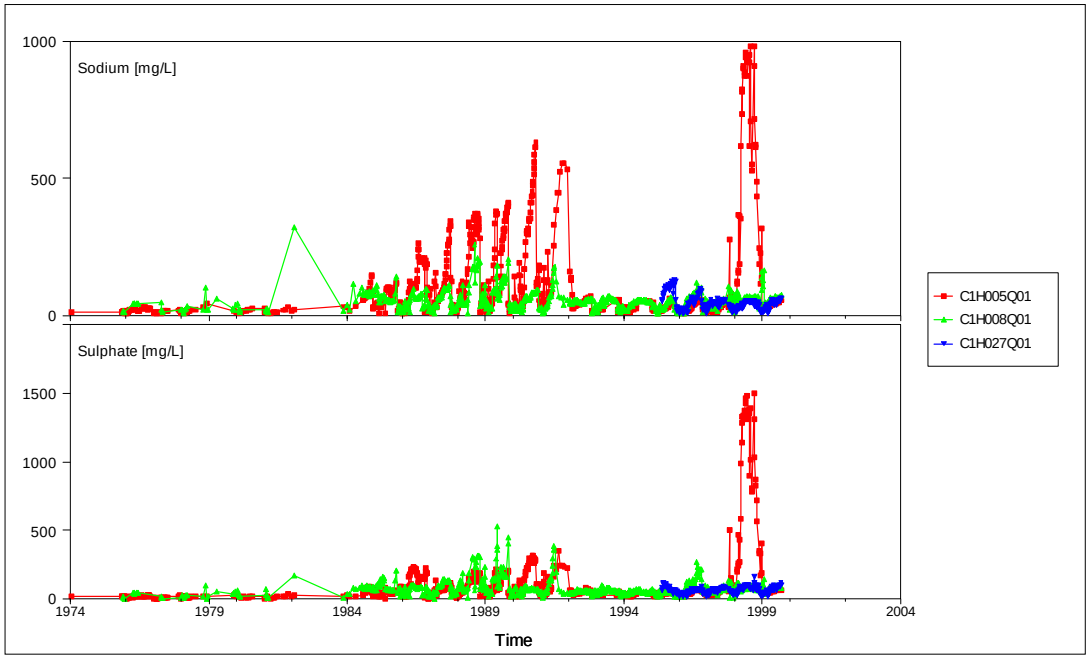


Figure 8. Water qualities in the Waterval and Vaal Rivers and Leeu Spruit.

- The rising trends, particularly in the sulphate concentrations in the surface waters, are alarming. It is inferred that the maximum pollution potential is not reflected by any of these graphs. The outcome, when all mines partake in intermine flow and decanting, is therefore the pressing question that needs to be addressed by this investigation.

Another way of demonstrating the degree to which sulphate, for instance, already dominates the surface water chemistry in these coalfields, is by making use of specialised chemical plots. A Piper diagram shows the water chemistries in Figure 9.

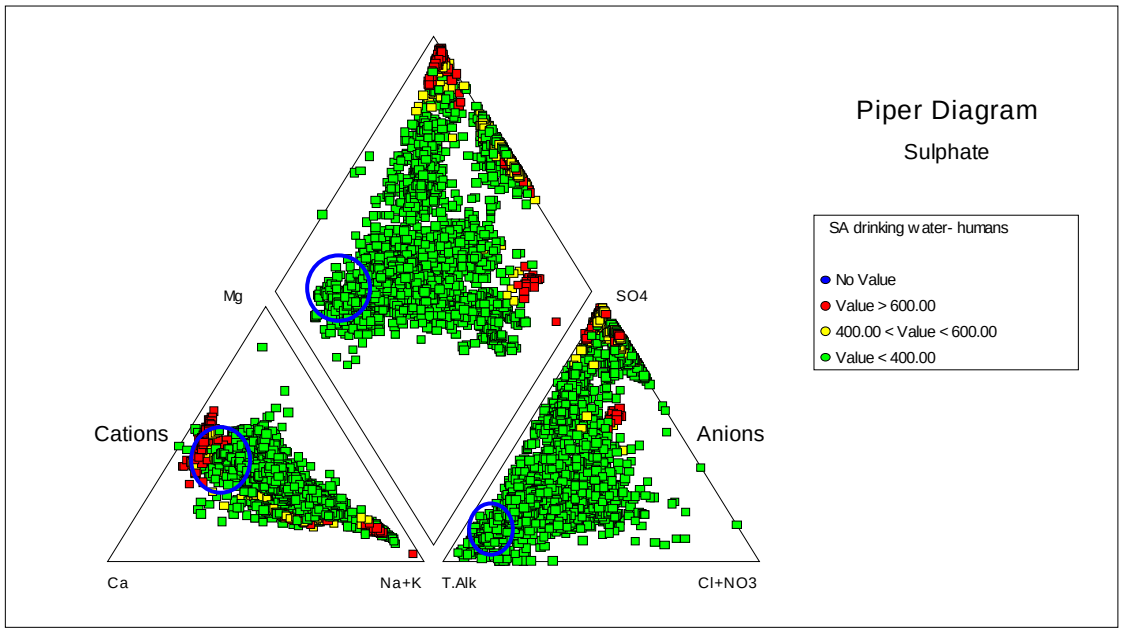


Figure 9. Piper diagram of the major elements at the sampling positions indicated in Figure 5.

The blue circles indicate the areas on this diagram where unpolluted water would plot

because of its natural Ca/Mg/HCO_3 content. Deviations from these areas imply different degrees of pollution. Two strong trends of pollution are present. The one is in the left triangle, suggesting sodium pollution. The other is in the right triangle, indicating sulphate pollution. Both types of pollution occur in the southern mines of the study area. In the north, pollution is in the form of sulphate, calcium and magnesium. Calcium and magnesium enrichment can also be seen in the left triangle.

Another interesting and conclusive way of comparing the different waters that emanate from the coalfields, is using the so-called Stiff diagram (Figure 10). This allows comparison between six components for each site. The six components are usually calcium, magnesium, sodium + potassium, alkalinity, sulphate and chloride + nitrate. Each of these is plotted on horizontal axes, with cations to the left and anions to the right. The extremities of the points are connected and the inside is coloured, thus creating unique shapes that represent the overall chemistries for each site.

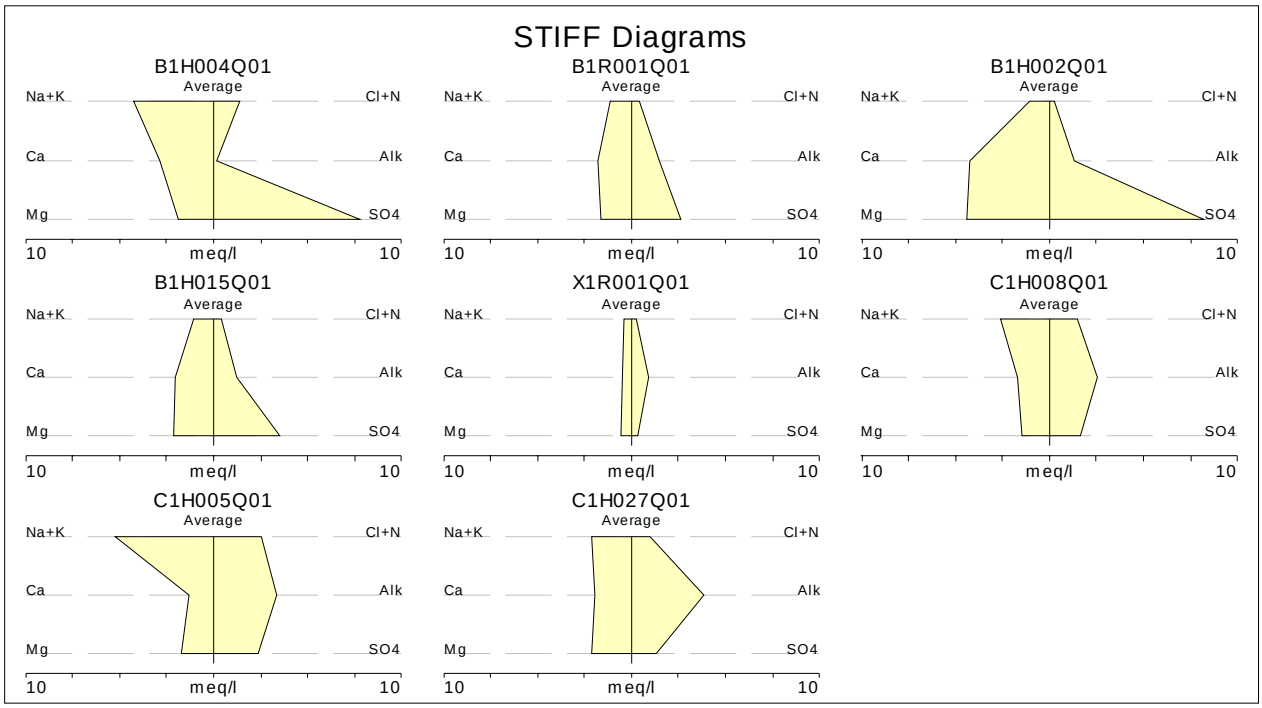


Figure 10. Stiff diagrams for the eight streams (Figure 5) leading water from the collieries.

While these diagrams may appear to be highly varied in shape, this is exactly the purpose of this plot. It characterises water with a unique signature, clearly showing the dominant constituents at each of the sites. The Klip Spruit (B1H004Q01), for instance, has a dominance of sodium and sulphate, exceeding its calcium and magnesium content. This contrasts with that in the Spook Spruit (B1H002Q01) where calcium and magnesium are the dominant cations. Similarly, the greater dominance of sodium over calcium and magnesium can be seen in the Leeu Spruit (C1H005Q01) compared to the Waterval (C1H008Q01) water.

A last way of regionally comparing concentrations in the surface water, is plotting

sites according to constituent concentrations. Sites with higher concentrations are shown by larger blocks in Figure 11.

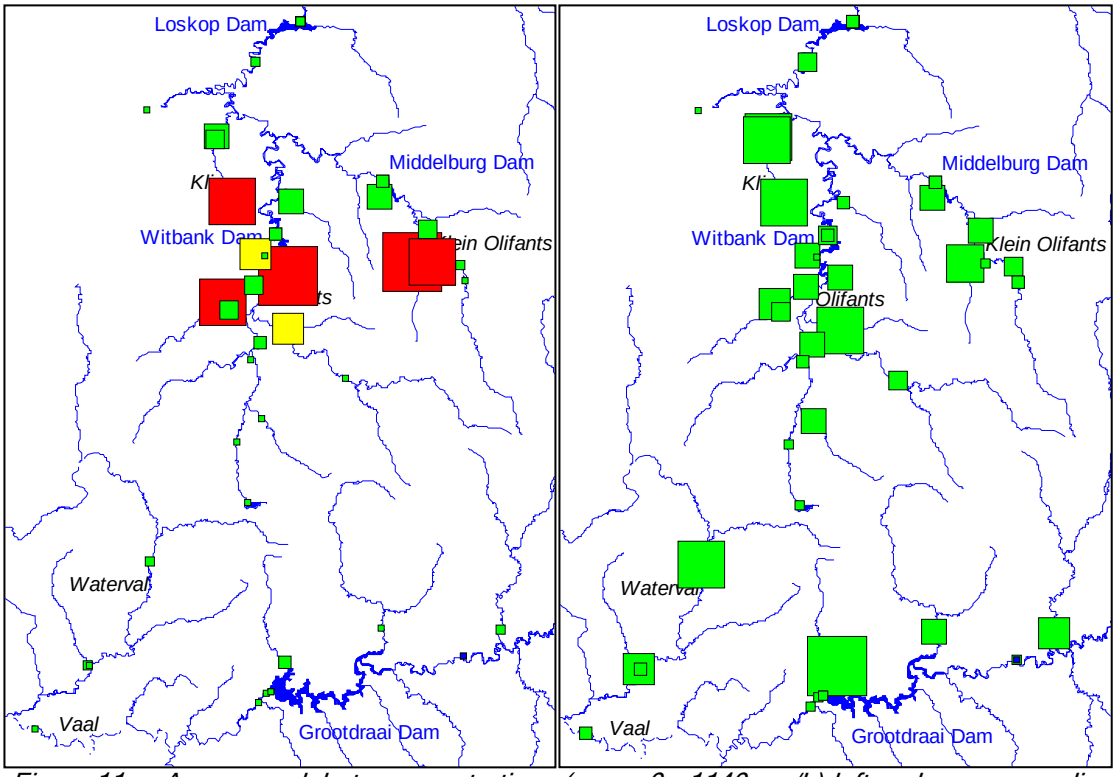


Figure 11. Average sulphate concentrations (range 0 - 1140 mg/L) left and average sodium concentrations (range 0 - 120 mg/L) right, demonstration the relative surface water contamination. (Red = very high; Yellow = moderately high; Green = Acceptable).

Conclusions from these presentations are as follows:

- The extent of surface water pollution due to pyrite oxidation in the north-flowing streams is severe. The pollution is expected to become worse with time, reaching a maximum when all the mines in this area close down and decant. Sulphate pollution, though present in the south-draining streams, is currently negligible compared to that in the north. Sulphate pollution to the south will, however, increase with time, as larger areas are oxygenated.
- Sodium pollution in the southern streams is due to the high availability and solubility of this element in these collieries. The coal as well as the overburden has high sodium content. Most of the mining in this catchment is by underground high extraction methods, which exposes both the coal and the overlying rock to circulating water. Apart from sodium, other soluble constituents such as chloride and fluoride are also naturally available. Sulphate, calcium and magnesium will be increasingly released from these mines in the future. If left uncontrolled, similar conditions to the north-draining streams are expected to develop over a period in the south.
- The two constituents illustrated in this report are not the only two that should be considered as part of the pollution scene. The information available from

the DWA&F contains many other parameters, such as pH, electrical conductivity, calcium, magnesium, chloride, alkalinity and fluoride. The examples provide, however, sufficient detail to suggest cause for concern. The overall picture will again be discussed in the current document, once all the information has been presented.

2.4 GEOLOGY AND GEOHYDROLOGY

2.4.1 Geology

The sediments of the coal-bearing Ecca Group of the Karoo Sequence were deposited on an undulating pre-Karoo floor, which had a significant influence on the nature, distribution and thickness of many of the sedimentary formations, including the coal seams. Post-Karoo erosion has removed large parts of the stratigraphic column, including substantial volumes of coal over wide areas.

The Karoo Supergroup comprises the Ecca Group and Dwyka Formation. The total thickness of these sediments ranges from 0 – 300 m. A general thickening occurs from north to south. The Ecca sediments consist predominantly of sandstone, siltstone, shale and coal (Figure 12).

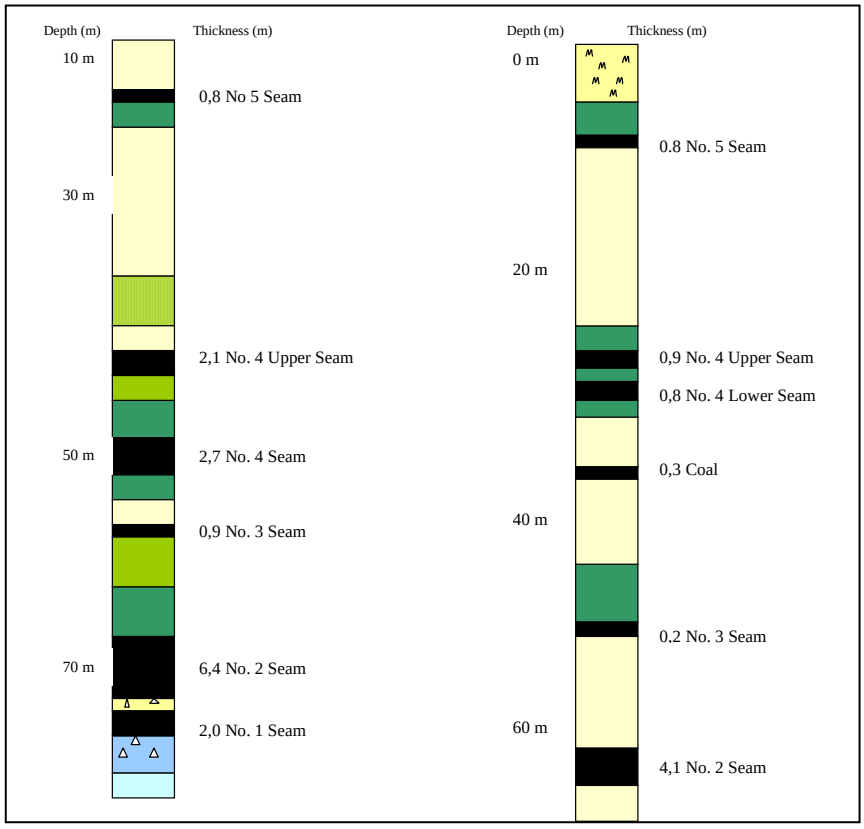


Figure 12. Two typical and generalised geological profiles for the Coalfields, showing the variability (modified from Hodgson et al., 1985).

Combinations of these rock types are often found in the form of interbedded siltstone, mudstone and coarse-grained sandstone (Figure 13). Typically, coarse-grained

sandstones are a characteristic of the sediments in the Witbank area.

Figure 13. Photographs of typical sedimentary rocks from the Mpumalanga Coalfields.

Five coal seams, numbered from bottom to top as No. 1 - 5, are present in the Witbank area. Only two of the seams are mineable over most of the area. These are the No. 2 and 4 Seams, which are usually separated by sediments of a total thickness in the order of 20 – 30 m. Seams 1 and 5 are, however, mined locally. In areas where coal has formed, the paleo-floor has had a controlling influence on the distribution and thickness of the seams (De Jager, 1976). The coal seams can be described as horizontal, slightly undulated layers. In other areas, such as Ermelo and Standerton, other names are used for the coal seams. The continuity of the coal seams over such a large area has not been proven, because of the large areas in-between that have not been mined.

Dolerite intrusions in the form of dykes and sills are present in the Ecca Group. The sills are highly undulating and some might conform to the ring structures described by Burger *et al.* (1981) in the southern Free State. Burger *et al.* (1981) have discussed their mechanism of emplacement in detail. The sills usually precede the dykes, with the latter being emplaced during a later period of tensional forces within the earth's crust.

The Ecca sediments overlie the Dwyka Group (loosely referred to as the Dwyka tillite). This formation consists of a proper tillite, siltstone and sometimes a thin shale development. The upper portion of the Dwyka sediments may have been reworked,

in which case carbonaceous shale and even inclusions of coal may be found.

The Dwyka sediments are underlain by a variety of rock types, such as the Bushveld Complex in the north, Witwatersrand Supergroup in the south, Waterberg Supergroup in the north-west and Transvaal Supergroup to the west.

Tectonically, the Karoo sediments are practically undisturbed. Faults are rare, except for displacement along dolerite ring structures. Fractures are common in competent rocks such as sandstone and coal.

2.4.2 Geohydrology

Three distinct superimposed groundwater systems are present. They are the upper weathered Ecca aquifer, the fractured aquifers within the unweathered Ecca sediments and the aquifer below the Ecca sediments.

2.4.2.1 The Ecca weathered aquifer

The Ecca sediments are weathered to depths between 5 - 12 m below surface throughout the area. The upper aquifer is associated with this weathered zone and water is often found within a few metres below surface. This aquifer is recharged by rainfall. The percentage recharge to this aquifer is estimated to be in the order of 1 - 3% of the annual rainfall, based on work in other parts of the country by Kirchner *et al.* (1991) and Bredenkamp (1995).

Observed flow in the catchment confirmed isolated occurrences of recharge values as high as 15% of the annual rainfall (Hodgson *et al.,* 1998). It should, however, be emphasised that in a weathered system, such as the Ecca sediments, highly variable recharge values can be found from one area to the next. This is attributed to the composition of the weathered sediments, which range from coarse-grained sand to fine clay.

The northwestern portion of the coalfields is characterised by coarser-grained sandstone and higher recharge values are expected here. It is concluded from this information that a recharge value in the order of 3% of the annual rainfall is feasible. In terms of the catchment size for the Witbank Dam for instance (3 250 km^2), this amounts to 60 Mm^3 recharge per annum (Hodgson *et al.,* 1998). Compared to the holding capacity of the Witbank Dam (104 Mm^3), this amounts to about half of this value.

Rainfall that infiltrates into the weathered rock reaches impermeable layers of sediments below the weathered zone. The movement of groundwater on top of these sediments is lateral and in the direction of the surface slope. Water reappears on surface at springs where the flow paths are obstructed by a barrier, such as a dolerite dyke, paleo-topographic highs in the bedrock, or where the surface topography cuts

into the groundwater level at streams. It is suggested that less than 60% of the water recharged to the weathered zone eventually emanates in streams (Hodgson *et al.,* 1998). The rest of the water is evapotranspirated or drained by some other means.

The aquifer within the weathered zone is generally low-yielding (range 100 – 2000 L/h), because of its insignificant thickness. A few farmers tap this aquifer by borehole. The excellent quality of this water can be attributed to the many years of dynamic groundwater flow through the weathered sediments. Leachable salts in this zone have been washed from the system and it is only the slow decomposition of clay particles, which presently releases some salt into the water (Hodgson *et al.,* 1998).

2.4.2.2 The fractured Ecca aquifers

The pores within the Ecca sediments are too well-cemented to allow any significant flow of water. All groundwater movement therefore occurs along secondary structures, such as fractures and joints in the sediments. These structures are better developed in competent rocks, such as sandstone, hence the better water-yielding properties of the latter rock type.

It should, however, be emphasised that not all secondary structures are water bearing. Many of these structures are constricted because of compressional forces that act within the earth's crust. The chances of intersecting a water-bearing fracture by drilling decrease rapidly with depth. At depths deeper than 30 m, water-bearing fractures with significant yield were observed to be spaced at 100 m or greater (Hodgson *et al.,* 1998). Scientific siting of water-supply boreholes is necessary to intersect these fractures.

Of all the unweathered sediments in the Ecca, the coal seams often have the highest hydraulic conductivity. Packer testing of the No. 2 Seam and underlying Dwyka tillite has a hydraulic conductivity distribution as indicated in Table 1.

Table 1. Statistics for results on packer hydraulic conductivity testing of the No. 2 Seam and Dwyka tillite (Hodgson et al.,1998).

Statistics	2 Seam Permeability	Dwyka Permeability
Mean (m/d)	0,1017	0,0034
Median (m/d)	0,0743	0,0024
Standard Deviation (m/d)	0,1295	0,0034
Minimum (m/d)	0,0007	0,0002
Maximum (m/d)	0,5007	0,0148
Number of tests	21	21

From this comparison, it is clear that seepage of water through the No. 2 Seam is possible. Due to its low hydraulic conductivity, the Dwyka tillite forms a hydraulic barrier between the overlying mining activities and the basal floor.

2.4.2.3 Pre-Karoo aquifers

Drilling in only a few instances has intersected the basement to the Karoo Supergroup. Very few of the farmers, if any, tap water from the aquifer beneath the Dwyka Formation. The reasons for this are:

- The great depth.
- Low-yielding character of the fractures.
- Inferior water quality, with high levels of fluoride, associated with granitic rocks.
- Low recharge characteristics of this aquifer because of the overlying impermeable Dwyka tillite.

In the southern portion of the catchment, dewatering of this aquifer has, to some extent, occurred because of the pumping in the Evander Goldfields. Here, the piezometric pressure in the deep aquifer is generally 10 - 50 m lower than that in the Karoo sediments (Hodgson *et al.,* 1998).

3 MINING METHODS IN THE MPUMALANGA AREA

3.1 INTRODUCTION

Coal extraction has been ongoing in the Mpumalanga coalfields for more than 100 years. At first, mining was mainly to the west of Witbank. Many of these mines have closed down and are presently a major source of pollution. The water quality in the Klip Spruit (Figure 6) is a reflection of the current state of affairs. This poor quality should serve as a warning of what would happen in the rest of the mining area, if intermine flow commences there.

Through the years, mining extended from its original position to the south and east. Many new collieries commenced with mining, particularly during the past 30 years. Since 1970, mining has increasingly been mechanised. All modern coal-mining methods are currently employed. These are:

- Bord-and-pillar extraction.
- Secondary mining through pillar extraction. This is commonly referred to as stooping.
- Underground high extraction through longwall and shortwall methods.
- Auger mining is presently considered on a limited scale for the extraction of thin coal seams.
- Opencast mining, using truck and shovel or dragline methods.

Initial underground mining was relatively shallow, in the range of 10 – 50 m below surface. Mining was mainly through underground methods. Access to the underground workings was commonly through inclined shafts.

As mining spreads to the south in particular, the coal seams deepen. The current deepest mining planned for is 270 m below the surface. Access to the deep mines is through vertical shafts.

3.2 BORD-AND-PILLAR EXTRACTION

Bord-and-pillar extraction has been the primary method of mining throughout the Mpumalanga Coalfields. Reasons for this are:

- It allows access to the coal in a structured and organised way.
- It can be maneuvered around geological or coal quality constraints.
- The extraction rate is reasonably high, ranging from 70% in the shallow mines to 50% in deeper areas.

A typical example of a bord-and-pillar mine lay-out is provided in Figure 14.

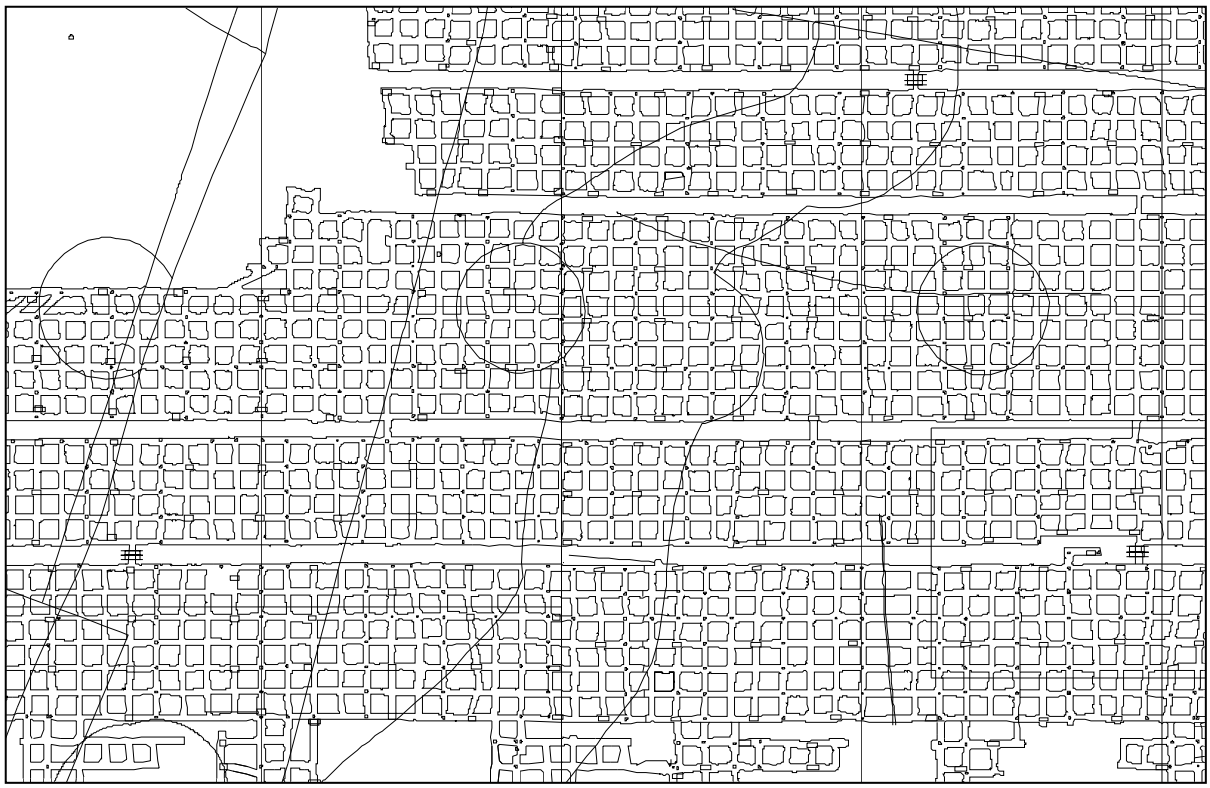


Figure 14. Example of bord-and-pillar mining in a modern underground colliery. Apart from the coal pillars, certain surface infrastructure such as streams, roads and survey markers are also shown.

It is clear from the example that the underground maze of tunnels is not necessary regular or symmetrical. Water in one end of the mine will not flow in a straight line

through the mine. It will follow the tunnels and be regulated by coal-floor gradients (Figure 15).

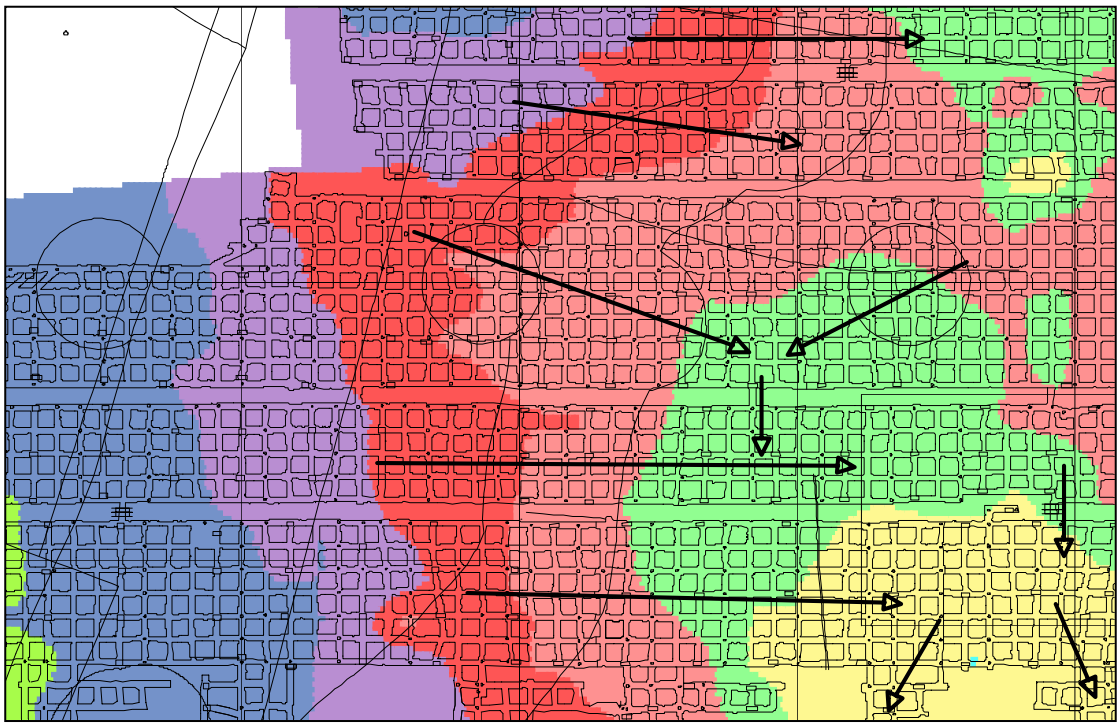


Figure 15. Floor contours and water flow vectors for bord-and-pillar mining.

Bord-and-pillar mining is usually accomplished using continuous miners. A certain amount of blasting may also be necessary. The coal itself is naturally riddled with fractures. Packer testing on the coal has revealed that the coal is generally the most permeable of all the sediments (Hodgson *et al.,* 1998), except in deep mines. Here, many of the fractures in the coal are filled with calcite. This decreases the hydraulic conductivity of the coal and at the same time increases its base potential.

Bord-and-pillar mining constitutes an estimated 65% of the past mining in the Mpumalanga Coalfields. In many of the mines, waste rock material is discarded in the underground tunnels. This could affect the long-term chemistry of the mine water.

The coal pillars in modern coal mines stand up well. Extensive research has been done by the Chamber of Mines on pillar stability. Most pillar failures have occurred in areas which were mined before research by the Chamber of Mines was done. Extensive areas where the roof of the coal has collapsed (>250 ha), occur to the west of Witbank (Figure 16). These subsidence structures act as recharge and potential discharge points for intermine flow. The long-term stability of all pillars in the Mpumalanga Coalfields is sometimes questioned. The gradual chemical decay of the pillars, because of pyrite oxidation and accompanying carbonate dissolution, could result in additional future collapses. This study does not allow for the very long-term instability of pillars. Only known interconnections to the surface are currently considered for the calculation of intermine flow.

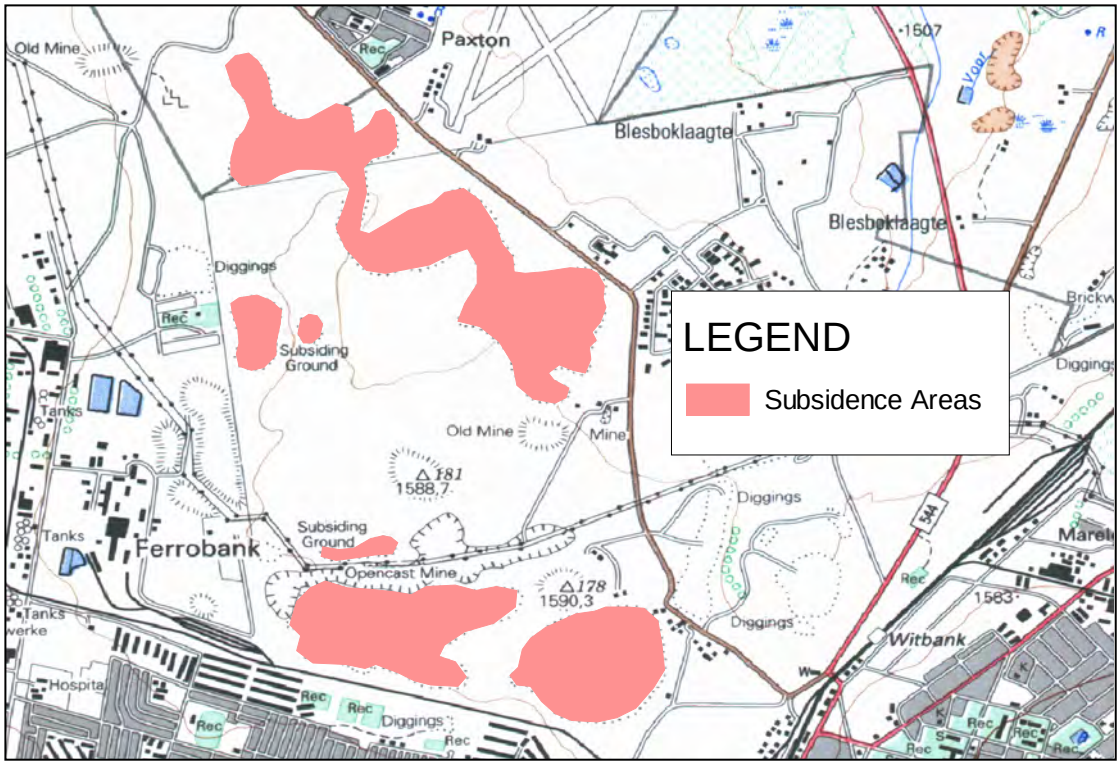


Figure 16. Extent of subsidence areas above bord-and-pillar mining west of Witbank. Scale from left to right is 6 km.

Influx of water into underground bord-and-pillar areas is usually low. While mining, water seeps are often encountered in the coalface. These soon dry up as mining progresses. The vertical hydraulic conductivity of the over- and underlying sediments are usually too low to convey water of any significance into the mines. The odd vertical fracture that may be present sometimes yields water for a limited period (weeks rather than months). In exceptional cases, a sustained flow of groundwater into the mine may be intersected by mining. Very little, if any, of the coal mining has been stopped because of water influx.

The quantification of water influx into bord-and-pillar workings on a mine basis is difficult, if not impossible. This is due to the vast number of depressions in the coal floor, where water accumulates before reporting to the central pumping facilities. Water on the coal seam is only pumped from the workings if it interferes with mining. The influx of water into bord-and-pillar areas, as reported in this document, is therefore derived from estimates by a number of collieries. In theory, influx into bord-and-pillar areas should correlate with the perimeter of the mine. This relationship between the mined area and groundwater influx, as established for the Mpumalanga Collieries, is demonstrated in Figure 17.

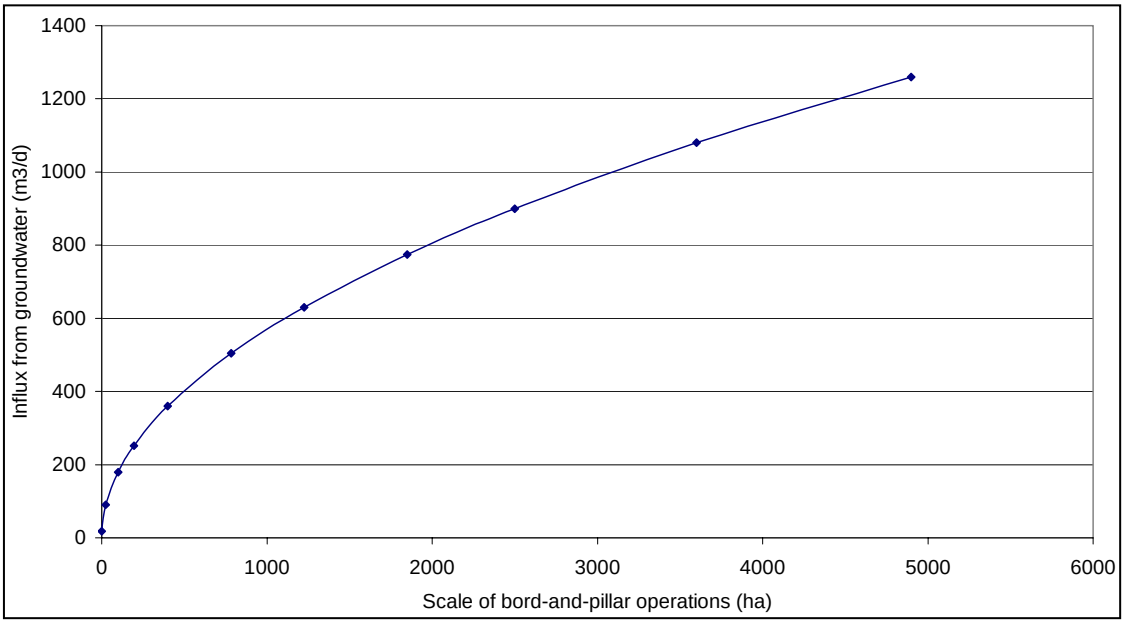


Figure 17. Empirical relationship between the area mined by bord-and-pillar methods and water influx.

3.3 STOOPING

Stooping has been done for at least 30 years in the Mpumalanga Coalfields. Usutu Colliery was one of the first where this was done on a significant scale (Figure 18). The mine lay-out in this figure should serve as an example for the rest of the mining industry. Of significance is the single entrance to the stooped area. Through the installation of water seals at this entrance the whole stooped area could be sealed off.

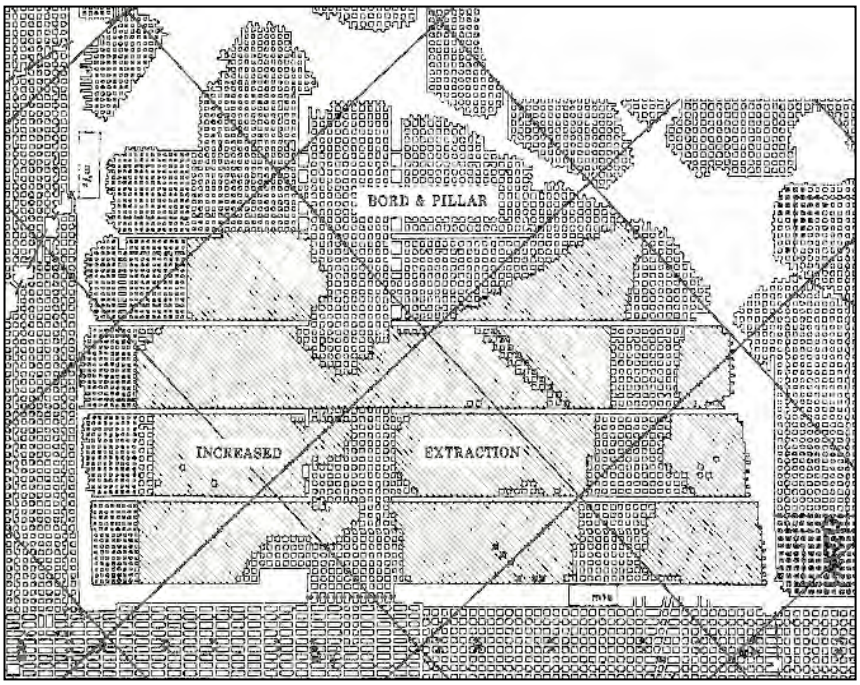


Figure 18. Example of stooped areas at Usutu Colliery with access to the central area only from the south.

The detail in stooped areas may vary considerable (Figure 19).

Figure 19. Detailed examples of bord-and-pillar with partial stooping.

Pillars may be halved, quartered or completely removed. Safety constraints rather than economic considerations often dictate the extraction pattern. The extraction rate per unit area is highly variable. Depending on many factors, such as mining depth, percentage extraction and rock competency, collapse of the overlying strata up to the surface may, or may not occur. In mines where stooping is done, classification of areas according to the likelihood of roof failure should be done (Figure 20).

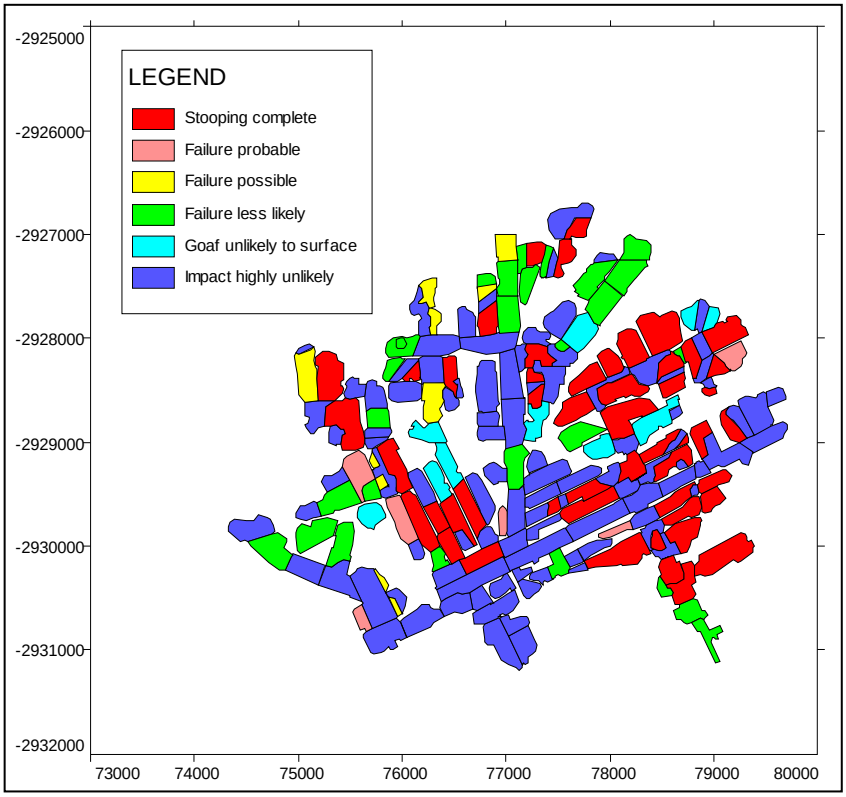


Figure 20. Example of a mine where stooping has partially been done and the classification of areas according to the likelihood of roof failure (goafing) to surface.

This information can then be translated into the probability of water in- or outflow

through collapsed areas. Decanting points of mine water are likely to coincide with collapsed strata in low-lying topographic areas.

Many collieries in Mpumalanga either have experimented with stooping or have done full-fledged stooping with the purpose of increasing the coal yield. Many current excess water and water quality problems stem from these operations. Water influx through collapsed structures will vary significantly from area to area, depending on many factors. One of the prime issues is the undermining of streams. Greater quantities of water should be expected from streams in instances of rock outcrop in a streambed, than from clayey streambeds.

As a rule of thumb, 6% of the average annual rainfall onto collapsed areas, should be taken for design purposes to calculate water inflow into stooped mines. This percentage includes lateral groundwater flow towards the cracks, because groundwater is derived from rainfall in the first instance.

Currently, vast areas of coal are extracted through stooping. This is particularly true for Secunda Collieries, but many of the other mines in Mpumalanga are also planning large-scale stooping as their coal reserves become limited. An escalation of the water handling problems in the Mpumalanga area is therefore predicted.

3.4 LONGWALL MINING

Longwall mining (Figure 21) has been done in the Mpumalanga Coalfields since 1979.

Figure 21. One of the first longwall operations in South Africa (1979).

Matla and Secunda Collieries were first to introduce longwall mining in Mpumalanga. New Denmark Colliery followed in 1983. In Matla Colliery, three overlying coal

seams, namely No.'s 2, 4 and 5 are extracted by longwall mining. At the other collieries, only one seam is mined. Because of complex geological structures and other problems at their mines, Secunda Collieries have converted their longwall operations to stooping during the past ten years. Excluding Secunda Collieries, the total area currently planned for longwall mining in Mpumalanga exceeds 20 000 ha.

Longwall mining is usually done in areas of few structural disruptions. Where structural discontinuities are present, this creates major problems and a "jumping" action is usually performed, whereby the problem area may be mined by conventional bord-and-pillar methods (Figure 22).

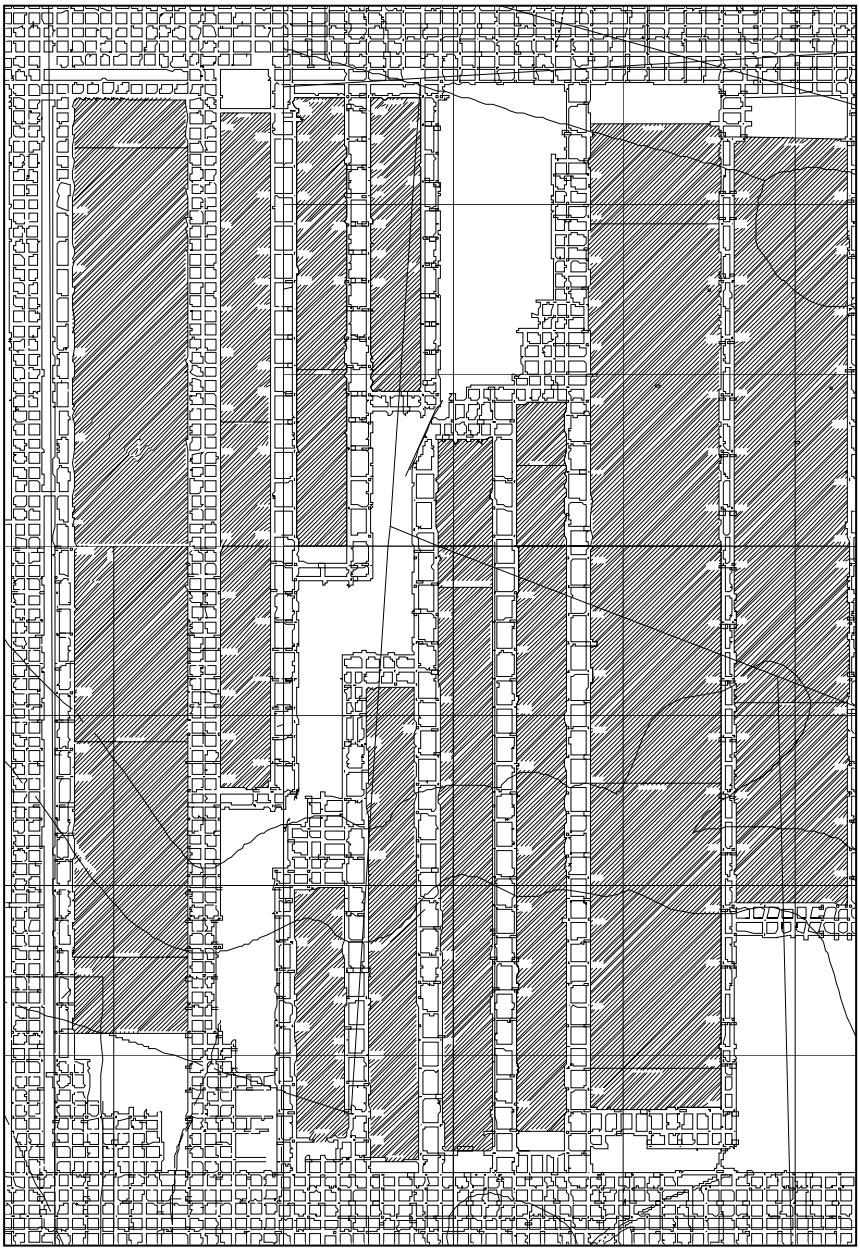


Figure 22. Example of longwall panel lay-out with a structural discontinuity transecting it, also showing bord-and-pillar mining to allow access.

Longwall panels are usually 200 m wide, but the width may vary depending on local

considerations. In some high extraction areas, as is shown in this figure, shortwalling may be done. Shortwall panels are usually 100 m wide. This span may, in certain circumstances, not be sufficient to collapse the overlying strata up to the surface. The top aquifer is therefore not drained.

The mining height in longwall panels is seam dependent, but 3 m is usually the maximum height catered for. In some of the mines, the average seam thickness is only 2 m. The amount of surface subsidence above the longwall panels has been documented extensively. This usually amounts to about 50% of the mining height. The mining depth does not play a major role in this percentage, which leads to the conclusion that most of the fractured material is in the horizon immediately above the coal seam. Higher up in the succession, large blocks of rock, sometimes more than 100 m in size, occur (Figure 23).

Figure 23. Circular fracture above a longwall panel, in a ploughed field, with a diameter of 160 m.

In terms of groundwater flow into collapsed areas, this is schematically displayed in Figure 24. The following important issues should be noted:

- The fractures slope inwards.
- Several aquifers are present.
- A dewatering cone establishes for each of the aquifers.
- The lateral extent of the dewatering cones is limited. The main controlling factors are the relatively low hydraulic conductivity of the aquifers and recharge from rainfall, which limits cone development in the upper aquifer.

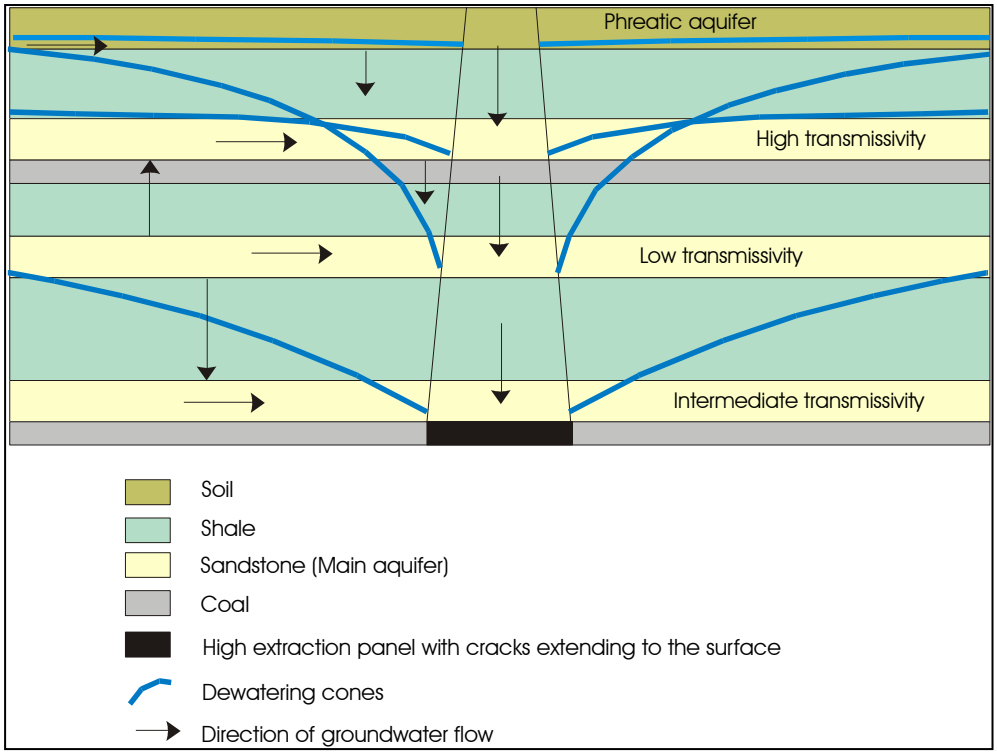


Figure 24. Schematic representation of dewatering cones and the vertical interaction between aquifers which are intersected by fractures from underground high extraction.

Longwall mining is not constrained by depth. At Matla Colliery, the shallowest longwall mining is less that 50 m below surface. The deepest planned longwall mining is at New Denmark Colliery, where a depth of 270 m will be reached.

Influx of water from the collapsed overlying strata, and from rainwater that enters into the cracks, is problematic at all longwall operations. As a rough rule of thumb, 0.2 ML/d is derived from each km^2 mined. This volume of water could be significantly reduced if streams are not undermined by longwall mining. However, so many streams are present in the Mpumalanga area, that on the current scale of longwall mining, this suggestion is not practical.

3.5 OPENCAST MINING

Opencast mining has escalated in Mpumalanga over the past 30 years with the introduction of dragline mining (Figure 25). Previously, all opencast operations were done through truck and shovel operations. The latter is still used in small opencast mining and, of course, to mine the coal itself in dragline operations. Draglines are only used for the removal of rock above the coal.

Pre-stripping of soil in accordance with guidelines set out by the Chamber of Mines (1981) is standard procedure at all large opencast collieries and at the majority of the smaller ventures.

Figure 25. Dragline mining in an opencast pit .

In the case of dragline mining, the stratigraphic column is more or less inverted. Selective spoil handling, on a limited scale, can be done by dumping at different angles with respect to the operating cut. This has, however, never been done and could be impractical in the field situation. Truck and shovel methods allow selective spoil handling – at least in theory. Spoil material could be transported, dumped and compacted in areas specifically allocated for spoil of specific compositions.

Mining depths in opencast operations normally range from 0 - 60 m below surface. Only one of the collieries (Rietspruit) has reached a depth in excess of 80 m below surface, in a staged dragline operation.

The large opencast collieries extract between approximately 7 - 14 Mt coal per annum. As part of this operation, an average of 50 Mt of overburden is moved and replaced per annum, at each large colliery. This amounts to approximately 400 Mt spoil per annum, for opencast mining within the Olifants Catchment (Hodgson *et al.,* 1998).

In terms of surface area mined, opencast pit areas for the larger collieries typically range from 1 000 – 5 000 ha. This is large by any standard. In several instances, such as at Kromdraai and Kleinkopje Collieries, opencast mining proceeds into areas previously mined by bord-and-pillar methods. Depending on the remedial measures to be taken after the opencast mining, this could improve the current situation in

terms of water pollution control.

Many opencast collieries produce coal for power generation. In most of these instances, the entire coal product is delivered to the power station. At other mines, the coal is put through a washing plant and a significant amount of coal discards and coal slurry is produced. These are disposed in a variety of ways, such as on the surface; on top of, or in opencast pits; and in underground workings.

Water in operating opencast pits is derived from various sources. Table 2 provides a breakdown of these sources as a function of the average annual rainfall or the total ingress of water into a pit.

Table 2. Water recharge characteristics for opencast mining (Hodgson et al., 1998).

Sources which contribute water	Water sources into opencast pits	Suggested average values
Rain onto ramps and voids	20 - 100% of rainfall	70% of rainfall
Rain onto unrehabilitated spoils (run-off and seepage)	30 - 80% of rainfall	60% of rainfall
Rain onto levelled spoils (run-off)	3 - 7% of rainfall	5% of rainfall
Rain onto levelled spoils (seepage)	15 - 30% of rainfall	20% of rainfall
Rain onto rehabilitated spoils (run-off)	5 - 15% of rainfall	10% of rainfall
Rain onto rehabilitated spoils (seepage)	5 - 10% of rainfall	8% of rainfall
Surface run-off from pit surroundings into pits	5 - 15% of total pit water	6% of total pit water
Groundwater seepage	2 - 15% of total pit water	10% of total pit water

On average, 20% of the rainfall is considered to recharge opencast pits during the operational phase, as well as after closure (Hodgson *et al.,* 1998*)*. This rather high percentage is due to:

- Internal run-off on much of the rehabilitated spoil.
- Pounding on top of the rehabilitated spoil and infiltration through the spoil.

These recharge values require further clarification:

- Not all rainfall that precipitates within ramps and voids contributes to the overall water balance. Much of this water is accumulated in local depressions, from where it evaporates. Many of the ramps and voids are likely to remain dry, even after a pit has closed down and has filled with water to the decanting level. It can be concluded that the percentage recharge through ramps and voids is a function of the slope of the pit floor and the degree to which these structures are filled with water. With the average evaporation potential from water exceeding the annual rainfall by 700 - 800 mm in the study area, the ramps and voids with standing water should have a negative affect on the water balance.
- Unrehabilitated spoil heaps constitute a significant percentage of the disturbed areas in South African coal mines. A survey indicates that rehabilitation lags 2 - 6 cuts behind the operating cut, per dragline operation. These spoil heaps have a high rainfall recharge potential. The lack of erosion scars on the spoil heaps

suggests that rainwater penetrates into spoil without much obstruction. Optimally, from the point of view of water control, rehabilitation should follow within two cuts of the operating cut.

- Spoil is usually levelled by dozing equipment. This often results in compaction of the upper portion of the spoil, particularly where a high percentage of shale is present. Sandstones and siltstones are usually not crushed during dozing and permeation of water through this material is still possible, though at a reduced rate. Run-off from levelled spoils easily erodes scars into the spoil. Erosion channels permit uncontrolled water recharge to the pit. With time, levelled spoils become less permeable, because of the decomposition of the argillaceous material and silting up of channels. It is concluded that the vertical hydraulic conductivity of levelled spoils is highly variable and depends on the amount of compaction, surface slopes, spoil composition and age.
- Levelled spoils are usually covered by soil. The soil is vegetated with different seed mixtures varying from one company to the next. Mixtures are designed to establish a quick growth, allowing the plants to develop over a number of years. The evapotranspiration properties of the soil and vegetation differ between mines and seasons. Areas exist where grass growth exceeds 2 m in height. At some of the collieries, channelling of surface run-off has, in places, eroded through the topsoil, thus exposing the underlying spoil. Infiltration in these areas is envisaged to be variable, ranging from high to low.
- Surface run-off from the unmined areas is usually diverted by cut-off trenches away from the pits. Despite these precautionary measures, some overland run-off still enters into the pits. Through good management, the contribution of water from this source should be small, compared to the total influx of pit water.
- Groundwater seepage into opencast coal mines as a percentage of the total water in the pit, is small. As indicated in the discussion on geohydrology and borehole yields, the Ecca rocks yield very little water. Groundwater influx mainly occurs from the upper weathered aquifer which, in turn, is recharged by rainfall. Groundwater seeps into the pits at the bottom of the weathered aquifer, or from weathered contacts next to dolerite dykes, from sporadic fractures in the unweathered Ecca sandstones lower down and from the coal seams themselves. These seepages are negligible in comparison with the scale of mining and recharge from rainfall.

The dewatering effect of opencast mining on the adjacent strata, because of groundwater influx into the pits, usually extends less than 40 m ahead of mining. This is substantiated by the necessity to perform pre-spitting during mining operations, to drain water from blasting holes ahead of high walls in many of the pits (Hodgson *et al.,* 1998).

The conclusion is that opencast mining is a significant consideration in the intermine flow equation, after they start decanting.

4 DATA ACQUISITION

Relevant data and detailed mine plans were obtained from individual collieries, regional mine offices, mine head offices, the Department of Minerals and Energy (DME) and the Department of Water Affairs and Forestry (DWA&F). Some of these data are confidential and are not reproduced in this report. These constitute, in some instances, detailed future mine plans.

Information was requested from the mines in hard copy and digital format. This included:

- Mine lease areas.
- Surface topography.
- Surface features including watercourses, evaporation dams, dumps and areas of subsidence.
- Thickness of the weathered zone.
- Extent of existing and future mining.
- Floor contour plans of all major coal seams.
- Geological structures including dykes, sills and faults.
- Elevations of water levels in boreholes in mined and unmined areas.
- Quantities and qualities of water.
- Abstraction and disposal methods of mine water.

Despite the fact that most of this information was readily available at the mines, it took the best part of 18 months to acquire it. Reasons for this vary, but in general, the following are prominent:

- Not all of the mines have approved EMPR's. Only some of these documents were made available by mines.
- A list of all the mines in the Mpumalanga Coalfield was obtained from the DME in Witbank. Not all the mines have submitted their EMPR's. Some of the submitted EMPR's were not available for viewing during visits to the DME.
- Floor-contour plans or coal seam elevations from prospect boreholes for the No. 2 Coal Seam were available for most of the mines. Floor contours for the other coal seams are not freely available. Some collieries do not include floor-contour plans in their EMPR's.
- The original surface topography, before mining, was digitised from 1:10 000 orthophotos. In isolated areas, these values were supplemented from 1:50 000 topographic maps.
- Watercourses were digitised from the 1:10 000 orthophotos, supplemented by information from the mines and the WR90 library from the WRC.
- Limited information was provided by the mines on localities of evaporation dams or coal discard dumps. This information is not essential for the current

investigation.

- Most collieries have not mapped their subsidence areas in detail. At best, mines could provide general outlines of subsidence areas.
- Limited information on final surface contours is available. Some of the mines have done detailed digital terrain modelling, which proved to be very useful information.
- Although the depth of weathering is available from prospect boreholes, this was not always supplied by the mines.
- Mine plans from the older collieries were generally drawn by hand and had to be digitised by the IGS. Some of the maps at the DME Office in Witbank do not show any co-ordinates (Figure 26). These maps were not included in the Geographic Information System for this project.

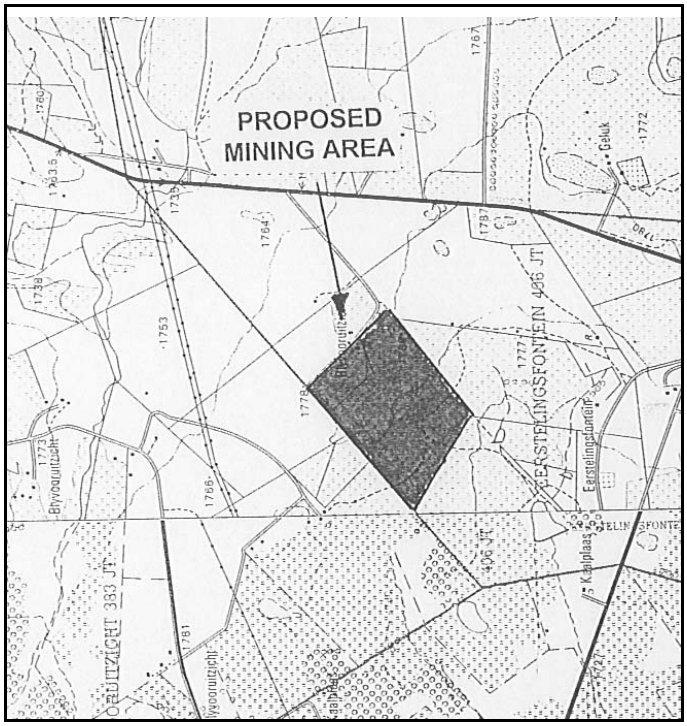


Figure 26. Typical map without co-ordinates, available at the DME in Witbank.

- Geological discontinuities that could eventually affect mine-water flow are few in the Lower Olifants Catchment Region. The Ogies dyke, striking east to west through the area, has not been mapped in detail, but its position is known with sufficient accuracy from mining plans. Other than this, major dolerite intrusions occur in the Goedehoop, Bank and Secunda Collieries, where dolerite sills transgress through the coal seams.
- Most collieries have monitoring boreholes. Some of this information has been entered by the collieries into HydroCom databases or Microsoft Excel spreadsheets. This made processing of the information relatively easy.

5 THE GEOGRAPHIC INFORMATION SYSTEM

5.1 INTRODUCTION

The WISH (Windows Interpretation System for Hydrogeologists) software package was selected for visualisation and interpretation of data. Reasons for this are as follows:

- WISH is easy to use and available from the Institute for Groundwater Studies (IGS).
- It consists of a map-drafting and display facility. Maps may be incorporated from other applications, such as through digitising, scanning, or from other GIS programs. Formats supported are: *.SHP, *.DXF, *.BMP, *.TIF and *.JPG.
- Data sets from Microsoft Excel may be superimposed on the maps. When it comes to relevant data processing, the processing power of WISH is unsurpassed by other software.
- Many options for data interpretation are included. The main categories are:

    - Spatial analysis using the combined map and data sets.
    - Time series analysis.
    - Specialised chemical diagrams.
    - Pumping test analysis.
    - Hydrogeological logs.
    - Contours and cross sections.

5.2 DATA HANDLING

Data is stored from two formats:

- Microsoft Excel for all data, except contour data.
- *.dat files for contour data.

Datasets for each of the mines have been kept separate. It is therefore possible to make available mine-specific datasets. WISH has the facility to interrogate more that one dataset simultaneously. All datasets for the Mpumalanga Coalfields, or only that of the selected databases, may therefore jointly be processed in WISH.

5.2.1 Data in Microsoft Excel

Data in Microsoft Excel is structured according to the requirements of WISH. This

necessitates the creation of several datasheets, one for each type of information to the entered. Examples of datasheets are Basic Information, Geology, Chemistry, Chemical Logs, Pumping Test Information and any other parameter that may be required.

Data structures within the datasheets are simple. The Basic Information sheet contains the site name and coordinates (Figure 27).

SiteName	Xcoord	Ycoord	Zcoord	SiteType	CollarHeight
63	17636.00	3500793.00	347.00	R	-1
B01	19125.00	3501905.00	360.00	B	0.03
B03	20966.00	3501394.00	367.00	B	0.03
B10	20109.00	3501794.00	365.00	B	0.02
4325CD00BH1	15413.00	3500812.00	346.00	B	0.02
4325CD00BH2	16186.00	3500718.00	347.00	B	0.02
4325CD00BH3	16882.00	3501217.00	346.00	B	0.03
4325CD00BH4	17929.00	3501858.00	350.00	B	0.03
4325CD00BH5	18672.00	3502442.00	361.00	B	0.03
4325CD00D28	18767.00	3502027.00	354.00	P	-1
4325CDDAM04	16939.00	3500228.00	344.00	Z	-1
RAIN GAUGE	18070.00	3502856.00	365.00	B	-1
SNAKE RIVER	15988.00	3500200.00	340.00	R	-1

Figure 27. Example of information required in the Basic Information sheet.

This information forms the essence of the data structure, without which no other information can be entered.

The chemistry datasheet may contain any of the measured chemical parameters. The column sequence is also not important. The only prerequisite is that the SiteName and DateTimeMeas fields be entered (Figure 28). This allows complete flexibility in terms of the range of parameters and the sequence of data entry. Calculations are allowed and can be done in additional columns.

SiteName	DateTimeMeas	pH	EC mS/m	TDS mg/l	Ca mg/l	Mg mg/l	Na mg/l	K mg/l	PALK mg/l	MALK mg/l	Cl mg/l	SO4 mg/l
S63	1/5/95 12:00 PM	5.04	142.1	1102	152	80	43	7.8	0	6	16	706
S63	1/10/95 12:00 PM	9.76	108.1	694	40	21	154	2.7	67	202	63	252
S63	1/17/95 12:00 PM	4.26	88.5	624	90	44	23	10.9	0	0	5	456
S63	1/24/95 12:00 PM	5.74	103.4	790	97	54	41	2.0	0	4	12	546
S63	1/31/95 12:00 PM	8.58	92.2	578	16	15	181	1.6	120	234	78	99
S63	2/14/95 12:00 PM	8.70	90.6	578	1	0	214	15.4	127	274	86	7
S63	2/20/95 12:00 PM	8.80	90.2	516	1	0	193	1.4	120	276	81	3

Figure 28. Example of the information in the Chemistry datasheet.

The geology is coded and full instructions are provided in the WISH Manual. An example of such a datasheet is included in Figure 29.

SiteName	DateTimeMeas	DepthTop	DepthBot	Lcode	Unit	PRIM_COI	SECO_CO	TEXTURE	PRIM_FEA	SECO_FE
BH1	1987/09/01 00:00	0.00	11.00	SOIL		C	Y		SN	LT
BH1	1987/09/01 00:00	11.00	16.00	SLSN		W	H	33		
BH1	1987/09/01 00:00	16.00	18.00	SNDS		W		33		
BH1	1987/09/01 00:00	18.00	23.00	SLSN		W	H			
BH1	1987/09/01 00:00	23.00	35.00	SNDS		W		44		
BH1	1987/09/01 00:00	35.00	40.00	SLSN		W	H			

Figure 29. Example of the information in the Geology datasheet.

Any of the time-dependent parameters, other that chemistry, are entered in their own

datasheets (Figure 30). These would typically be rainfall, water levels, tonnage of coal produced over time, discard tonnage, water influx into the mine, water pumped from the mine and many others.

SiteName	DateTimeMeas	WaterLevel	Comment
BHo1	2/1/95 10:00 AM	3.290	
BHo1	3/1/95 10:00 AM	3.430	
BHo1	4/1/95 10:00 AM	3.230	
BHo1	5/1/95 10:00 AM	3.120	
BHo1	6/1/95 10:00 AM	3.250	
BHo1	7/1/95 10:00 AM	3.390	
BHo1	8/1/95 10:00 AM	3.580	
BHo1	9/1/95 10:00 AM	3.650	

Figure 30. Example of the information in the Water-level datasheet.

In addition to these parameters, the data structure also caters for parameters that are measured in boreholes, such as geophysical or hydrochemical logs. An example is included in Figure 31.

SiteName	DateTimeMeas	Depth	DO mg/l	ORP	PH	SPCOND n	Temp C
BH2	1/19/98 12:00 AM	20.50	5.75	126.00	7.91	7.60	19.89
BH2	1/19/98 12:00 AM	20.70	5.75	128.00	7.90	7.40	19.88
BH2	1/19/98 12:00 AM	20.90	5.77	130.00	7.89	7.20	19.72
BH2	1/19/98 12:00 AM	21.20	5.78	132.00	7.87	7.20	19.46
BH2	1/19/98 12:00 AM	21.40	5.17	134.00	7.84	7.10	19.19
BH2	1/19/98 12:00 AM	21.70	4.66	136.00	7.80	7.10	18.94
BH2	1/19/98 12:00 AM	22.00	4.28	138.00	7.77	7.00	18.70
BH2	1/19/98 12:00 AM	22.30	4.14	139.00	7.74	7.10	18.45
BH2	1/19/98 12:00 AM	22.70	4.42	141.00	7.71	7.00	18.23

Figure 31. Example of the information in a Hydrochemical Log datasheet.

It is safe to say that almost any type of mining-related data can be handled by WISH, in conjunction with the Excel datasheets.

5.2.2 Data in *.dat files

For contouring, WISH requires the raw data and not pre-contoured information. Geological information should be extracted as x,y,z-information and saved as *.dat files. These files may have multiple columns (Figure 32).

SiteName	x-coord	y-coord	SurfElev	Depth	Softs	SoftsElev	SNDSTop	SNDSBot	SHLEBot	CoalBot
B101	-92648	2957191	1434.10	74.36	0.00	1434.10	1398.76	1388.75	1376.15	1369.37
B102	-92428	2957074	1432.50	72.74	0.00	1432.50	1400.70	1390.00	1376.73	1369.90
B103	-92382	2957062	1432.20	68.50	11.23	1420.97	1401.55	1391.35	1378.05	1370.49
B104	-92278	2956972	1432.40	38.30	10.19	1422.21	1421.43	1416.10	1405.00	1398.87
B105	-92327	2957198	1434.60	84.49	14.42	1420.18	1394.05	1383.44	1370.03	1360.50
B106	-92272	2957139	1433.10	66.22	12.31	1420.79	1401.68	1392.58	1380.37	1373.90
B107	-92227	2957133	1433.90	55.12	11.70	1422.20	1409.83	1404.08	1393.60	1387.25
B108	-92185	2957123	1435.20	56.13	11.46	1423.74	1411.82	1402.72	1391.54	1386.10
B109	-92138	2957100	1435.10	64.02	0.00	1435.10	1411.20	1400.94	1388.90	1381.23
B110	-92105	2957108	1435.40	67.79	12.74	1422.66	1408.88	1399.60	1387.97	1380.73
B111	-92072	2957070	1435.10	63.67	14.21	1420.89	1409.15	1399.25	1387.40	1380.43

*Figure 32. Example of the information in a *.dat file for contouring.*

Any, or all of the columns may be contoured. Calculations are allowed in the contouring package of WISH, which facilitate contouring of features such as coal

volume extracted or water volume in a specific layer.

5.2.3 Information on maps

Maps may be in degrees or in x,y co-ordinates. Mines usually provide their mine plans in the x,y-system. WISH has a conversion facility between the two systems.

Bitmaps, such as the 1:50 000 topographic maps, may be imported and displayed as background to the mine maps. This feature is particularly useful in the location of underground workings in relation to surface features (Figure 33).

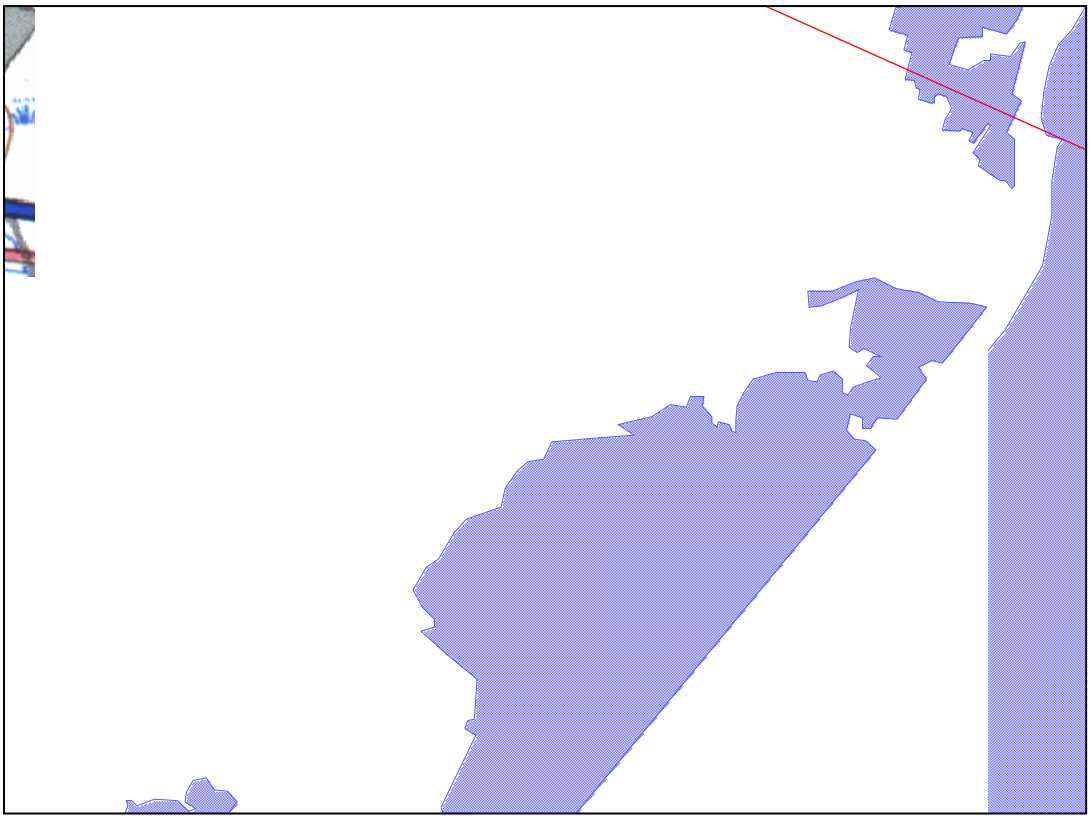


Figure 33. Example of underground mine workings superimposed on 1:50 000 topographic information.

Many other cartographic features are available in WISH. Some of these are:

- Area calculations; measure distances; overlays; draw, text and map editing.
- Unlimited zoom on NT and 2000 computers.
- Contours and sections with elevation read-out.
- Flip, mirror and offset, to correct false co-ordinate systems.

While all these features are handy in the compilation of maps and interpretation thereof, it is particularly the last one that proves very valuable. Many of the mines are working in false co-ordinate systems and their maps need to be offset, flipped or

mirrored to fit onto a true projection for the area.

6 REGIONAL MINE PLANS

A series of regional plans have been compiled from all the information received during this investigation. These are shown in Figure 34 - 40. These maps are also available in *.ws2 format, which is the format supported by WISH. It is difficult to see detailed features in the figures of this document. The files included on the CD-ROM should be opened on a computer. This will allow zoom and pan to see detail.

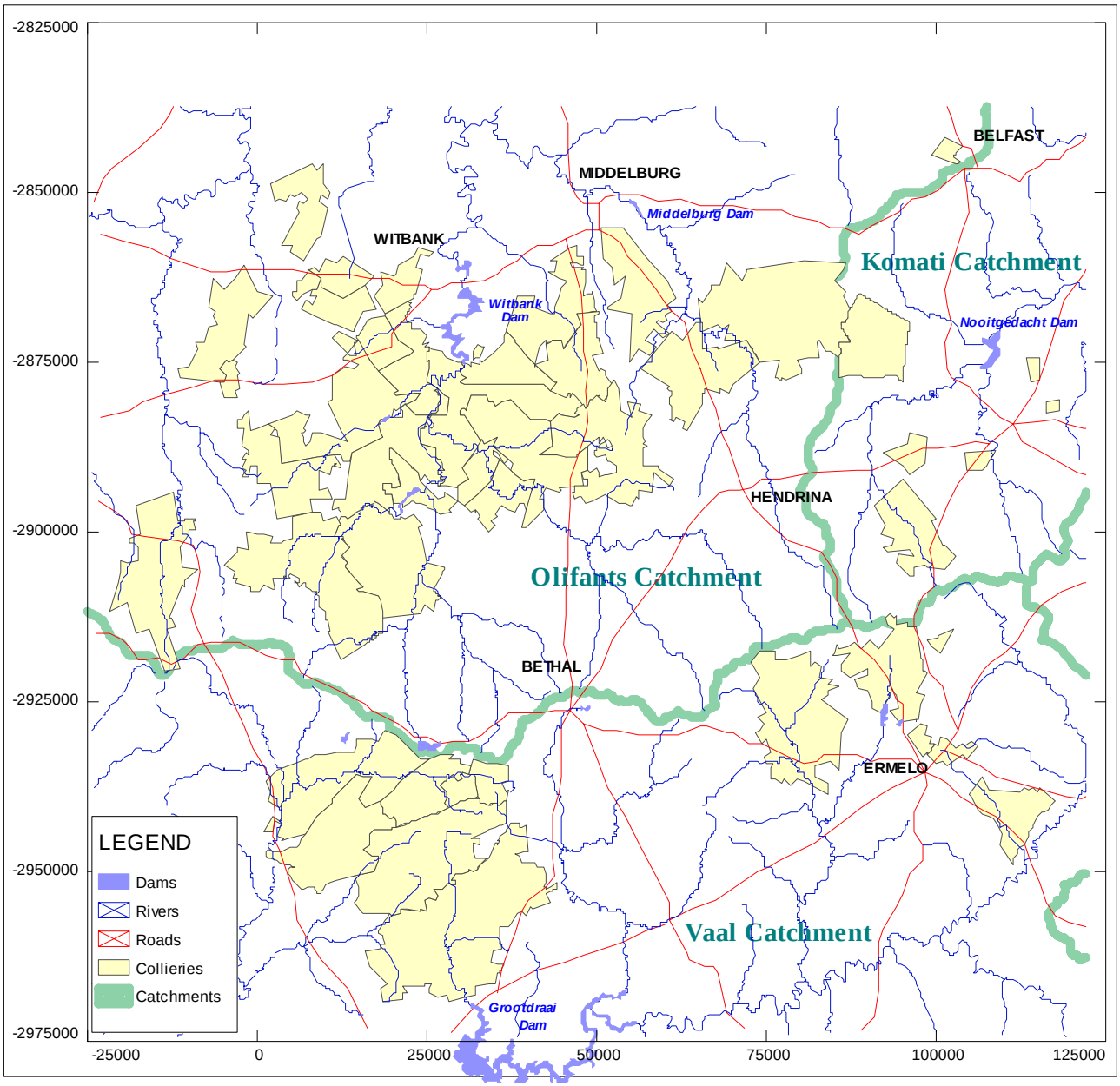


Figure 34. Locality plan of the mine lease areas in the Mpumalanga Coalfields.

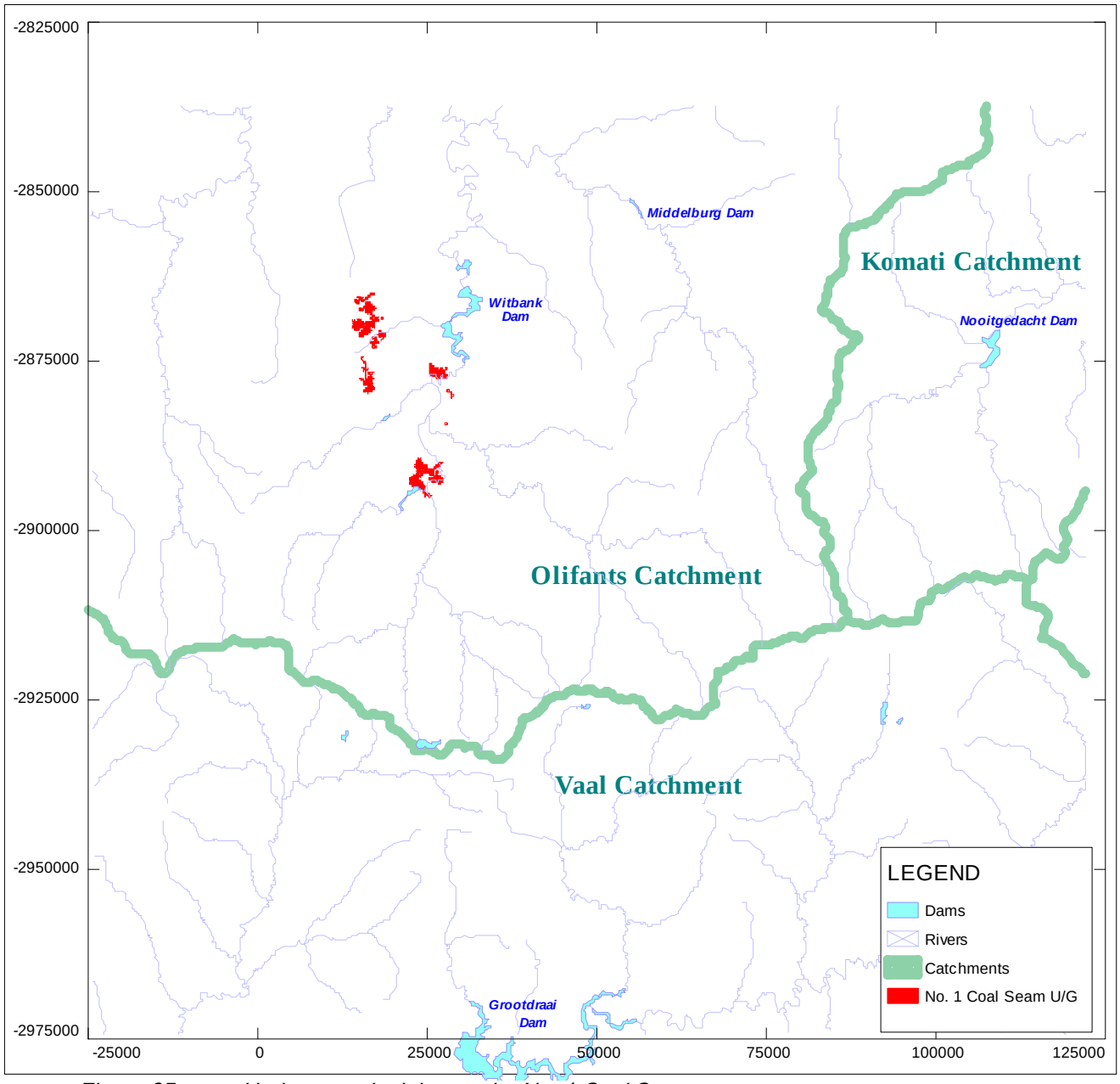


Figure 35. Underground mining on the No. 1 Coal Seam.

Statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)
U/G 1 Seam	2525	0	2525	2.5	60.00%	38	25	0	0	25

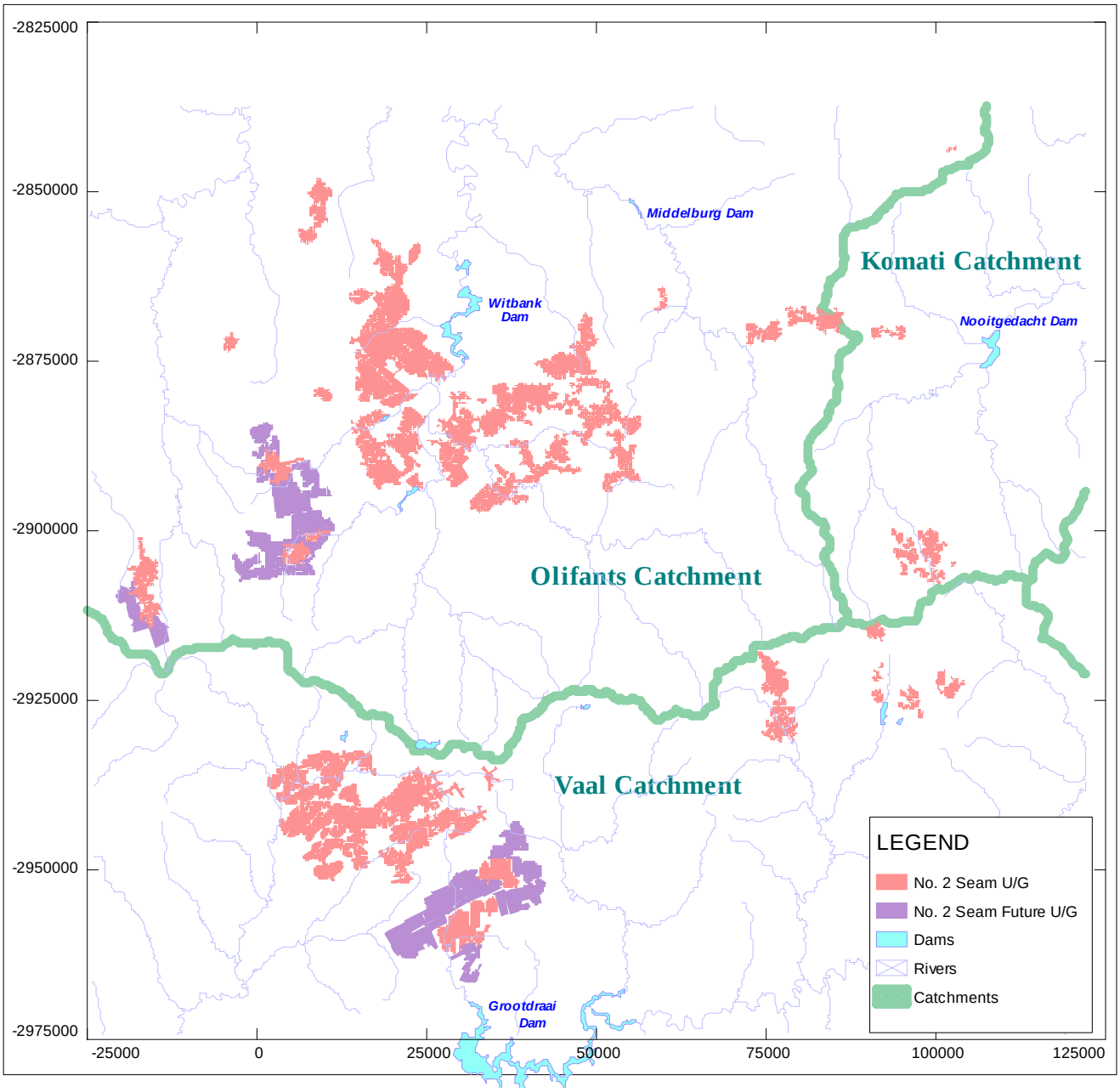


Figure 36. Underground mining on the No. 2 Coal Seam.

Statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)
U/G 2 Seam	98549	39695	138244	3	65.00%	1922	1281	774	516	1797

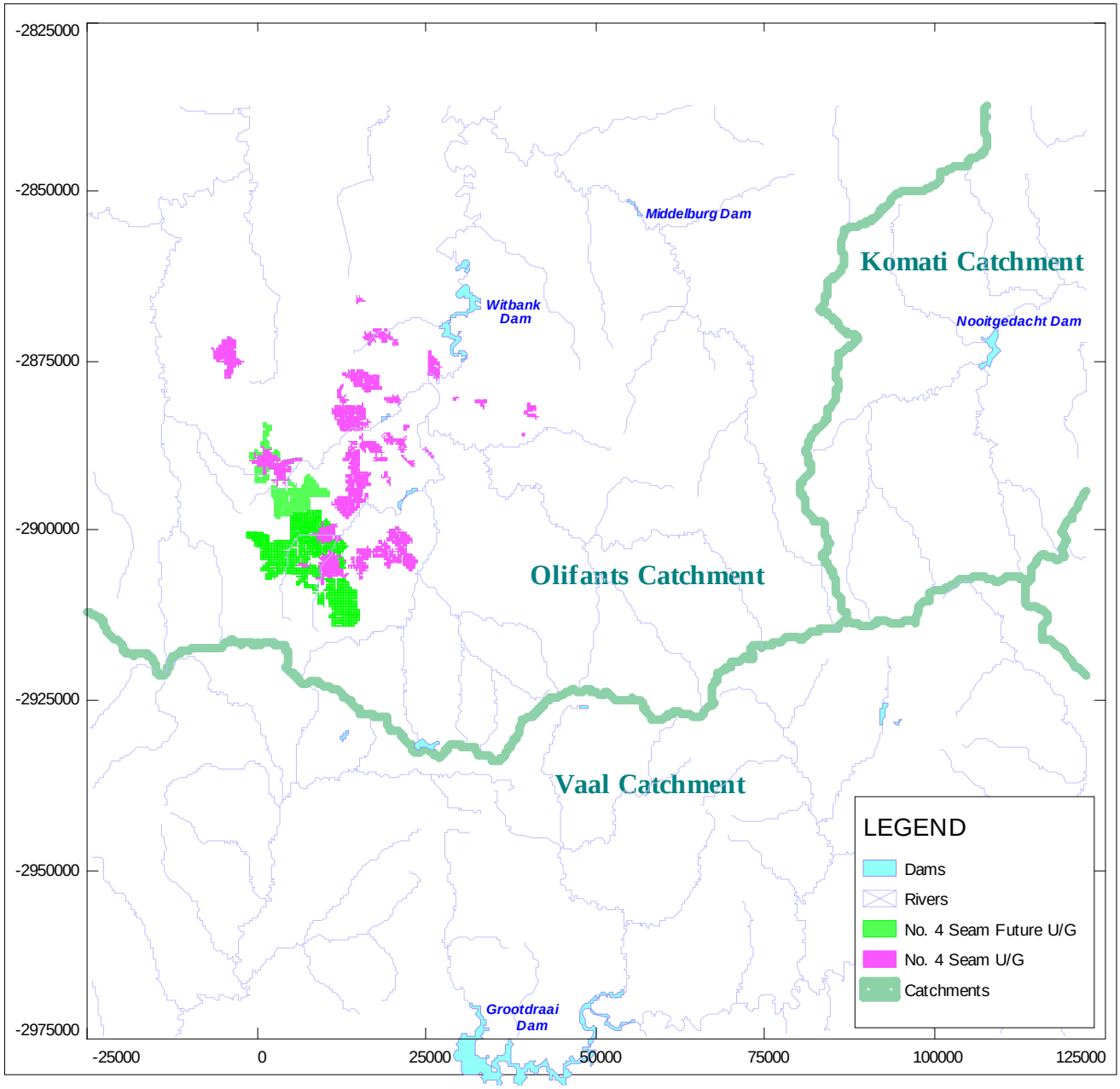


Figure 37. Underground mining on the No. 4 Coal Seam.

Statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)
U/G 4 Seam	13485	2833	16318	3	65.00%	263	175	55	37	212

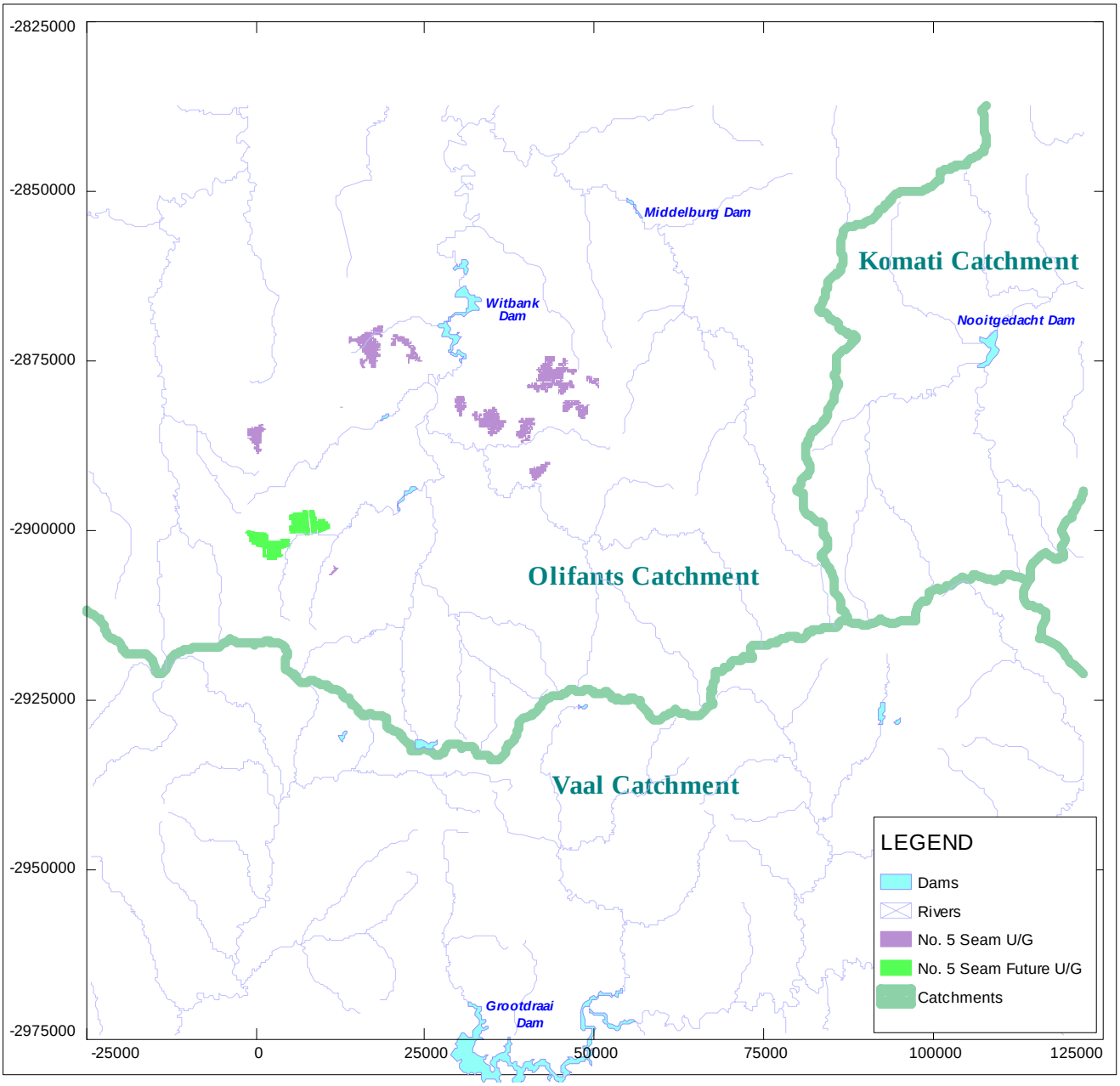


Figure 38. Underground mining on the No. 5 Coal Seam.

Statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)
U/G 5 Seam	7842	7051	14893	2.5	65.00%	127	85	115	76	161

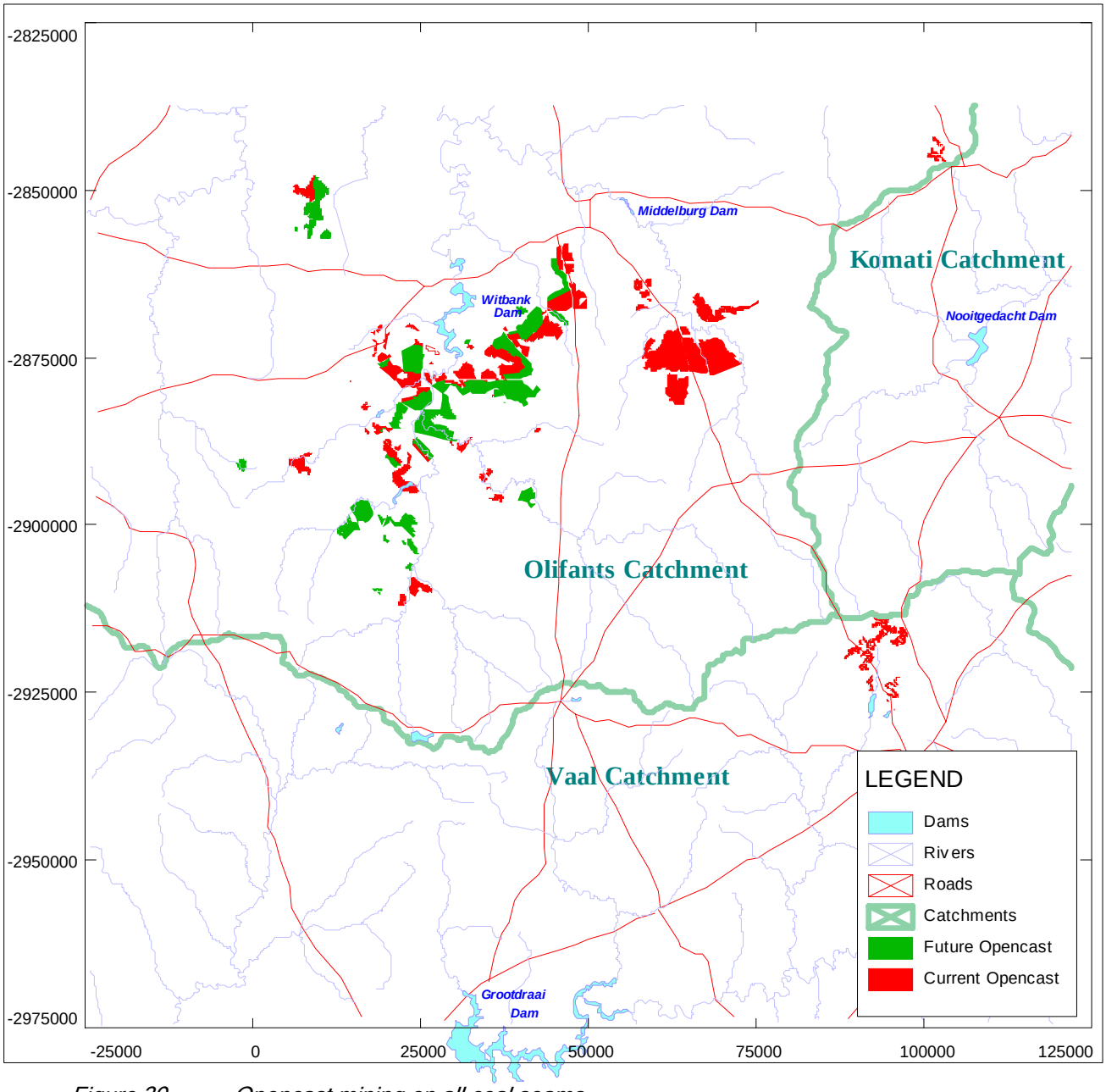


Figure 39. Opencast mining on all coal seams.

Statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)
Opencast	23557	14480	38037	3.5	90.00%	742	495	456	304	799

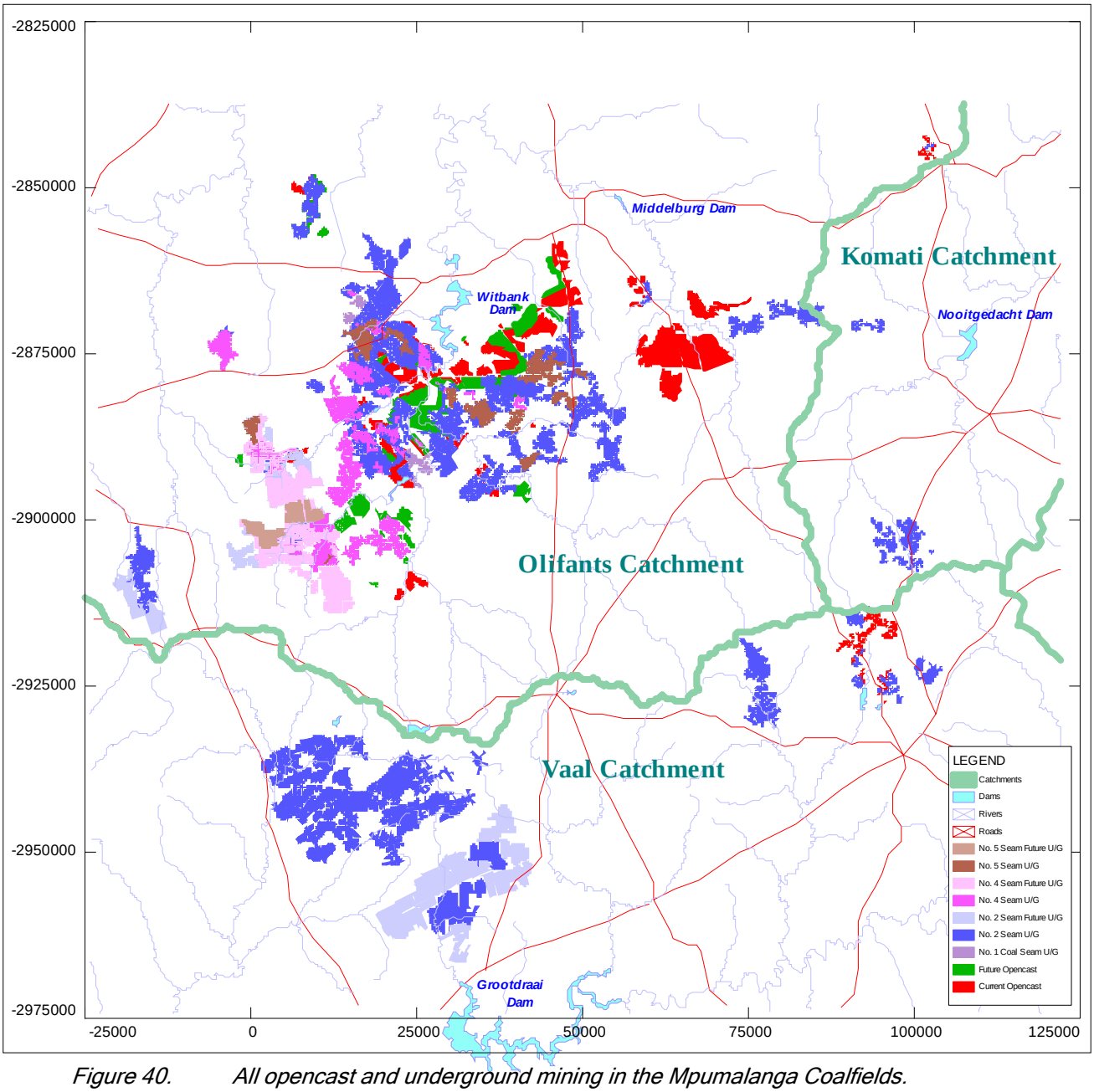


Figure 40. All opencast and underground mining in the Mpumalanga Coalfields.

Combined statistics are:

Mining type	Current mined area (ha)	Future area to be mined (ha)	Total area to be mine (ha)	Extraction height (m)	Extraction rate %	Volume coal mined (Mm3)	Tonnes coal mined (Mt)	Future volume coal to be mined (Mm3)	Future tonnes coal to be mined (Mt)	Total planned production (Mt)	% from each horison or method
U/G 5 Seam	7842	7051	14893	2.5	65.00%	127	85	115	76	161	5.4
U/G 4 Seam	13485	2833	16318	3	65.00%	263	175	55	37	212	7.1
U/G 2 Seam	98549	39695	138244	3	65.00%	1922	1281	774	516	1797	60.0
U/G 1 Seam	2525	0	2525	2.5	60.00%	38	25	0	0	25	0.8
Opencast	23557	14480	38037	3.5	90.00%	742	495	456	304	799	26.7
Totals	145958	64059	210017			3092	2061	1400	933	2995	

All information in the preceding diagrams are available as part of this project. The WISH software to interrogate this information may be purchased from www.uovs.ac.za/igs/wish. The detail on the maps can only be seen by running the

software and zooming in on areas of interest (Figure 41).

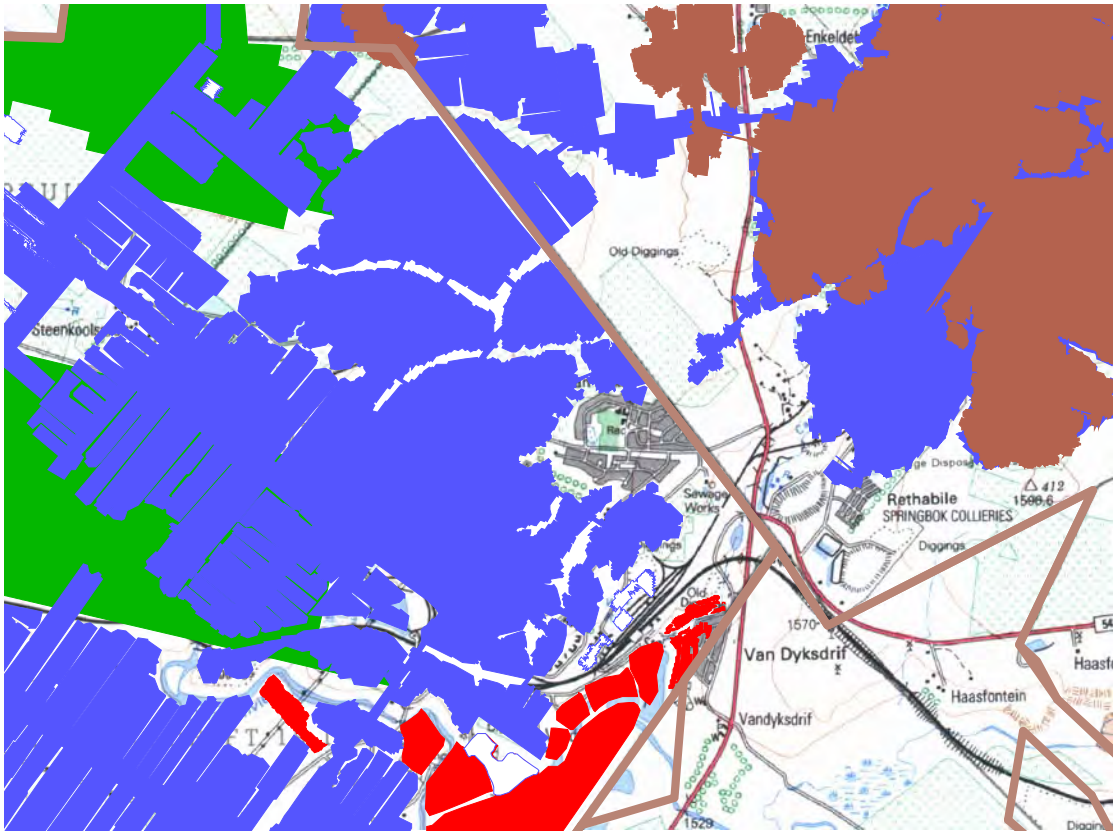


Figure 41. Example of a zoomed area (Vandyksdrift) with the 1:50 000 topographic map in the background.

The computer requirement to run this software is as follows:

- 1GH processor..
- Microsoft NT or XP.
- CD-Rom for the installation of the software and database.
- Hard disk with at least 50 Mbytes spare capacity.
- 256 Mbytes RAM

Note: To incorporate bitmaps and contours, considerably larger memory is required - 512 Mbytes are recommended for easy use.

7 COAL FLOOR CONTOURS AND MINE-WATER FLOW

Water in mined-out areas will flow along the coal floor and accumulate in low-lying areas. This is a simplistic view of the situation. In reality, water would accumulate in many isolated areas, where it would dam against barriers of coal or dolerite dykes left

in place (Figure 42). This is true for all types of mining.

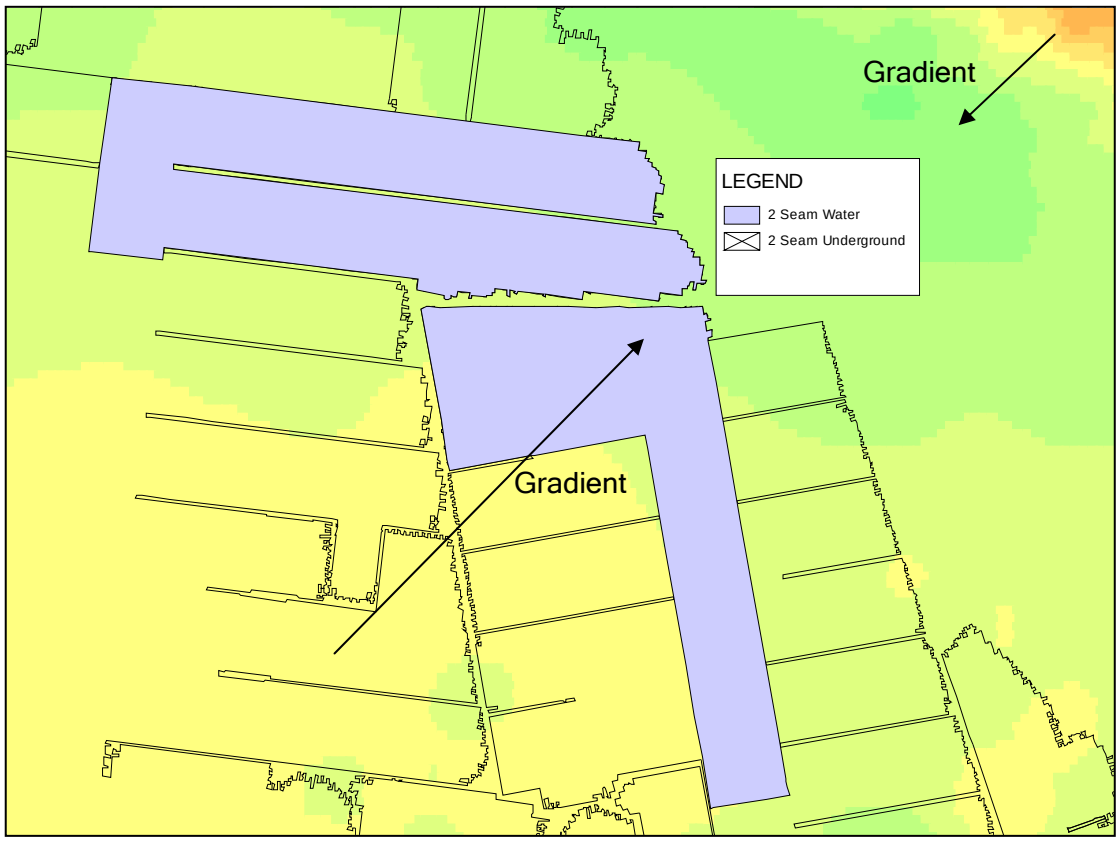


Figure 42. Example of water occurrences in underground bord-and-pillar areas, also showing floor contours. Scale from left to right is 2 700 m.

It is clear from the diagram that water accumulations are often contained artificially in mines. The regional groundwater gradient will come into effect only when the mines have completely filled up with water. Natural or artificial barriers in the mine will still exist; restricting mine-water flow, but the overall flux will be in the direction of the decanting points.

In most mines, significant undulations in the coal floor are present. These may be in the form of small variations in slope, or large regional structures. Examples of regional and local coal-floor contours are included in the figures that follow. Conclusions are provided with each diagram.

Figure 43 shows the No. 2 Coal Seam floor elevations for the whole of the Mpumalanga Coalfields.

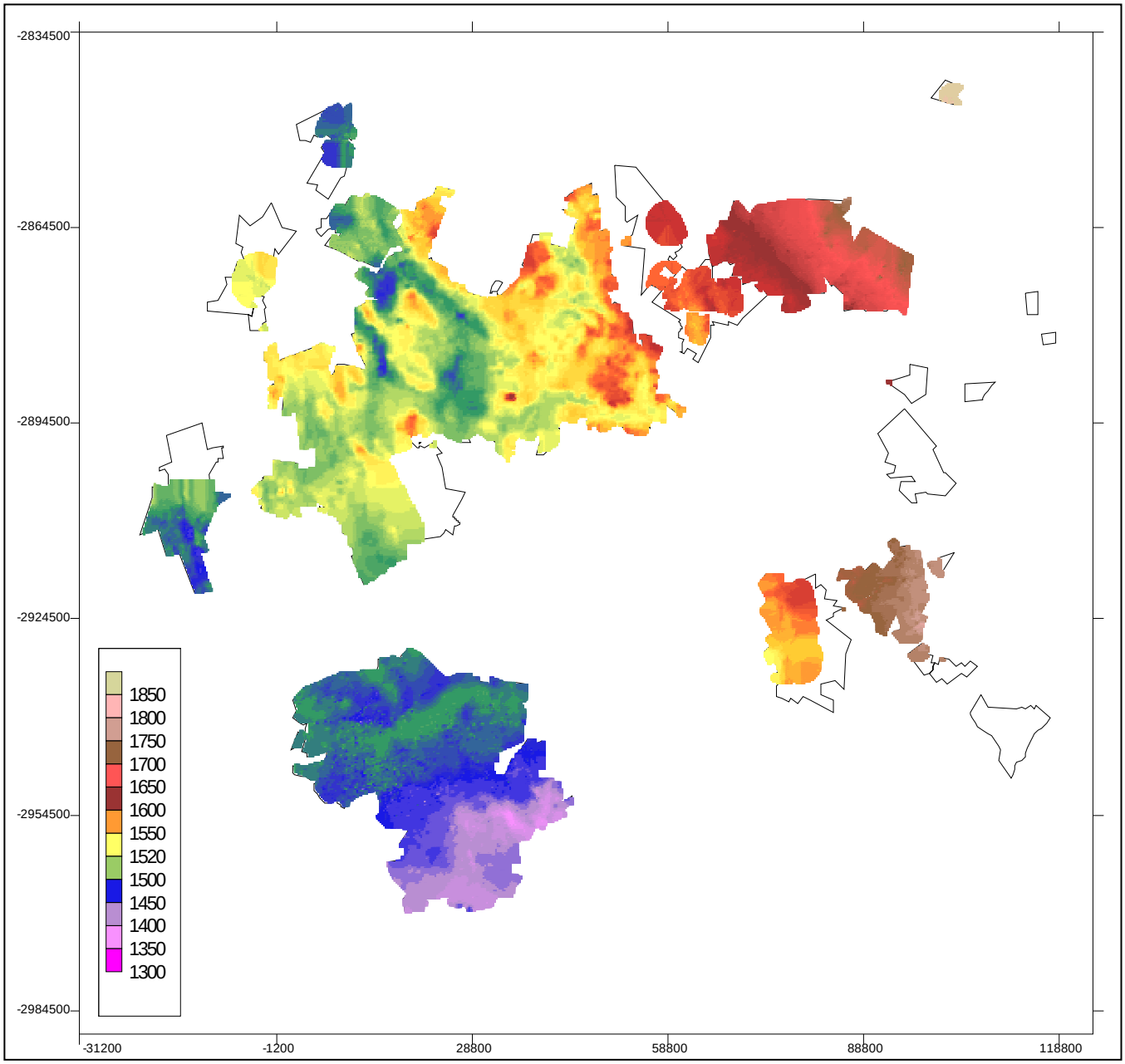


Figure 43. Floor contours elevations for the No. 2 Coal Seam in the Mpumalanga Coalfields.

These contours ignore local anomalies, such as dykes, sills and faults. Sills in particular, commonly have displacements of up to 30 m in Goedehoop, Bank, TNC and the Secunda Collieries. These would certainly have local impacts on mine-water flow directions, but will not alter the regional picture significantly.

Another important issue is the name “No. 2 Coal Seam” as used in this document. This name is not commonly used in Secunda and Ermelo Areas. It is merely used in this report to indicate the bottom major coal seam that is mined in the area. It does not imply a lithological correlation with the No. 2 Coal Seam in the Witbank area.

The conclusion is that significant elevation differences exist across the coalfields. Paleo-drainage channels and regional highs can be recognised (Figure 44). These are mainly north to south trending in the Witbank area, and east to west in the Secunda area.

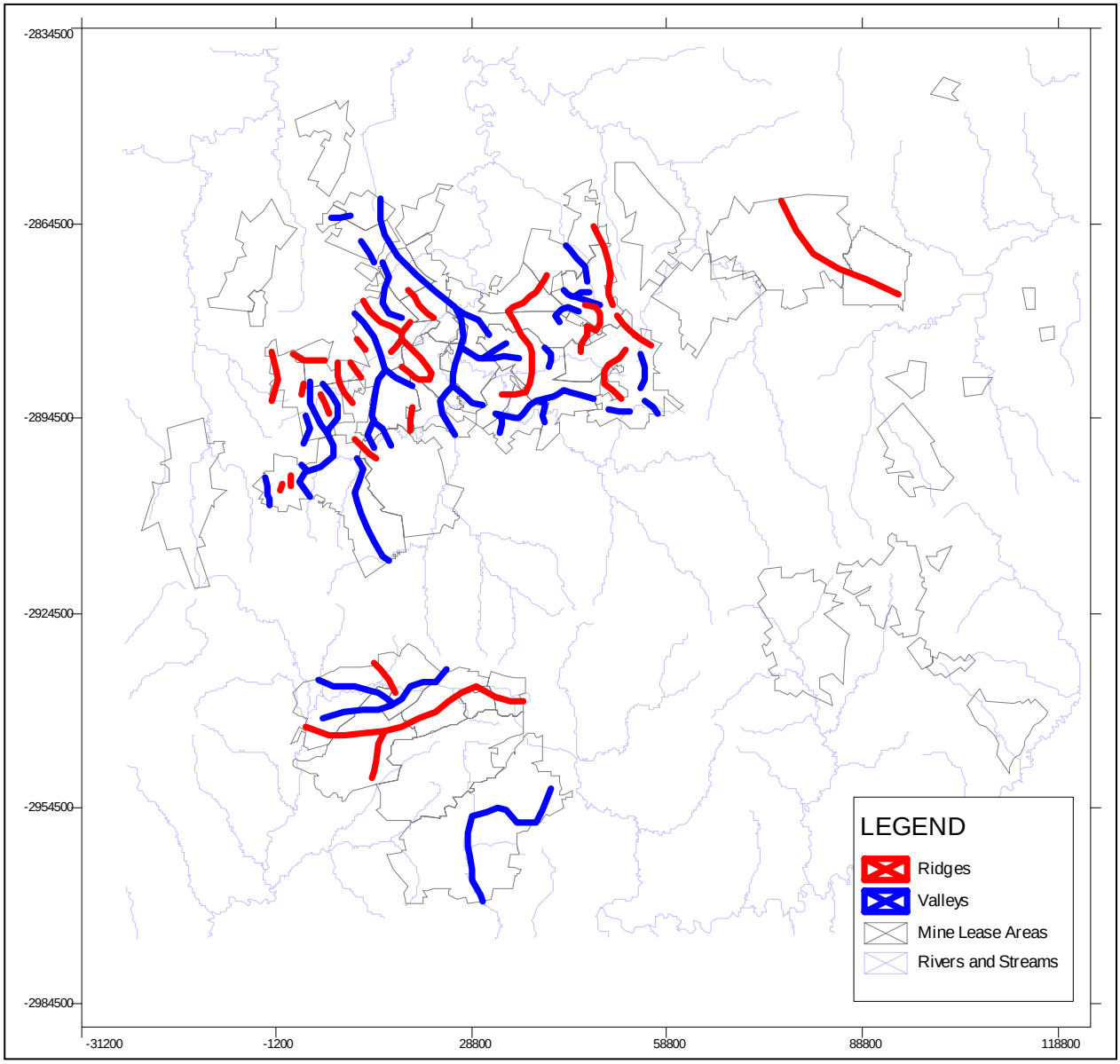


Figure 44. Regional lows (blue lines) and highs (red lines) on the No. 2 Coal Seam floor elevations.

For general interest, the current surface drainage pattern has been superimposed in this diagram. The fact that many of the drainage directions in the two systems coincide approximately, is purely coincidental. About 300 Ma separate the two systems.

The surface drainage system is obviously important in intermine flow management, because topographically low areas would be the areas where decanting from mines are expected. The most vulnerable areas would be areas where connections between mines and the surface occur, and where this coincides with a surface low.

Such areas are expected to be along the major rivers in the area.

In view of the small scale of the two preceding diagrams and the intensive mining in the area south of Witbank, this area has been enlarged in the diagrams that follow. The CD-ROM that accompanies this report allows the reader to zoom in on any area.

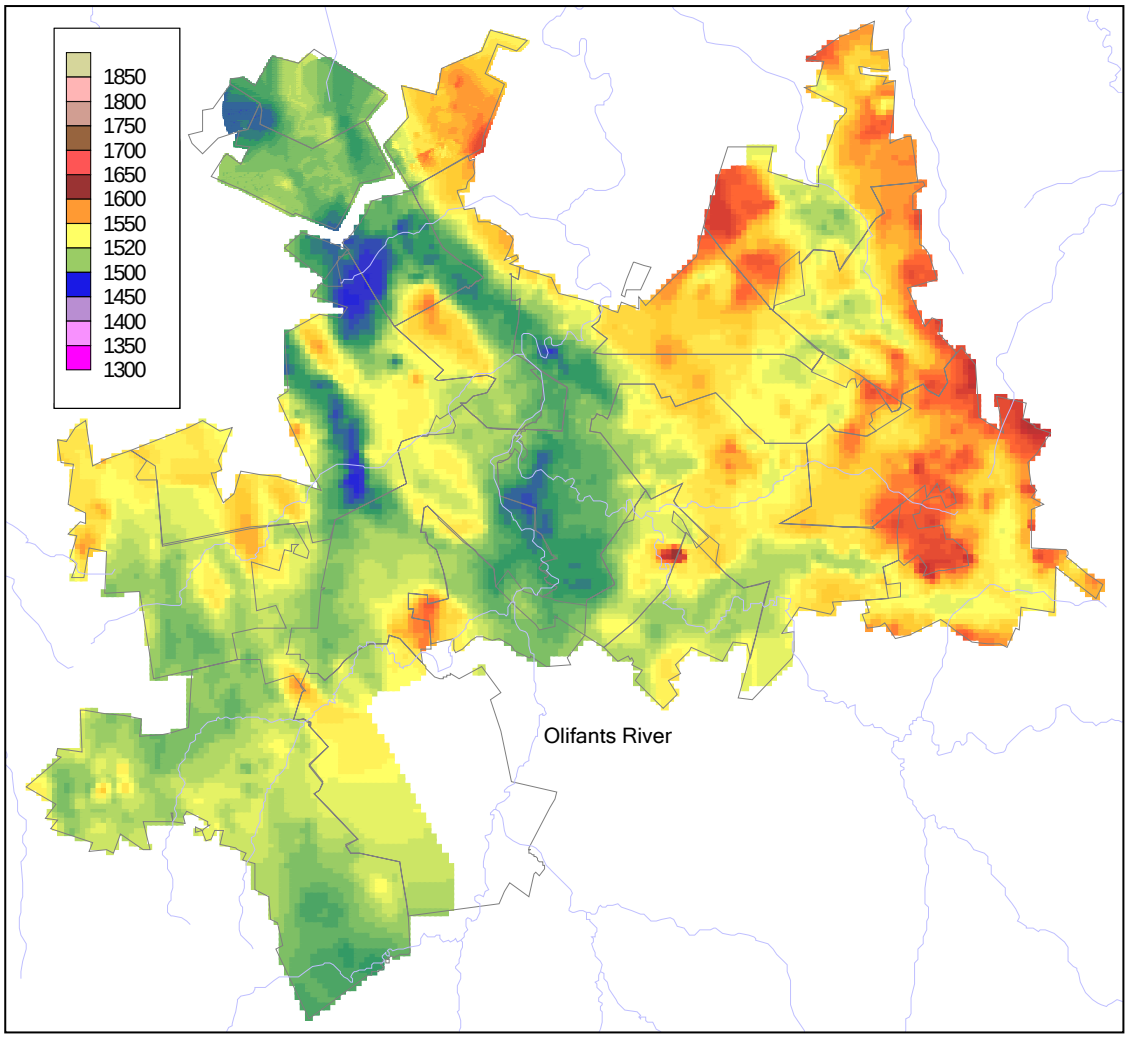


Figure 45. No. 2 Coal Seam floor contours for the collieries in the Witbank area. Scale from left to right is 67 km.

The co-existence of the Olifants River and a major low in the floor of the coal seam would be the obvious target where decanting onto the surface will be a major long-term problem. Almost 50 % of the mine water from this area could flow through the mines and barriers towards this topographic low. The surface elevation of the Olifants River in this area lies between 1 505 – 1 510 mamsl. Taking 1 505 mamsl, at the start of the Witbank Dam, as the main decant elevation, Figure 46 has been compiled. This suggests that only about 25% of the coal seam in this area lies below the decant level. This has serious implications in terms of flooding as much of the coal seams as possible. Flooding is an essential water quality control mechanism that excludes oxygen. This demonstrates the importance of having sufficient barrier pillars between the mines, to retain as much water as possible, thus flooding as much as possible.

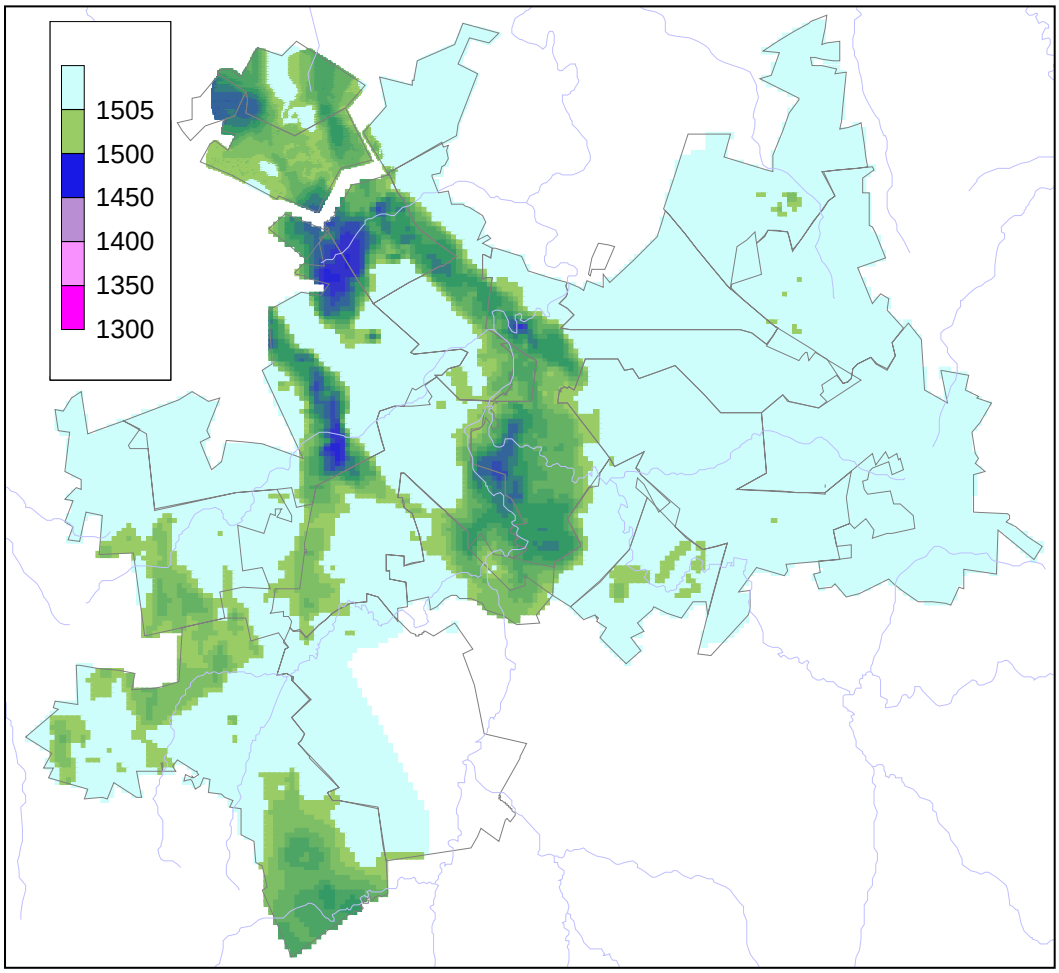


Figure 46. No. 2 Coal Seam floor elevations below the possible decant level of 1 505 mamsl in the Olifants River.

From this diagram, the first impression is that three isolated pools of water will exist at equilibrium conditions in the mines. This is an over-simplification of the situation and not valid on a more detailed scale. Figure 47 shows the many paleo-highs and -lows in the area. If one would zoom in further, many more such structures would be present. It is estimated that so-called “dry mines” would actually contain about 15% water in isolated areas, during the operation of the mine. This water is usually present in small puddles or in larger secluded areas.

In terms of the orientation of the highs and lows in Figure 47, one could conclude that the paleo-drainage was actually to the north. This would contradict previous beliefs that the sediments were deposited into the sedimentary basin from the north. To circumvent this problem of interpretation, it is suggested that the paleo-channels actually fan out from the north into a delta. It is beyond the scope of this report to pass any further judgment on this topic.

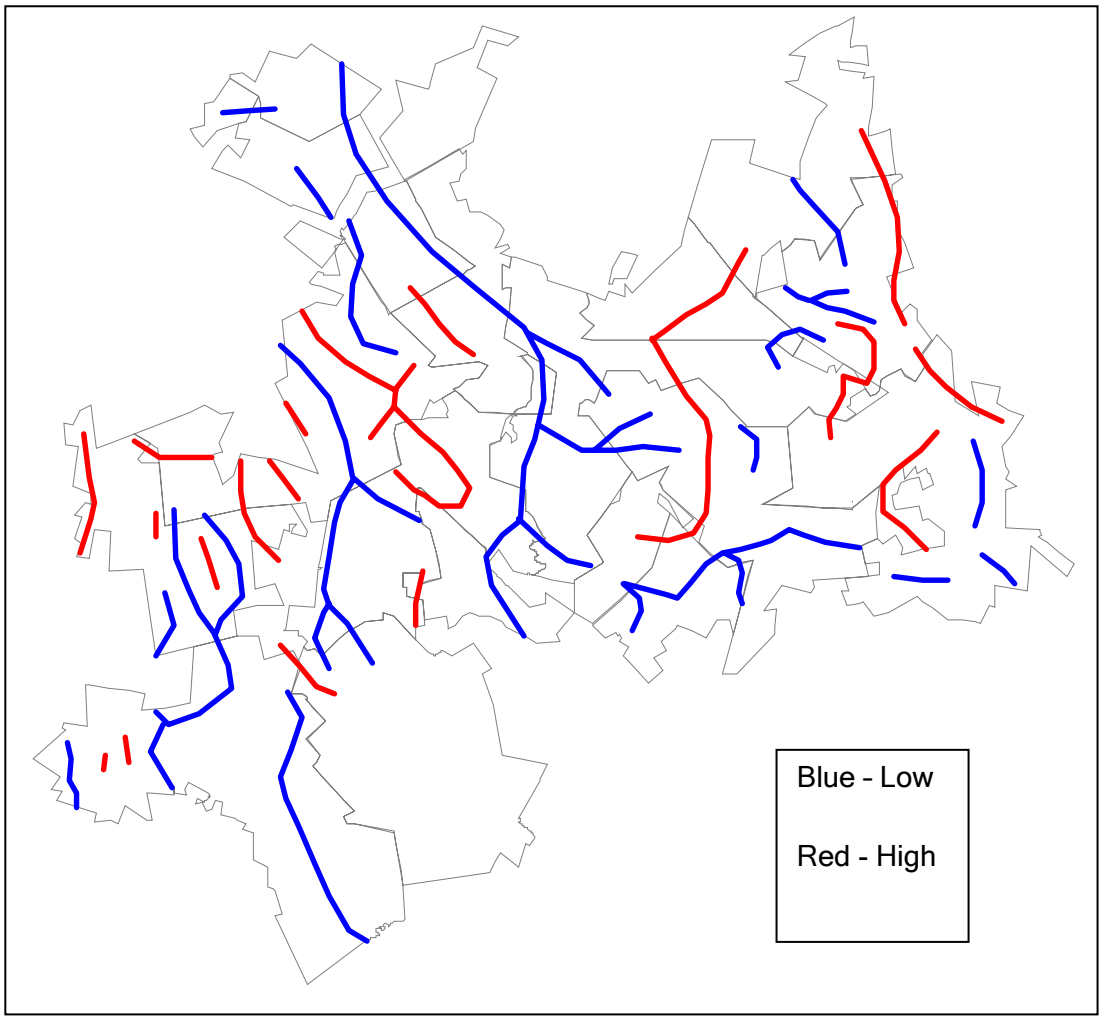


Figure 47. Paleo-highs and -lows on the No. 2 Coal Seam horizon in the Witbank area.

One of the prime questions is the intermine flow on the coal seam horizon between mines. The above information, would, for a start, suggest directions for mine-water drainage. Mine barriers would obviously play a role in possible water migration and so would the sequence in which mines fill up with water. The regional flow pattern will, however, not come into effect before all the mines have filled up with water. Two periods must therefore be considered:

- The period of active mining, including closure, up to the stage before steady state is reached in the water levels.
- The period after steady state has been reached.

Both these situations will be discussed in the section that follows. Obviously, many short-term and interim gradients will develop while mining is in progress. The net result, after steady state has been reached, is demonstrated in Figure 48.

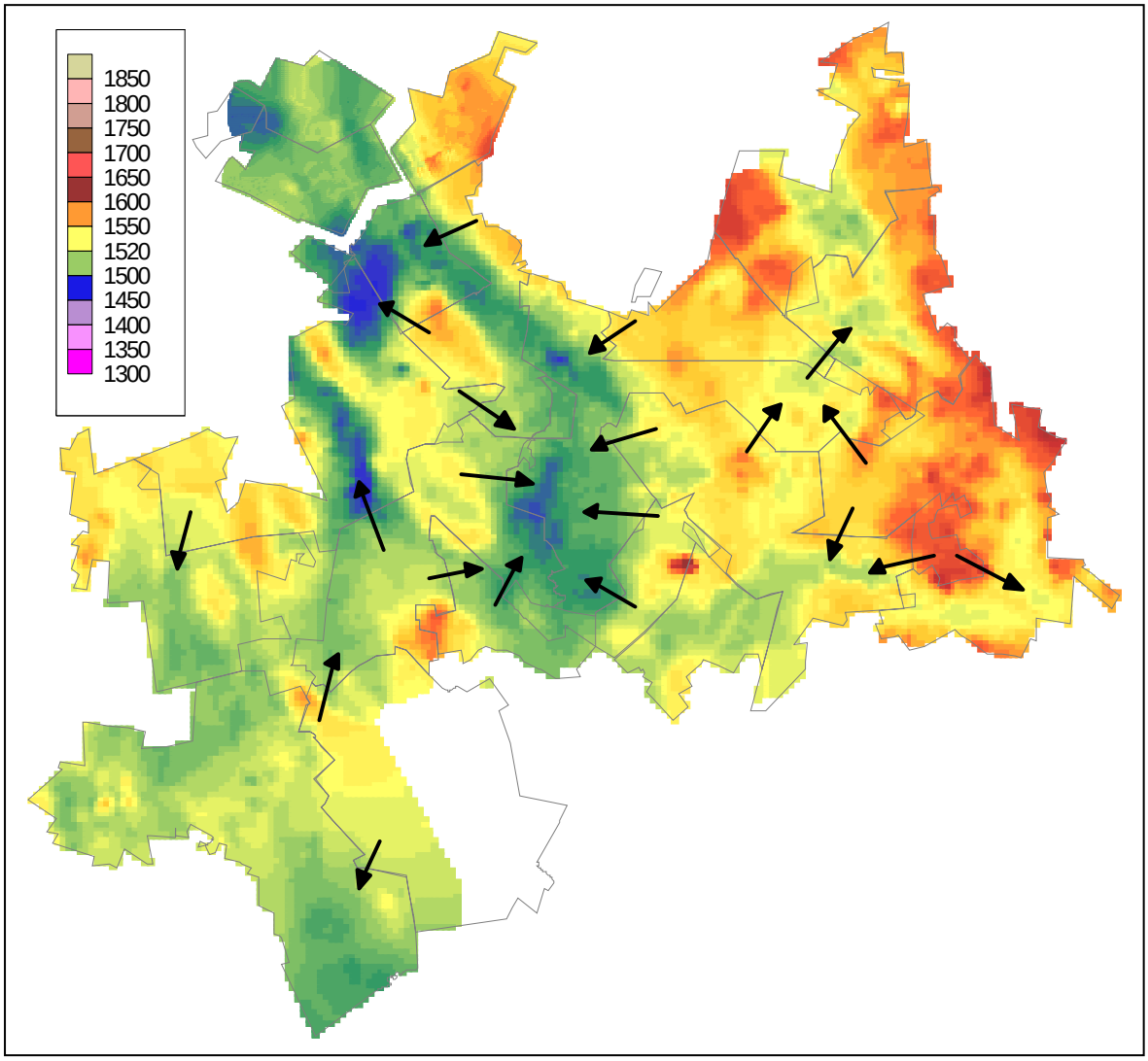


Figure 48. Major areas of possible intermine flow between collieries after full recovery of the water levels in all the mines.

By simply looking at the coal seam gradients and paleo-drainage channels, areas and pressure directions can be identified where mine water would, at some stage in the future, pressurise the barrier pillars between two mines. The amount of seepage though the barriers will depend on the groundwater gradient across the system and its hydraulic conductivity. It is trivial to calculate the seepage, once these parameters are known. However, to date only one of the collieries have done hydraulic testing in a barrier pillar. This is an obvious shortcoming in the quantification of possible intermine flow on the coal seam levels.

As an alternative, the values by Hodgson (1981) may be used. These tests were done at Ogies, Klippoortje and WCCM collieries. An average hydraulic conductivity for the coal of 0,1 m/d has been reported.

As far as the hydraulic gradient is concerned, this depends on the difference in water level in two adjacent collieries and the barrier thickness. In the general sense, it is a requirement that at least a 30 m barrier pillar should be left between adjacent

collieries. In practice, this distance may be very little at some of the collieries. As part of this study, mine maps were obtained where the mine workings in two adjacent mines plotted on top of each other. This is, of course, not possible. In the case demonstrated in Figure 49, subtle changes in the mine lay-out of the underground workings were necessary, where it borders onto the opencast.

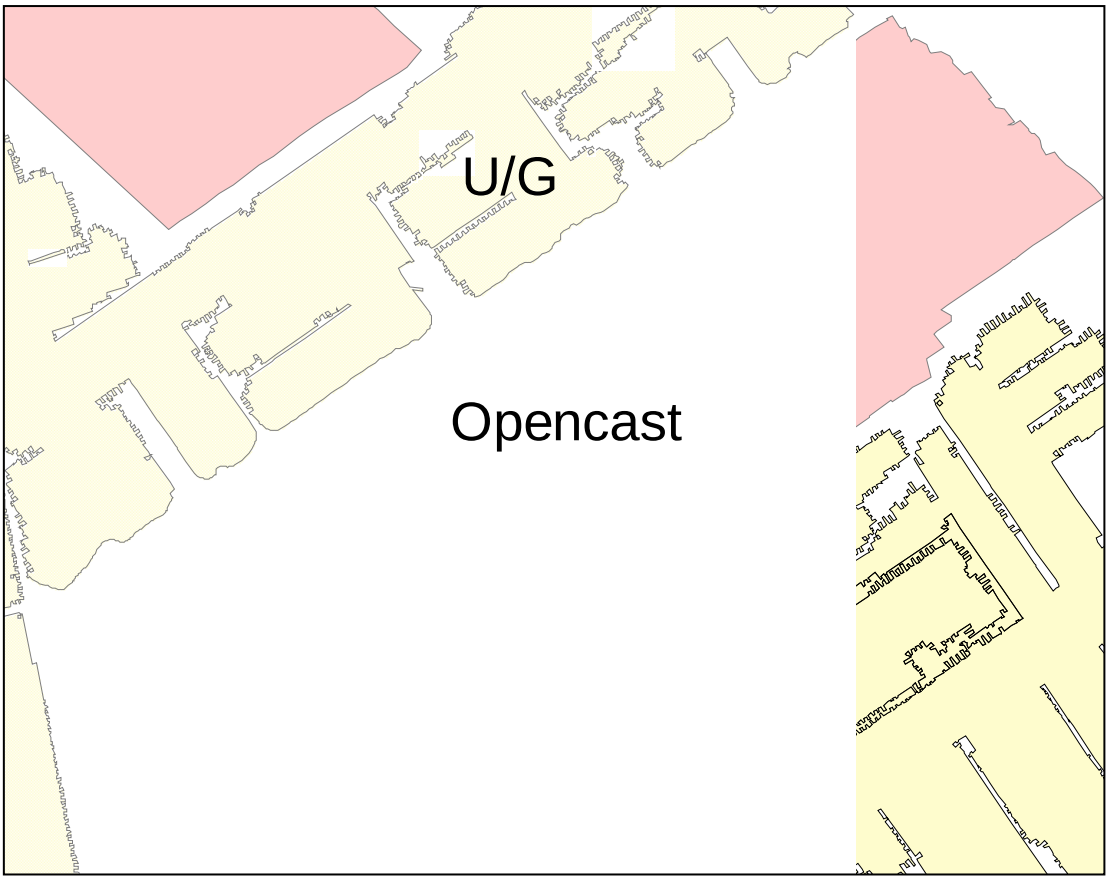


Figure 49. Example of a complex arrangement of underground an opencast mining in close proximity of each other.

In this particular instance, water passes freely from the opencast to the underground. Water levels in both entities are currently at the same levels. Recharge on the opencast totals 20% of the annual rainfall, while the underground workings receive only minimal recharge through groundwater seepage. The flux gradient is therefore from the opencast to the underground. When both mines are eventually full of water, this gradient will reverse, since the decanting point for the total system lies to the east of the opencast.

An even more complex arrangement is that illustrated in Figure 50.

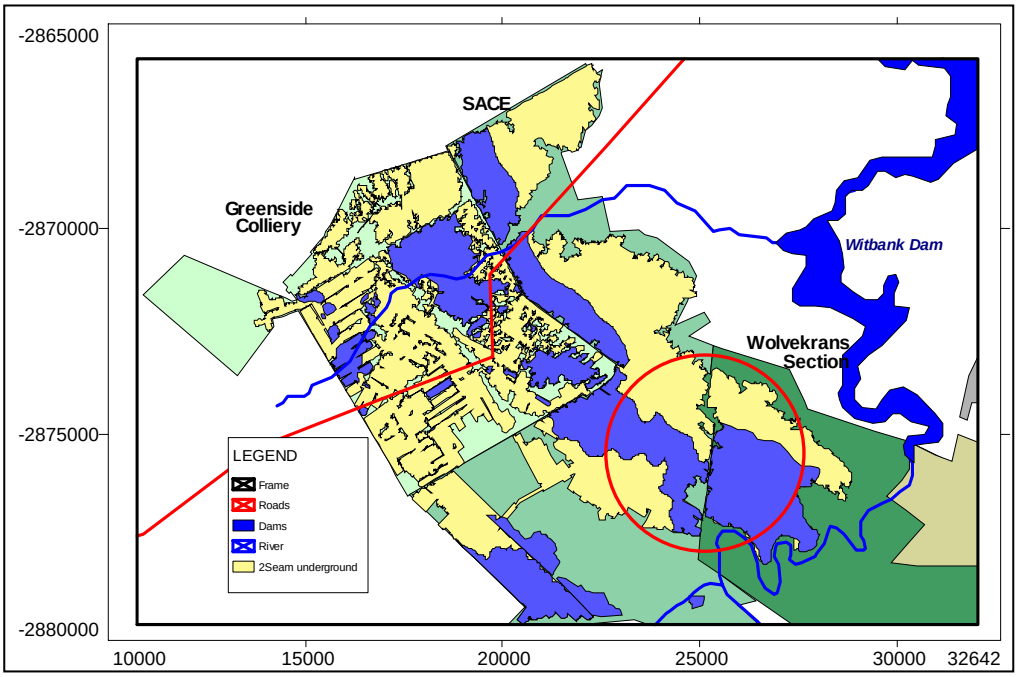


Figure 50. Complex arrangement of mining, underground water bodies and surface water.

The red circle on the diagram highlights the area of interest, where intermine flow currently occurs.

In this case, the barrier pillar varies considerably in thickness (Figure 51), ranging from 30 – 600 m. The calculation of the seepage rate through such an irregularly shaped boundary is complex and can only be done accurately through finite element modelling (Figure 51). The theory behind certain principles is presented in Appendix A.

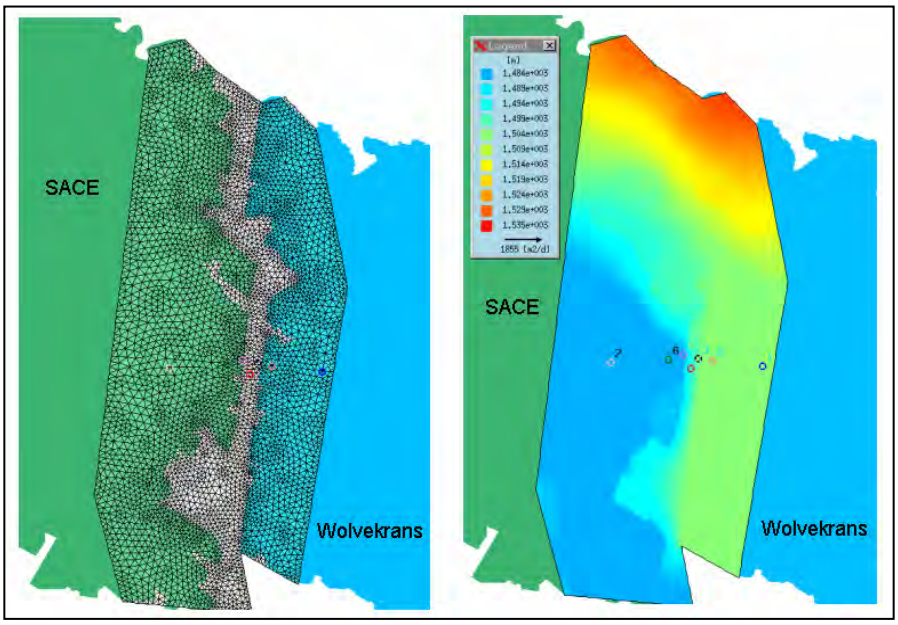


Figure 51. Finite element network (left) and simulated water pressures across the mine boundary (right).

A water pressure of up to 2.3 bars is exerted on the eastern side of the boundary

pillar. Hydraulic conductivity tests suggest an average value of 4 m/d for this portion of the barrier. This value is high compared to that reported by Hodgson (1981), but not unlikely. It is known from underground observations west of the barrier, that significant seepage is taking place from the east.

Based on the finite element simulation, a flux of 1.6 ML/d is predicted. This is one of the few instances in the current investigation where sufficient information was available at the mine to perform a finite element simulation with confidence.

Other, even more complex situations exist. A typical scenario for intermine flow between different seams in underground mines; opencast on different seams; and mining in adjacent mines is shown in Figure 52.

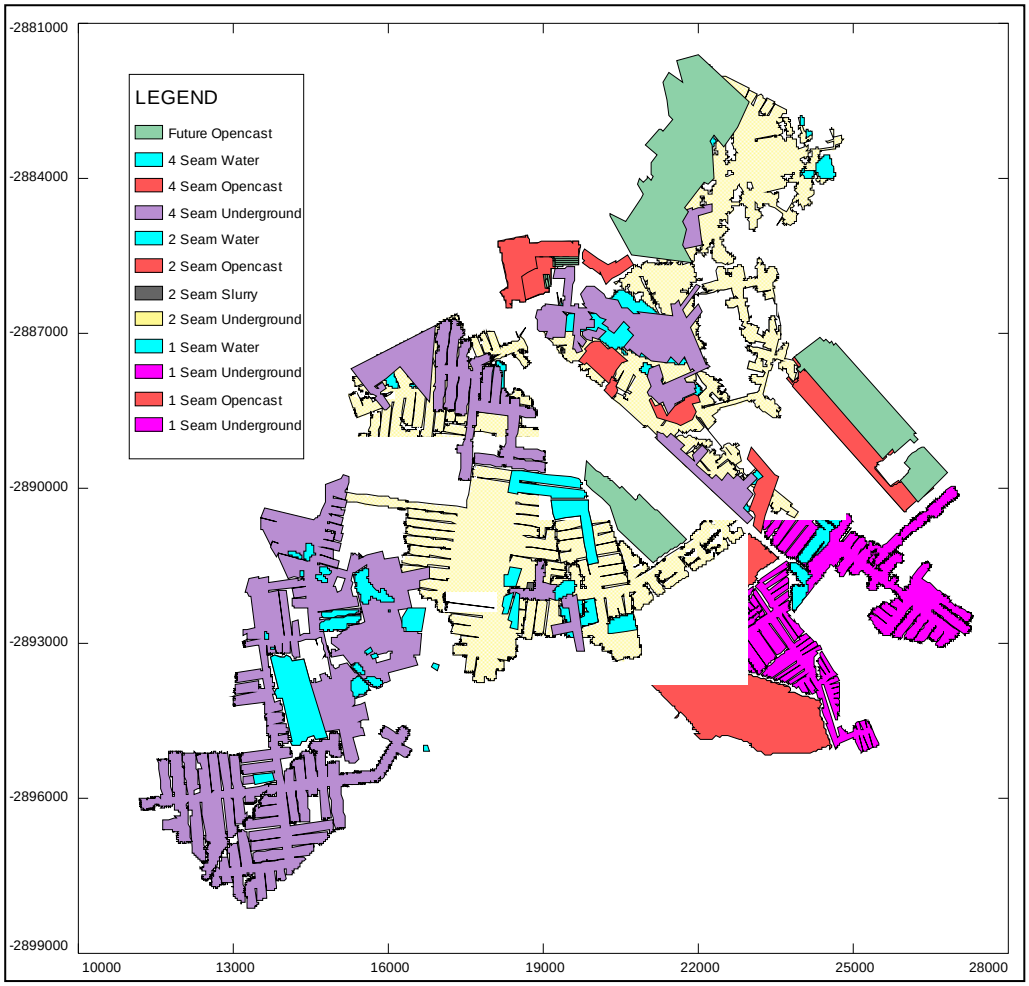


Figure 52. Complex arrangement of mining on an intermine level.

Coal floor contours for the No.'s 2 and 4 Coal Seam are presented in Figure 53 and 54. Decanting positions and elevations at which this will take place onto the surface are also shown. Depending on the interconnectivity of the mine workings, one or more decanting points will establish upon closure of all the mining. Since mining is still ongoing at both mines, structures could be put in place to influence the decanting points and elevations.

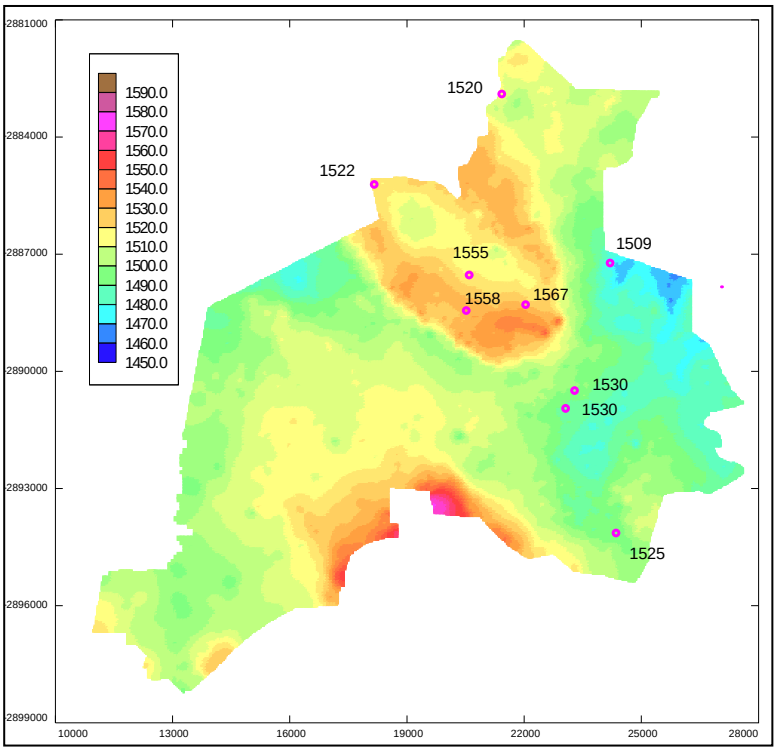


Figure 53. No. 2 Coal Seam floor contours, also showing decanting positions and elevations.

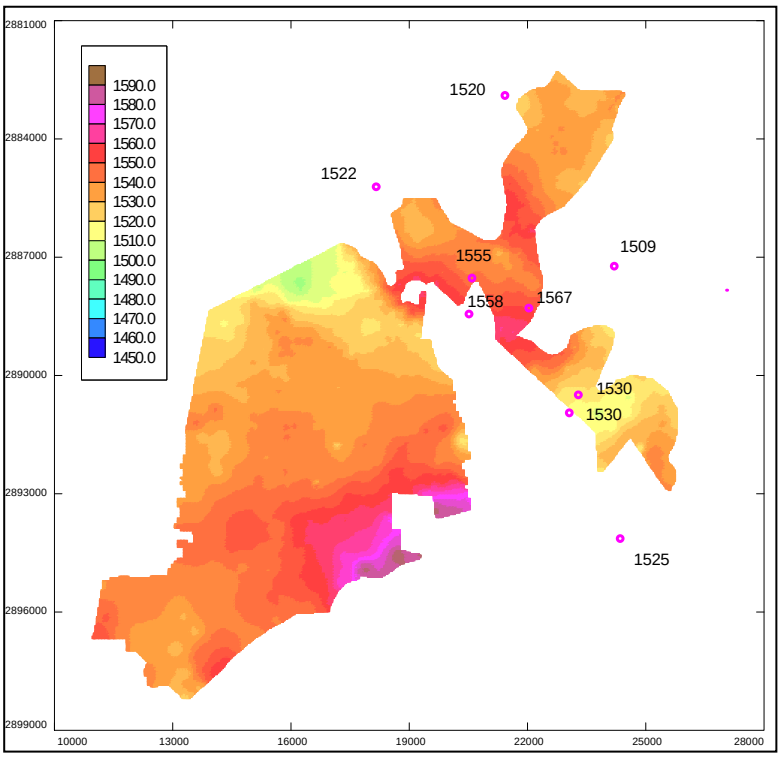
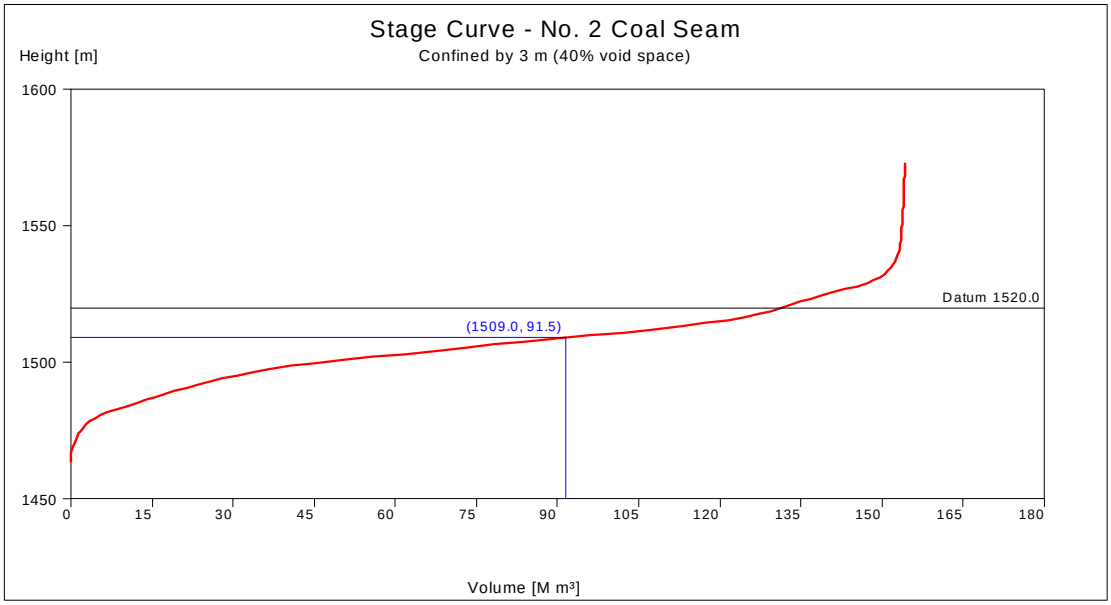


Figure 54. No. 4 Coal Seam floor contours, also showing decanting positions and elevations.

The approximate percentage of the mine workings that will be flooded can be read from the two diagrams, depending on the decanting elevation opted for. A more precise way to present this information would be as stage curves for the mined-out areas (Figure 55).



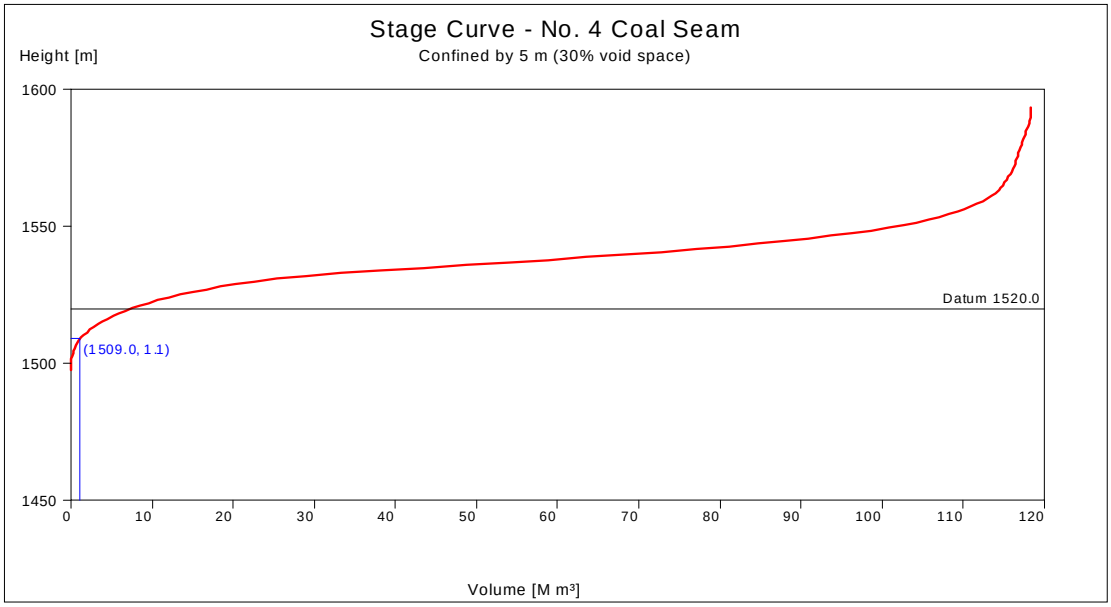


Figure 55. Stage curves for No.'s 2 and 4 Coal Seams.

The following information is extracted from the stage curves:

	Volume of water at 1509 mamsl (Mm3)	Volume of water at 1520 mamsl (Mm3)	Total capacity of seam(Mm3)	% of seam to be flooded at 1509 mamsl	% of seam to be flooded at 1520 mamsl
No. 4 Coal Seam	1.1	5	71	1.55%	7.04%
No. 2 Coal Seam	92	132	154	59.74%	85.71%

The conclusions from this information are:

- By blocking off the decanting point at 1 509 mamsl, the next decanting level will be at 1 520 mamsl.
- By doing so, the flooded capacity of the No. 4 Coal Seam will increase from 1.6% to 7.0% - which is negligible. There will be almost no advantage in raising the decanting level, in terms of limiting acid generation on this seam.
- On the No. 2 Coal Seam, the flooded capacity will increase from 60% to 86% - which is very significant. In terms of acid generation, there is a vast advantage in raising the decanting level.
- By blocking off the decanting point at 1 509 mamsl, future decanting will be diverted into another stream, some 5 km to the northwest. This is significant in terms of catchment pollution control, because it may be easier to handle decanting water in the other catchment.
- Another consideration for decanting from this mine complex should be to install release valves in the barrier to the east (not shown in these diagrams). In the eastern mine, the decant elevation will be at 1 507 mamsl. It should therefore be possible to release all water from the above example into the mine workings east of it, to allow combined treatment of the water.

The conclusion is that the significant differences between coal-floor elevations and regional gradients must be utilised optimally in the planning of future decanting positions. Mines should serve as the natural conduits for mine water to central treatment facilities. Release mechanisms to regulate flow between mines should be installed. To demonstrate how this would work in a real-life situation, Figure 56 has been compiled. The mine lease areas included by this polygon have a surface area of 900 km^2. This includes most of TNC, Goedehoop, New Clydesdale, Douglas, Middelburg South, Rietspruit, Tavistock, Tweefontein, SACE, Greenside, South Witbank and Schoongezicht Collieries. The coal-floor gradient in this area is dominantly towards the centre low-lying portion of the polygon. From here, water abstraction should be regulated and treated, rather than decanting uncontrollably. The total calculated potential water yield from these systems is in the order of 100 ML/d.

Similar regional schemes could be envisaged for other areas in the Mpumalanga Coalfields. These will, however, be smaller, because of the limited scale of mining and possibility lack of interconnection of the coal seam horizon. The Secunda and New Denmark mines would be an obvious target for combined treatment of their water. The projected decanting volume from the Secunda Collieries is in the order of 109 ML/d and that from New Denmark is 20 ML/d. Most of the Secunda mines lie in the Waterval Catchment, while New Denmark lies in the Grootdraai Dam Catchment. Underground transfer of decanting mine water from New Denmark into the Waterval Catchment should, in theory, be possible from about 60% of the mine. This is based on the principle of constructing underground compartments at New Denmark while

mining. Once mining has been completed, specific compartments should be interconnected to allow decanting into the Waterval Catchment at an elevation of 1 581 mamsl. The details are shown in Figure 57.

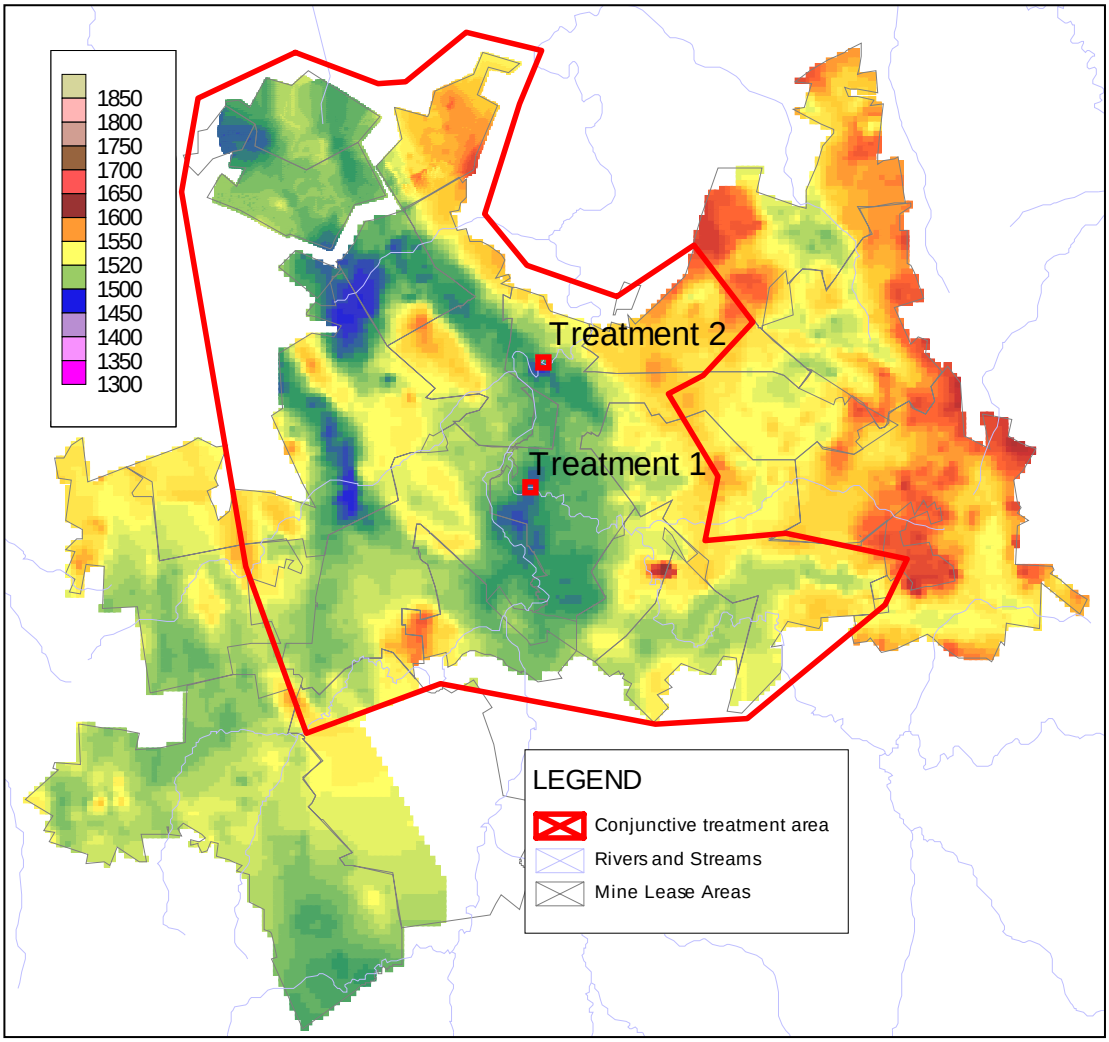


Figure 56. Area for combined water treatment management.

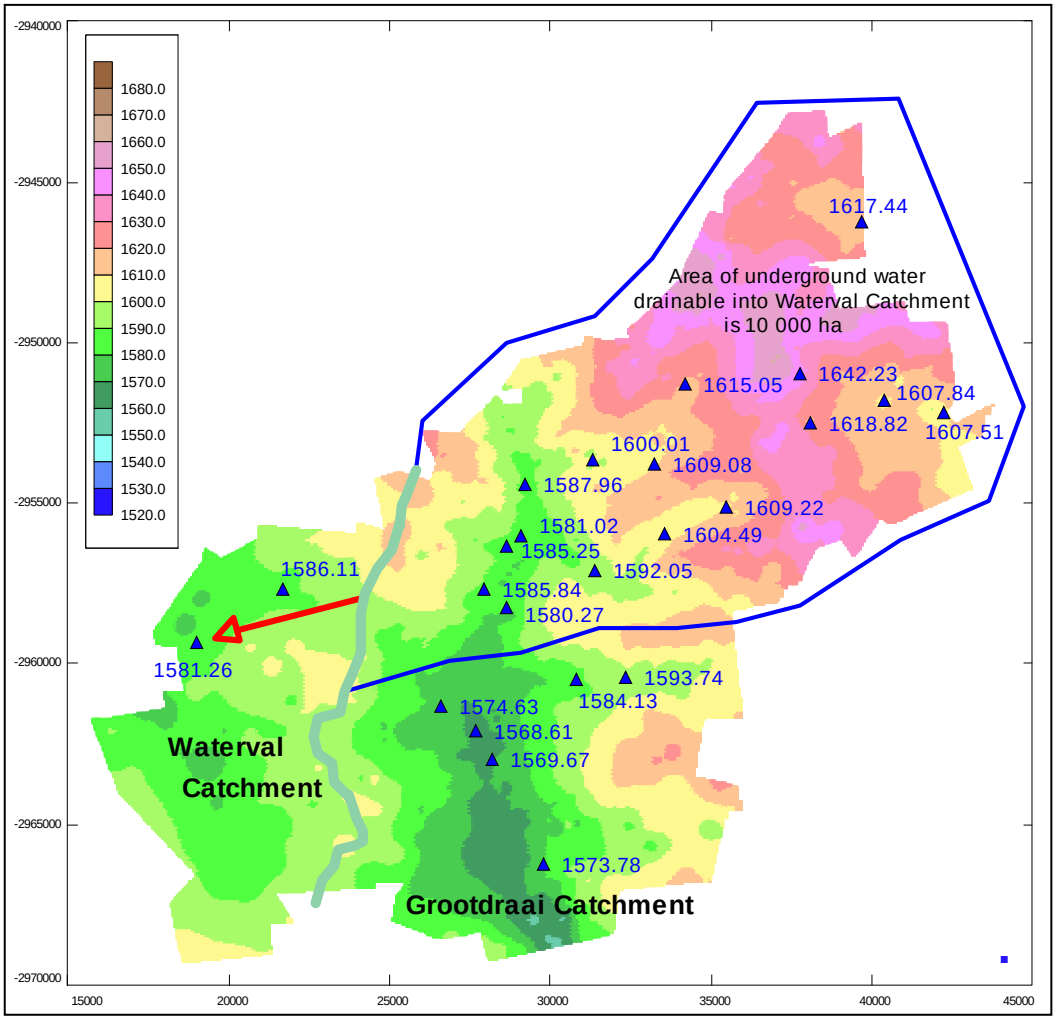


Figure 57. Example of redirecting underground decant from one surface catchment to the next to decant at 1581 mamsl. Also shown are the numerous decanting points that will result, if left to decant uncontrolled.

This will reduce the salt load to the Grootdraai Dam, which supplies water to the Secunda Complex. The mine water from both the Secunda and New Denmark mines will still end up in the Vaal Dam, which lies downstream form the Waterval Catchment. A salt load in the order of 130 t/d is expected to be released from these mines when the decanting stage is reached.

There is a finite amount of salt in the mines. Salt will not forever be released from the mines. Not all salt is equally soluble or reacts equally fast. In principle, the soluble salts should be released first. If oxidation reactions occur in the mine, then sulphate, calcium and magnesium will be released. Based on the results from this investigation, it is considered possible to flush underground collieries. This will improve the mine-water quality so that this water could afterwards be used for selected domestic purposes. While this is possible for all the mines in the Secunda – New Denmark area, two examples from other parts of the coalfields will be used to demonstrate this principle. The first example lies in the eastern part of the coalfields (Figure 58).

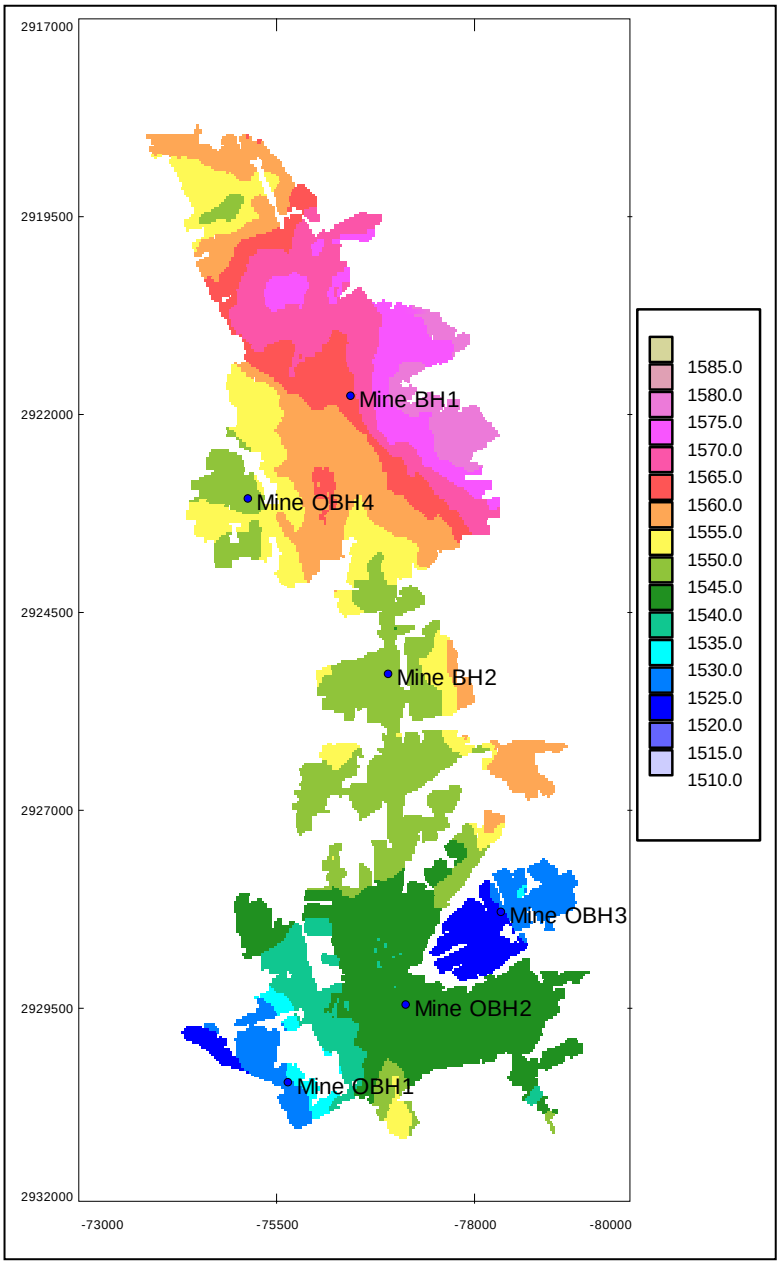


Figure 58. Coal-floor contours and monitoring borehole positions

This mine is isolated from other mining and no intermine flow on the coal seam horizon is possible. The coal floor dips from north to south. Decant from the mine will be in the south at an elevation of 1 625 mamsl. The entire coal floor lies below this level. The mine will therefore be flooded in total, before water starts to decant onto the surface. The full capacity of the mine is 38 Mm^3 (Figure 59). The mine is currently only 20% filled with water (Figure 60). It will take an anticipated 37 years for the mine to fill. In this time, significant water quality deterioration of the mine water is expected. Once flooded, oxygen will be excluded and pyrite oxidation should become minimal. The predicted decanting rates from the mine are in the range of 1 500 – 6 000 m^3/d.

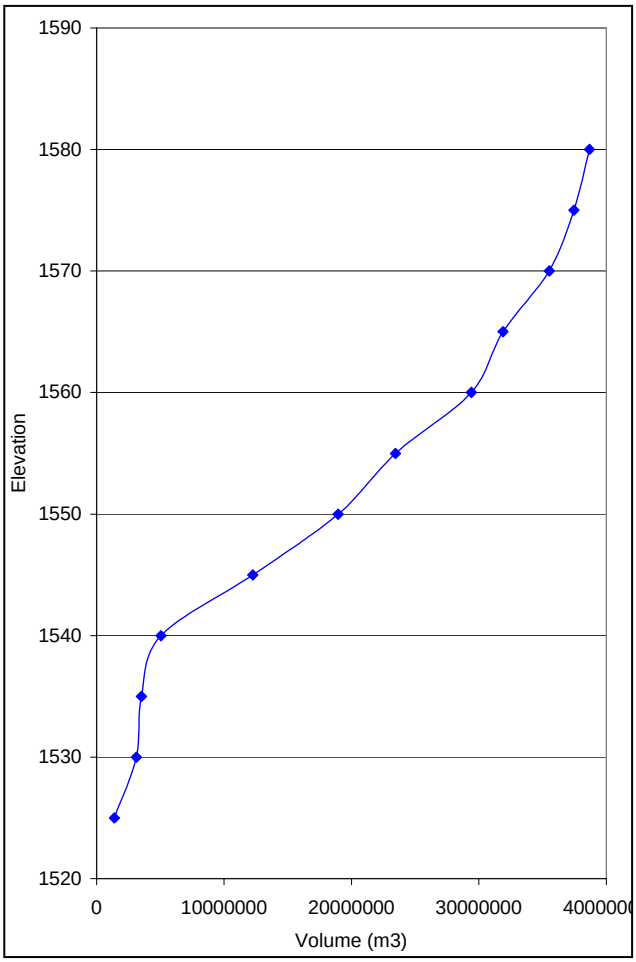


Figure 59. Stage curve for the underground water-holding capacity of the mine.

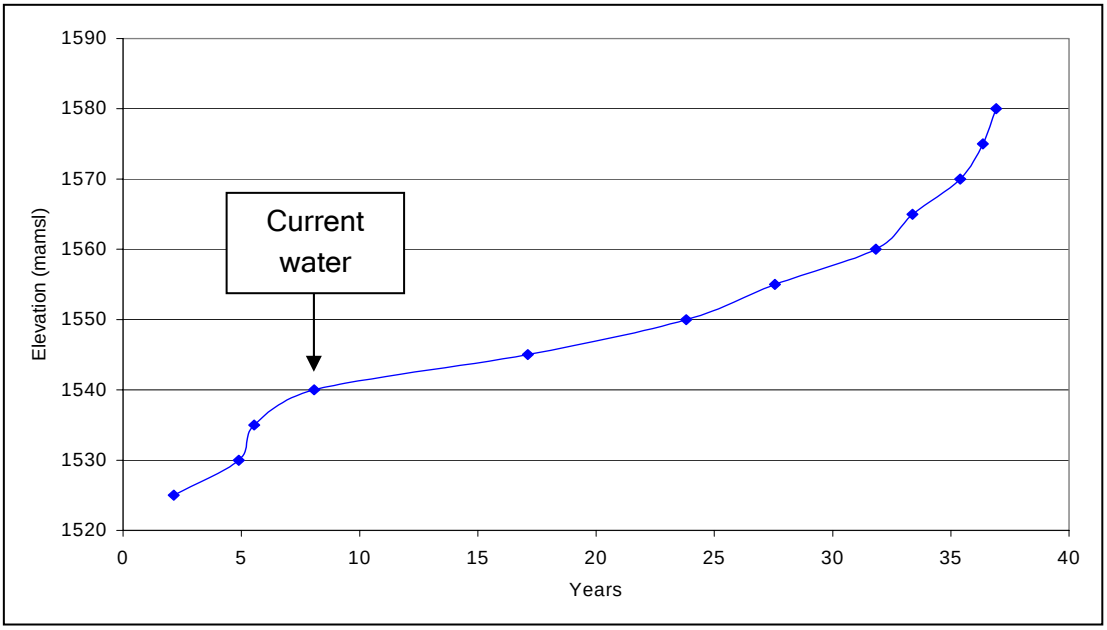


Figure 60. Projected water-levels and fill time for the mine.

Simulations indicated that the salt concentrations in the mine (Figure 61) and the tonnage of salt to be decanted (Figure 62) will significantly be reduced through the

flushing of mine water onto the surface.

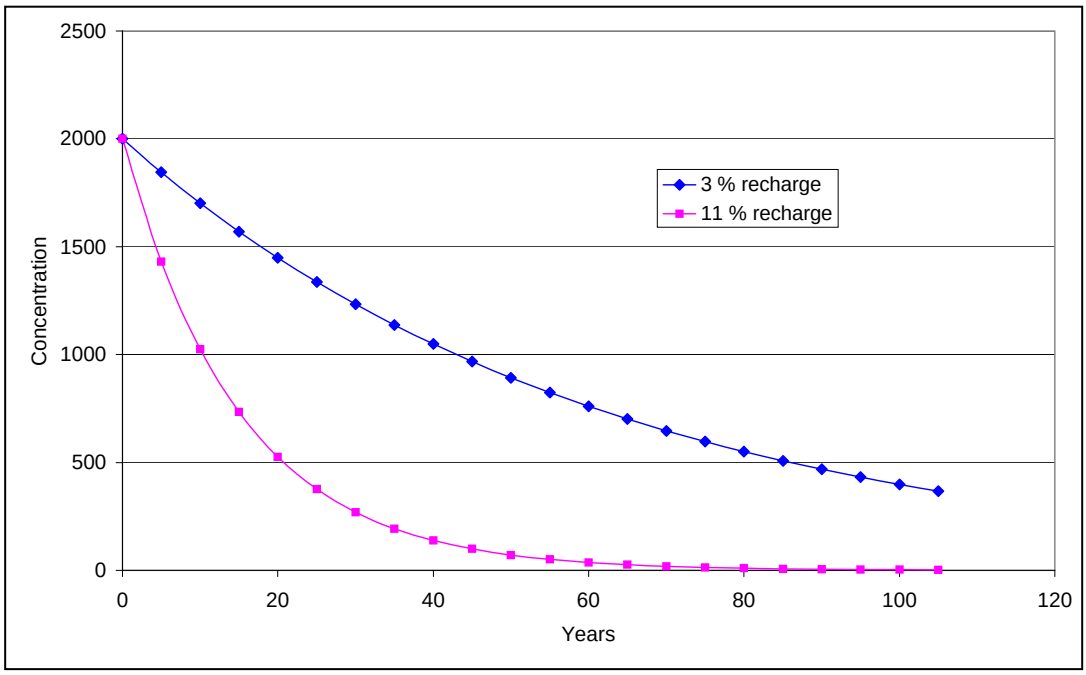


Figure 61. Flushing rates of mine water (mg/L) based on different recharge rates.

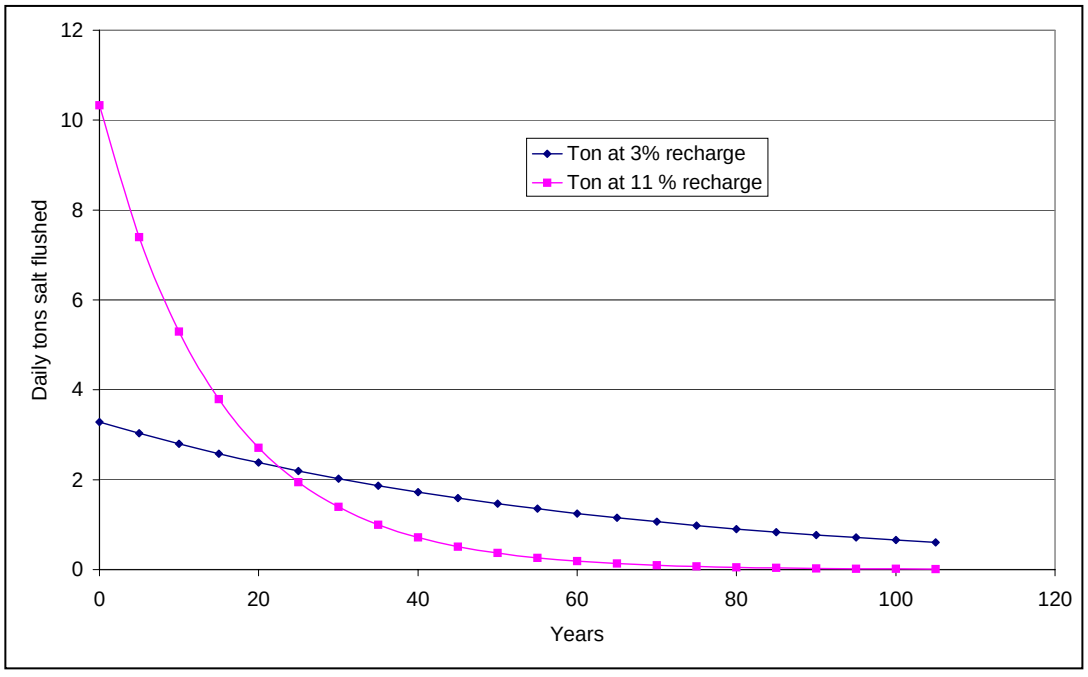


Figure 62. Flushing rates of mine water (t/d) based on different recharge rates.

The conclusion is that the flushing of a mine is a management option that should seriously be considered. It is possible through flushing, as demonstrated above, to use the underground workings of a mine for storage of water of good quality after a limited number of years. The emphasis should be on flushing as much water and possible in as short a time as possible, through the system. This could be

accomplished through artificial recharge of surface run-off. The next example emphasises this point.

This example is drawn from the area west of Witbank. The following are the main characteristics of the area

- The area mined constitutes 225 ha (Figure 63). The mining depth ranges from 12 – 77 m below surface.
- Three areas of subsidence on surface have been filled in. Currently no signs of surface subsidence exist. Little evidence of past mining activities remains on the surface. Shafts have been rehabilitated and so has the coal discard dump. There is no evidence of seepage from the rehabilitated coal discard dump and its regional environmental impact is considered negligible.
- The colliery has filled with water due to groundwater seepage and artificial recharge from boreholes in a dam (Figure 64). The holding capacity of the mine at the decanting level is 48 Mm^3 (Figure 65). Some 300 000 – 400 000 m^3/a water from groundwater seepage and run-off passes through the mine.
- Over the past nine years, water from the mine has been utilised for irrigation of some 80 ha.
- If mine water is not abstracted for irrigation, it will decant through a borehole onto the surface. At the decant level, 95% of the mine is flooded.
- The water quality in the mine is remarkably uniform. Average values are: pH of 6.8; EC of 200 mS/m and sulphate of 1 360 mg/L. It is predicted through modelling that sulphate concentrations will drop by another 50% over the next 25 years (Figure 66), on condition that flushing from groundwater seepage and recharge continues.
- In terms of mine-water management, the ideal mine-water management scheme is already in place. Excess water is utilised for irrigation and the mine water is simultaneously being flushed. It is suggested that this practice continue. Much is to be learned in terms of flushing and the use of mine water for irrigation. Such information will not only benefit the mine, but will be useful in the planning of similar ventures at other collieries that have ceased mining.

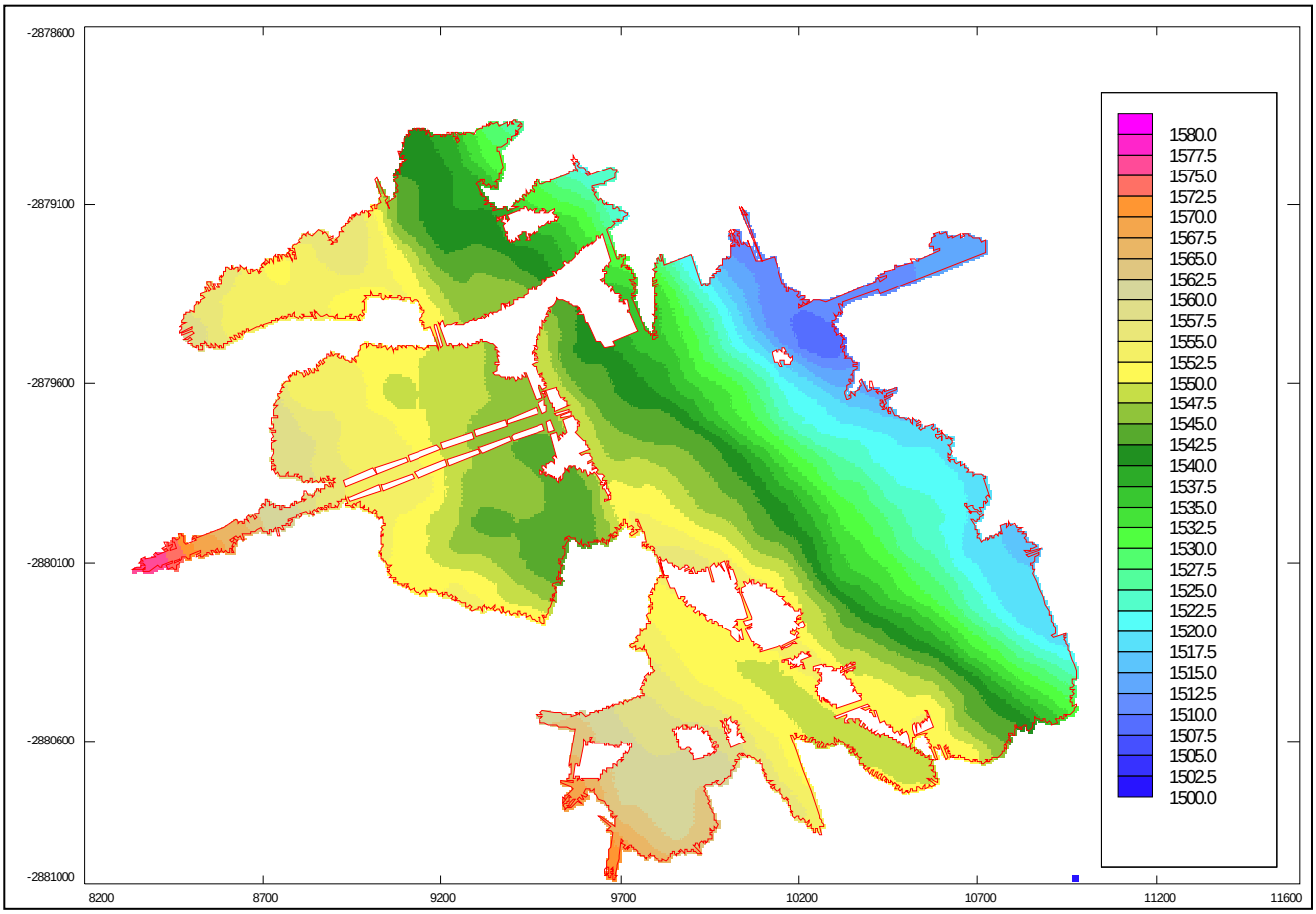


Figure 63. Mine extent and coal-floor contours.

Figure 64. Dam with surface run-off and recharge borehole from the dam into the mine

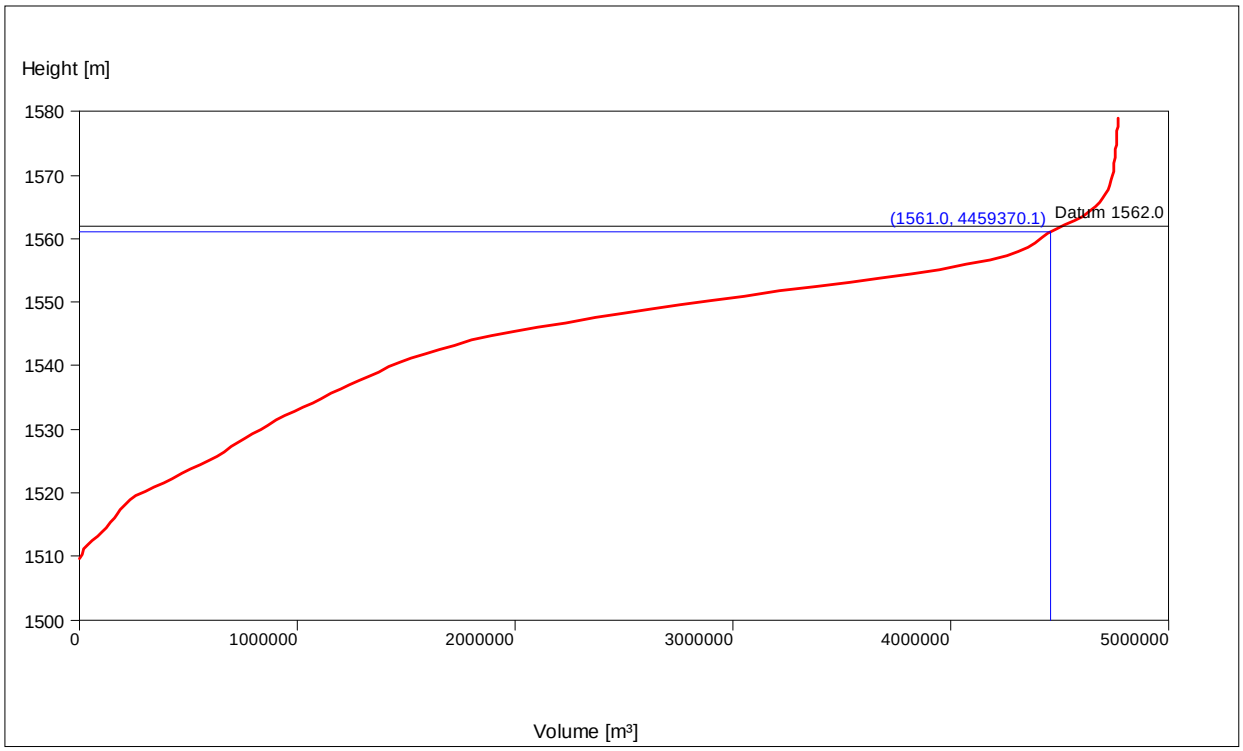


Figure 65. Stage curve showing the current water level in the mine (1 561 mamsl) and the decanting level (1 562 mamsl).

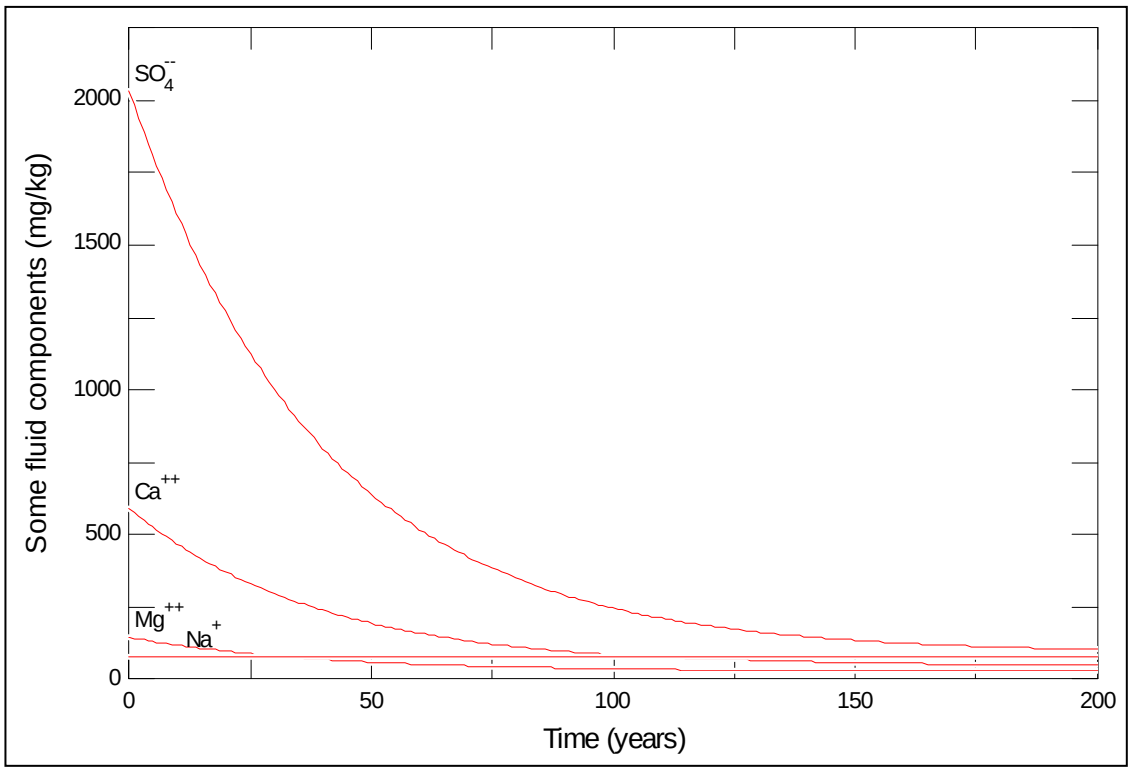


Figure 66. Simulated constituent concentration in the mine water against time, using Geochemist Workbench software.

The total volume of water expected to eventually decant from the mines is difficult to anticipate. This can, at best, be estimated from current information; even if we are

looking at a time frame some 40 years down the line. Through the application of existing methodologies as described in Section 3, the potential decanting volumes have been calculated for each of the mines that submitted information for evaluation. This constitutes an estimated 95% (by surface area) of all the collieries present in the Mpumalanga Coalfield. The predicted decanting volumes are graphed in Figure 67.

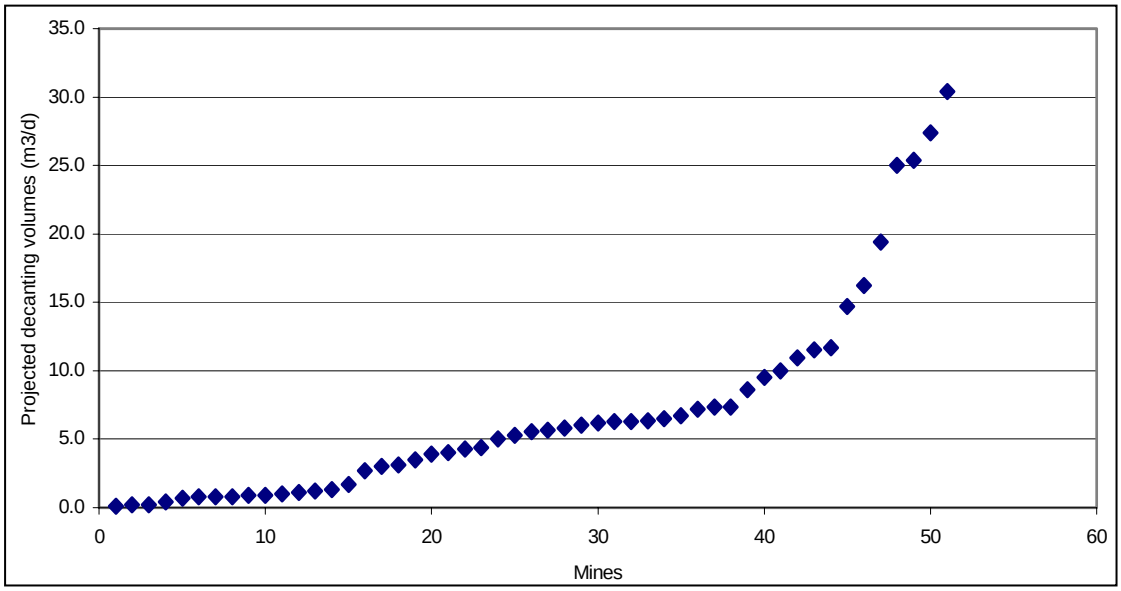


Figure 67. Projected flux from existing collieries during future decanting.

The volumes are related to the size of the mine but also to the mining method applied. The mines with the anticipated high volumes are the modern high-extraction mega-mines. In this respect, it should be stressed that the actual decanting quantity in the above graph is meaningless without also considering the circumstances at the mine itself. These values are merely issued as a range of values for mines to examine their relative position to the whole ensemble of collieries.

In total, about 360 ML/d will decant from all the mines together. On a catchment basis, the breakdown is:

Wilge/Klip	Olifants	Klein Olifants	Vaal	Komati
(ML/d)	(ML/d)	(ML/d)	(ML/d)	(ML/d)
23	170	45	120	2

For illustrating the significance of these volumes, the total volume to decant from these mines is in the same order as the annual natural run-off into the Witbank Dam.

The aforementioned information demonstrates that intermine flow can and must be managed. It is essential that mines start planning their water management in conjunction with adjacent collieries. Two levels of planning are suggested:

- The local level where neighbouring mines plan together.

- The regional level where corporate head offices and the government get together. At this level policy matters and regional planning matters should receive attention. The prime question that needs to be addressed should be how to minimise and manage the problem on a catchment-specific basis.

Up to now, emphasis in this report has mainly been on the flow of water. This is clearly not the only issue relating to intermine flow. Water quality is another aspect, which is equally important. It is senseless to control volumes while contaminating clean mine water with dirty water. To address this matter, a detailed understanding of the potential of each mining operation to generate salt and/or acid water is necessary.

8 SALT BALANCE

The water quality in the Mpumalanga Coalfields has been discussed at length in many reports to the mines and the WRC. The most complete documents on this topic are those by Hodgson *et al.* (1998), and Van der Berg *et al.* (2000) and Hodgson *et al.* (2000).

None of these reports, however, contains the latest information and views on this matter. For that reason, some of the information gathered during this investigation will be included in this report to supplement that already published. In terms of water quality in the mines, the issues at stake are:

- The geochemistry of the sediments and coal.
- The mining type.
- The degree of flooding.
- The through-flow rate of fresh water.

The geochemistry of the Ecca sediments has been researched in detail at a significant number of collieries. These are: Douglas, Middelburg, Tavistock, Kriel, Optimum, TNC, Minnaar and Ermelo Collieries. The tests performed are:

- Static acid-base accounting and leaching tests for more than 400 samples of the sediments and coal.
- Numerous test pits have been dug into the spoil in opencast areas at Tavistock, Kriel and Optimum, to examine the characteristics of the spoil that has been lying under field conditions.
- Numerous boreholes into spoil and underground mines have been logged using a multi-parameter chemical logger.
- Numerous water samples have been analysed for macro and micro

constituents.

A selection of these results are presented in the diagrams that follow. The theory behind certain principles is presented in Appendix B.

Static acid-base accounting involves two tests:

- Determination of the acid potential by oxidation of the pyrite in a rock or coal sample, through addition of H_2O_2 and the analysis of the fluid for sulphate.
- Determination of the base potential through acid addition and back titration to a pH of 7.0.

Four bits of information are thus obtained, namely:

- The initial pH of the rock or coal before oxidation.
- The final pH of the rock or coal after oxidation.
- The total acid-generating potential.
- The total base neutralising potential.

From the last two, the net neutralising potential (NNP) of the sample is calculated by subtracting the acid potential from the base potential. A plot of pH-values against their NNP's is shown in Figure 68.

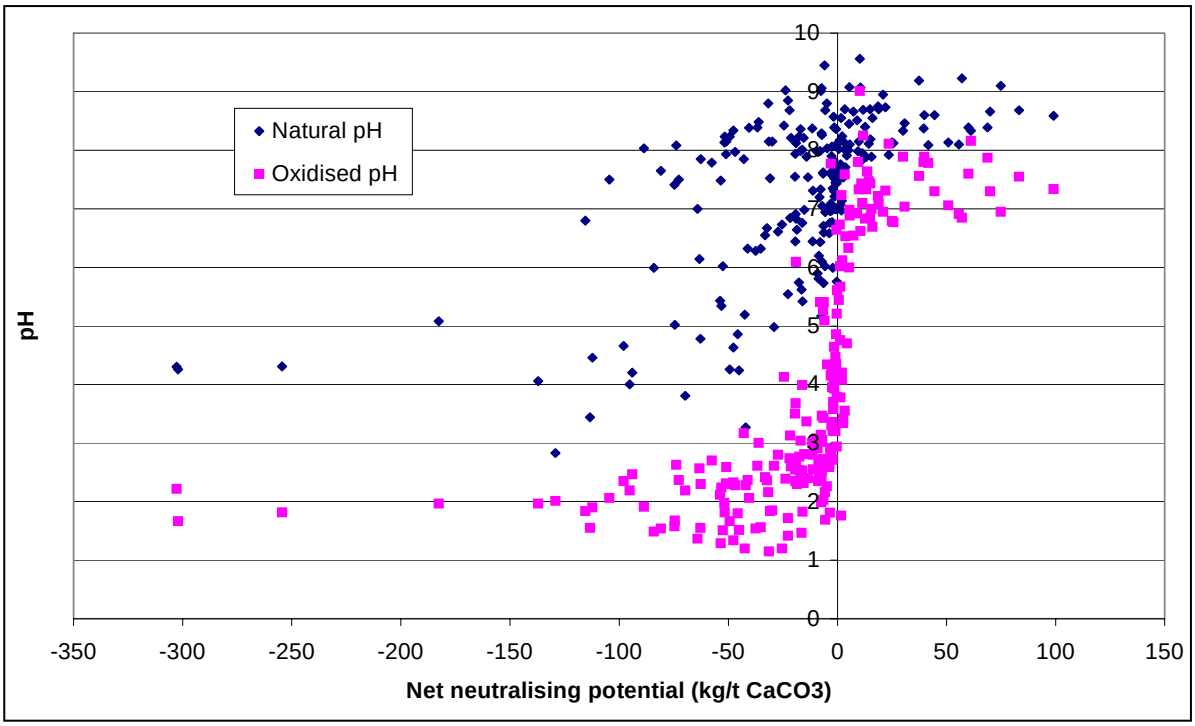


Figure 68. Natural and oxidised pH-levels of coal and rock samples versus their NNP's.

This picture is typical for all the collieries in the coalfields. The NNP is wide-ranging and does not necessarily follow a pattern in terms of the rock types. Some correlation, often exists between the degree of weathering of the rock and these values. The logic behind this statement lies in the following:

- Pyrite needs oxygen to oxidise. Pyrite will therefore only oxidise when it lies above the groundwater table. These sediments are usually moderately to highly weathered and sometimes devoid of pyrite through oxidation over many years.
- The base potential, in the form of calcium/magnesium carbonate, does not require oxygen to dissolve. It dissolves in circulating groundwater because the water is usually undersaturated in terms of its dolomitic component.
- Naturally circulating groundwater only occurs in the top 10 – 40 m of the Karoo strata in Mpumalanga. Deeper down, the sedimentary rocks are usually tight and very little groundwater circulation, if any, takes place under natural conditions.
- Following the above arguments, a zone of leaching should therefore establish, which from top to bottom have specific acid-base characteristics:

Description	Depth (m)	State	Hydraulic character	Composition
Top zone	3 - 10	Weathered strata	Vertical seepage of rainwater	Pyrite and base potential almost totally leached
Middle zone	20 - 40	Fresh strata	Lateral seepage of groundwater	Pyrite in tact, base potential partially leached
Bottom zone	Below above	Fresh strata	Almost no flow of groundwater	Pyrite and base potential in tact

This principle could also be demonstrated from actual chemical analyses (Figure 69).

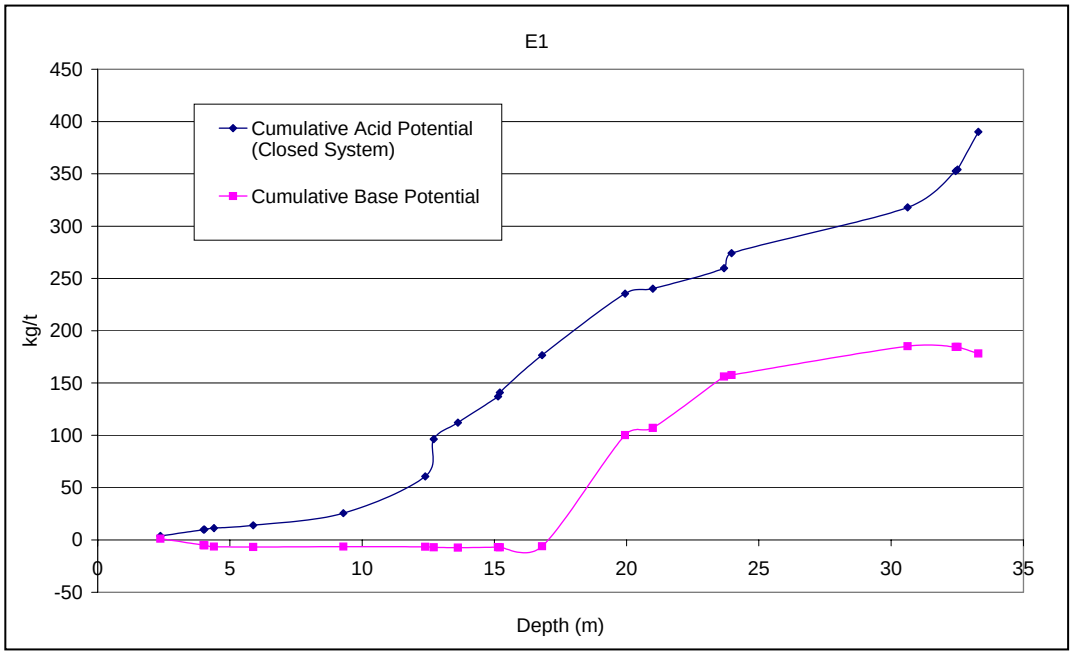


Figure 69. Example of cumulative acid and base potentials down a borehole.

Interpretation of this information is as follows:

- The acid potential rises slowly with depth, down to the water table at 10 m. Deeper down the borehole, significantly higher sulphur concentrations are present, hence the steeper gradient for the cumulative values.
- The base potential is totally depleted to a depth of 18 m. Deeper down, similar

base concentrations compared to the acid concentrations exist, thus a similar slope for the base potential graph.

- The net excess acid potential can therefore be attributed to the zone from 0 – 18 m. Mining in this zone will lead to almost immediate acidification of the rock when exposed to air. Such systems are sometimes present in the area south and west of Witbank.
- This idealistic example of acid and base potential distribution is clearly not the rule, but rather the exception. It should be emphasised that rock does not lie under ideal conditions in the field. Many secondary features, such a fractures, intersect the rock. Deviations from this idealistic presentation are the rule rather than the exception.
- In all instances of acid-base accounting, careful examination of the core, before it is pulverised and analysed, would pay dividends when it comes to interpretation of the results.

Apart from the NNP (Figure 68), the pH-values also reveal a lot in terms of the geochemistry of the samples. The following are concluded from the pH-values in this diagram:

- The high pH-values, i.e. >8.5, are attributed to sodium carbonate in the sediments. This sodium carbonate occurs naturally in the connate water in rock and is not a mineral. If the rock is dried completely, in a laboratory for instance, the sodium carbonate would obviously precipitate, but is still available to impact on the pH of the water, when submerged. Sodium concentrations in the rock vary from area to area in the Mpumalanga Coalfields. In a general sense, the sodium content increases from north to south, i.e. in the direction in which the coal seam deepens. Southwards from Kriel Colliery in particular, the sodium content in the mine water increases drastically. At Secunda and New Denmark Collieries, sodium concentrations exceeding 600 mg/L are often found. High pH-values are always associated with the high sodium values in the natural groundwater at mining levels. The sodium has an important initial buffering function. It is sufficient to buffer the mine water against acidification for many years (even decades), before being depleted. Unfortunately, sodium, being highly soluble, is released into the mine water at its maximum available rate. When mine water is removed from the mine, this base potential is also removed and discarded. This is confirmed in Figure 70, which shows the water quality in a stream below one of the mines. Careful planning of the mine lay-out, to keep as much of the water underground as possible, would reduce pyrite oxidation and keep the base potential and pH-levels up.

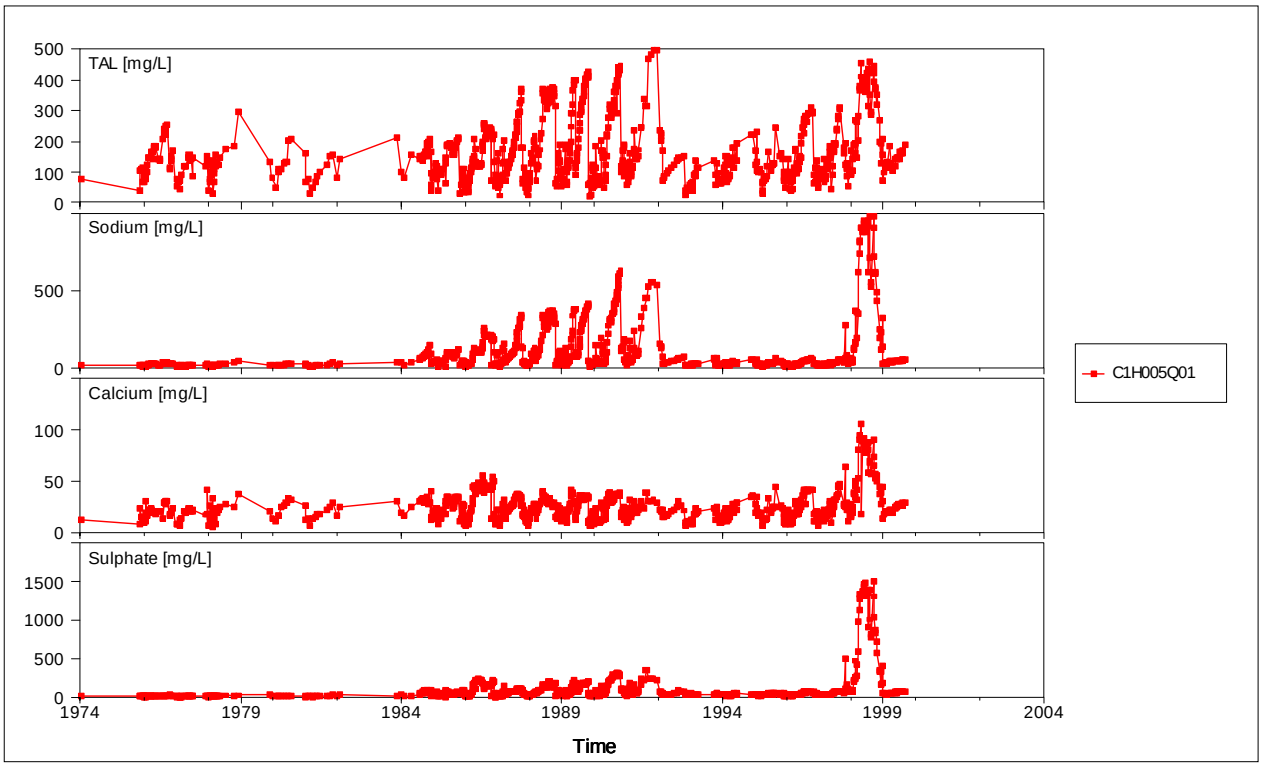


Figure 70. Water qualities in a stream receiving periodic discharge of mine water.

- The second feature on the pH-axis in Figure 68 is the separation of the natural and oxidised pH-values on the zero NNP line. This line represents the line where all base potential, i.e. the sodium, calcium and magnesium carbonates, are depleted. Other carbonates, such as iron carbonate, are not included since the iron generates an equivalent amount of acidity. This separation confirms that the experiment was set up properly. In addition, the drop in the pH is sudden and all the way down to below 3.0, which is the conversion point of Fe^{2+} to Fe^{3+}. The conclusion is that none of the other minerals in the system, such as kaolinite or illite (the clay minerals) have a significant short-term impact on the oxidation chemistry. The clay minerals would certainly play a role in the very long-term chemistry of mine water, i.e. over hundreds of years. They all have the capacity to readjust the pH of the water to normal levels. This can be demonstrated though computer simulation using Geochemist Workbench software, the results of which are presented in Figure 71. In the short term, i.e. over the life of a mine, the impact of the clay minerals can be ignored.

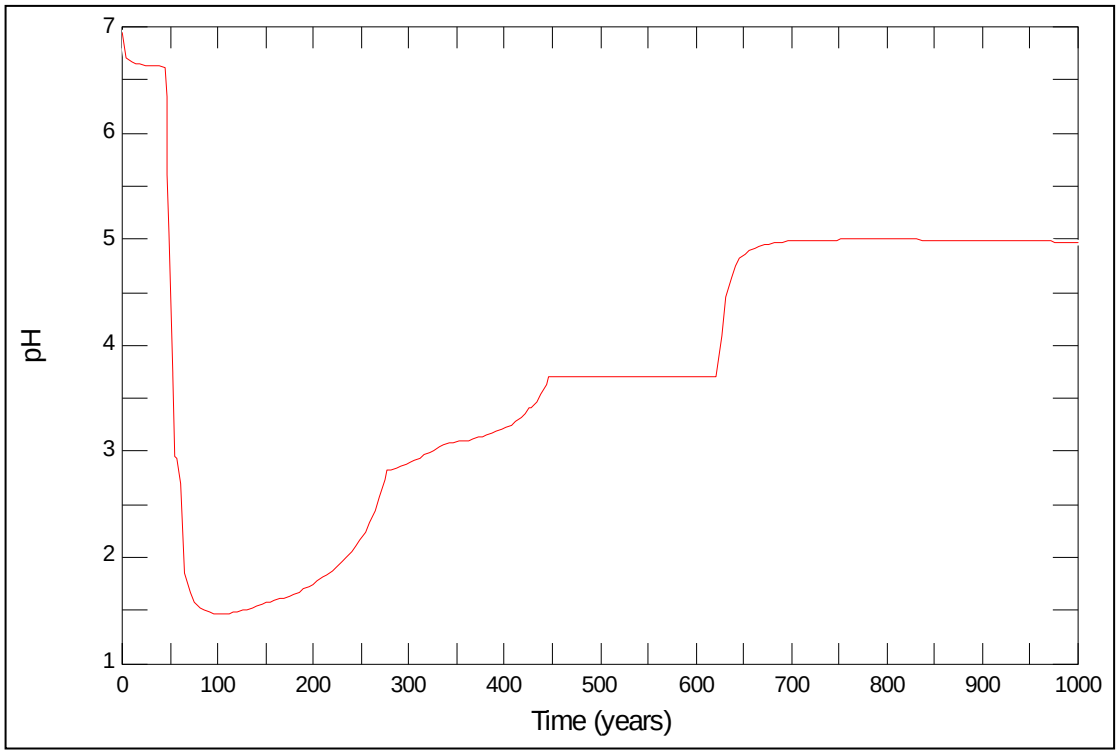


Figure 71. Typical reaction path pH-conditions for rocks and coal mined in the Mpumalanga Collieries.

- The reaction path in Figure 71 is typical of what would be found in all Mpumalanga Collieries. The time scale is not accurate and should therefore not be quoted for the average of the collieries. Also, sections of the graph may be absent in certain instances, or stretch over shorter or longer periods. These are all related to the actual mineral composition of a specific rock in the larger context of the coalfields. Each of the dips or rises on the graph has specific meaning in the mineralogical sense.
- The drop of the pH to around 2.0 has been observed in many of the collieries. At this low pH, sulphate levels could exceed 10 000 mg/L and heavy metals are present in high concentrations. The availability of the heavy metals over the whole pH-range may be viewed in Figure 72 - 75. Three categories of elements are present. Firstly, there are those such as iron, with a variation in their solubility characteristics with a drop in the pH. This is due to the conversion of iron from Fe^{2+} to Fe^{3+}. Secondly there are those that increase linearly with a drop in the pH. Finally, there are those that are not affected by a drop in the pH. These are usually the ones available in limited quantities in the rock or coal.

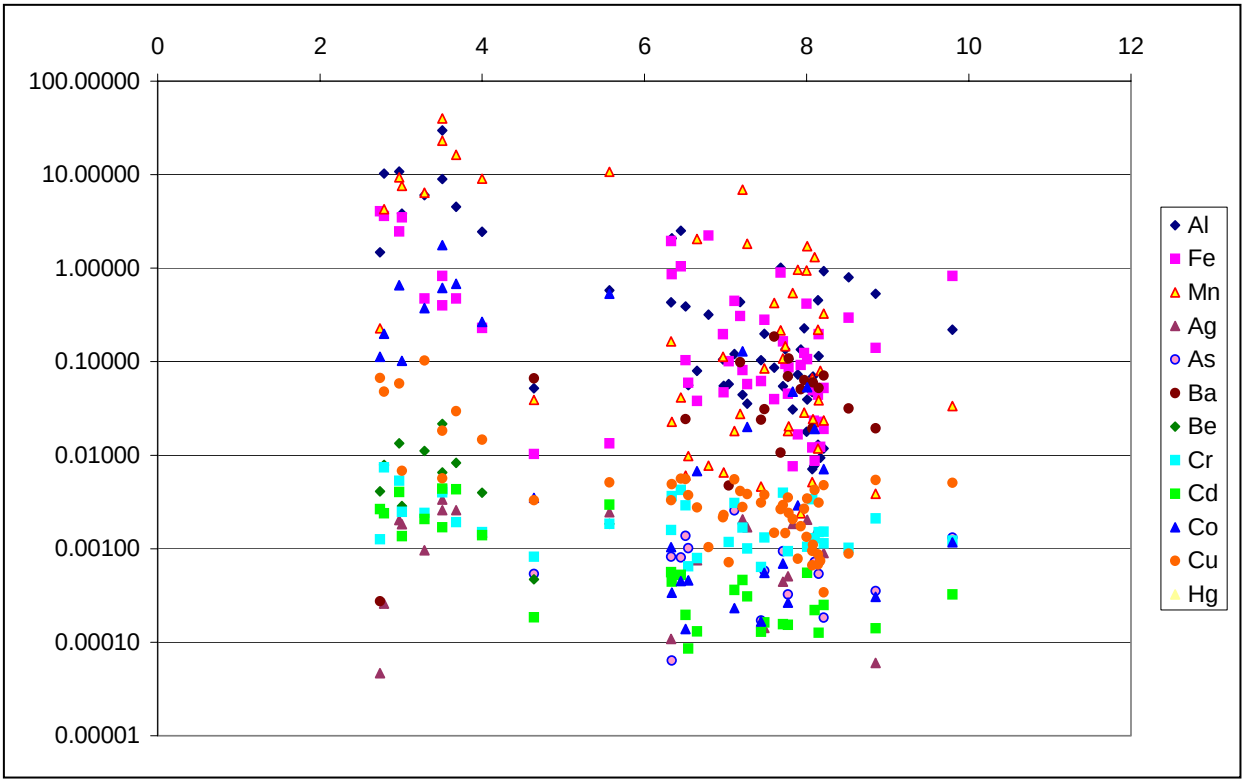


Figure 72. Availability of elements from rock and coal at various pH-levels.

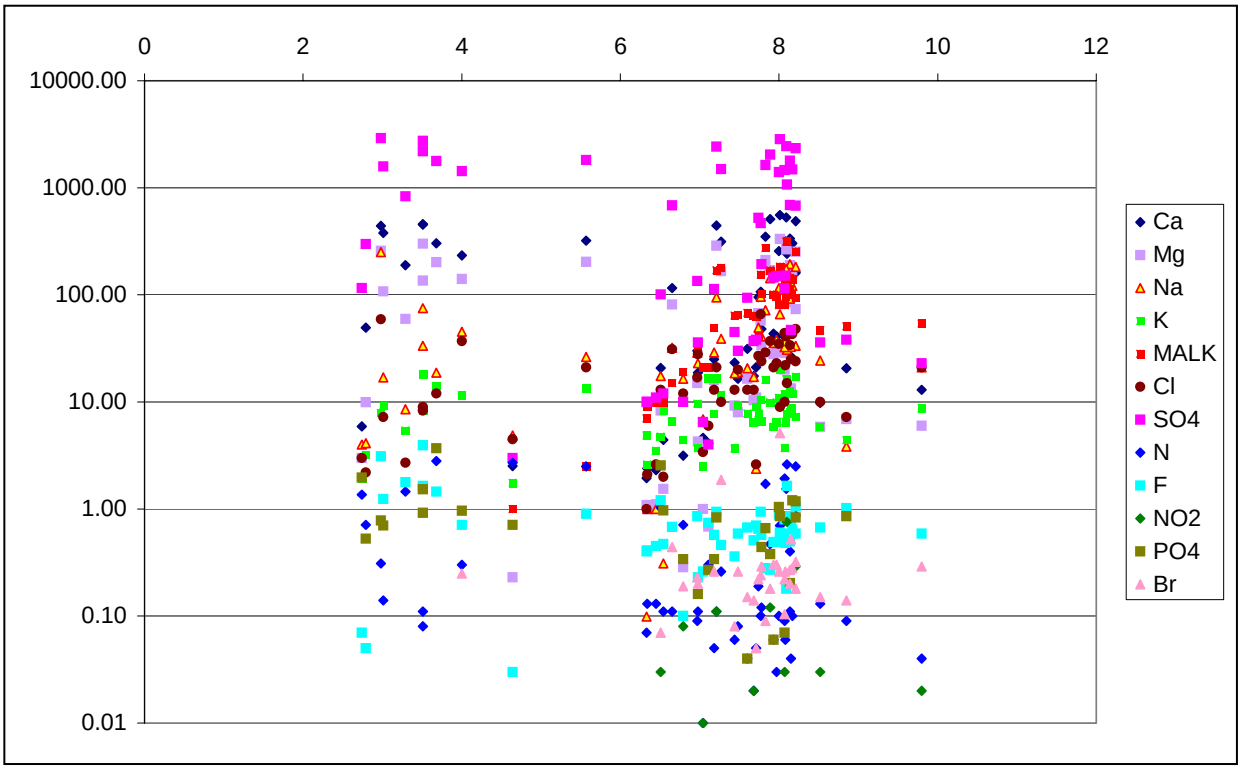


Figure 73. Availability of elements from rock and coal at various pH-levels (continued)

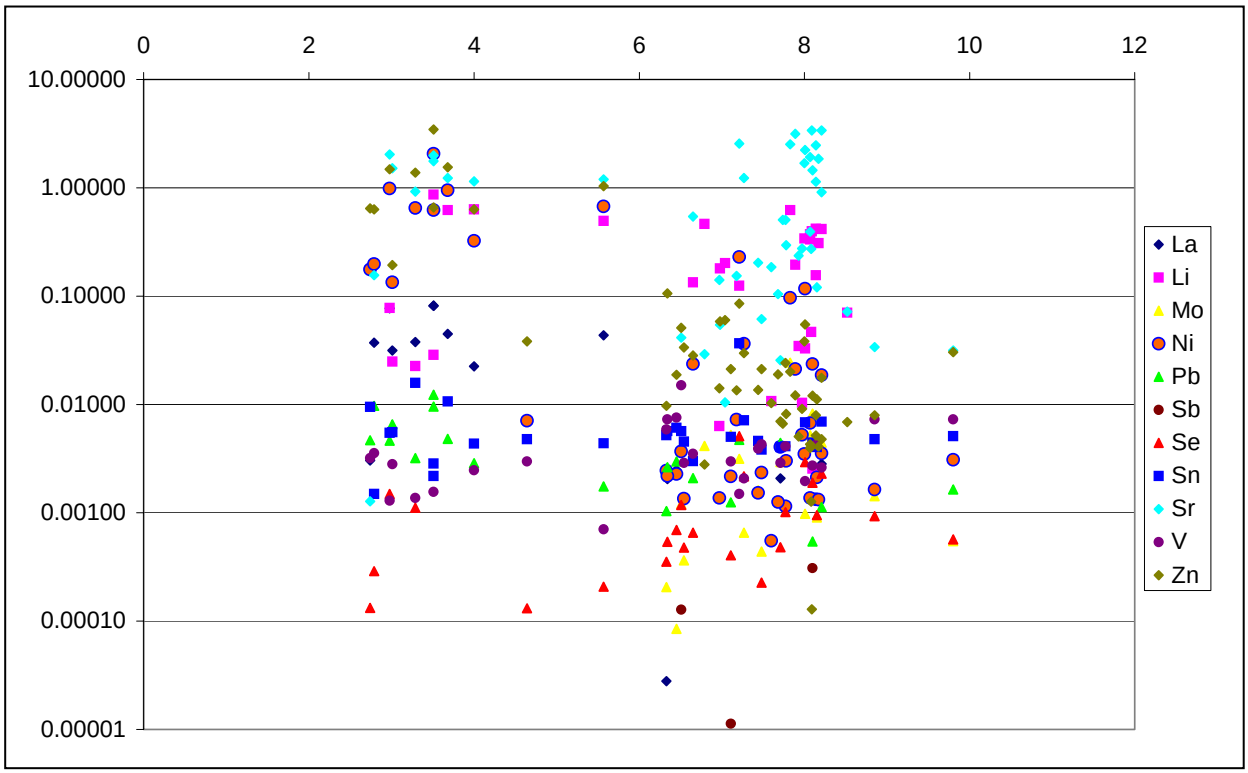
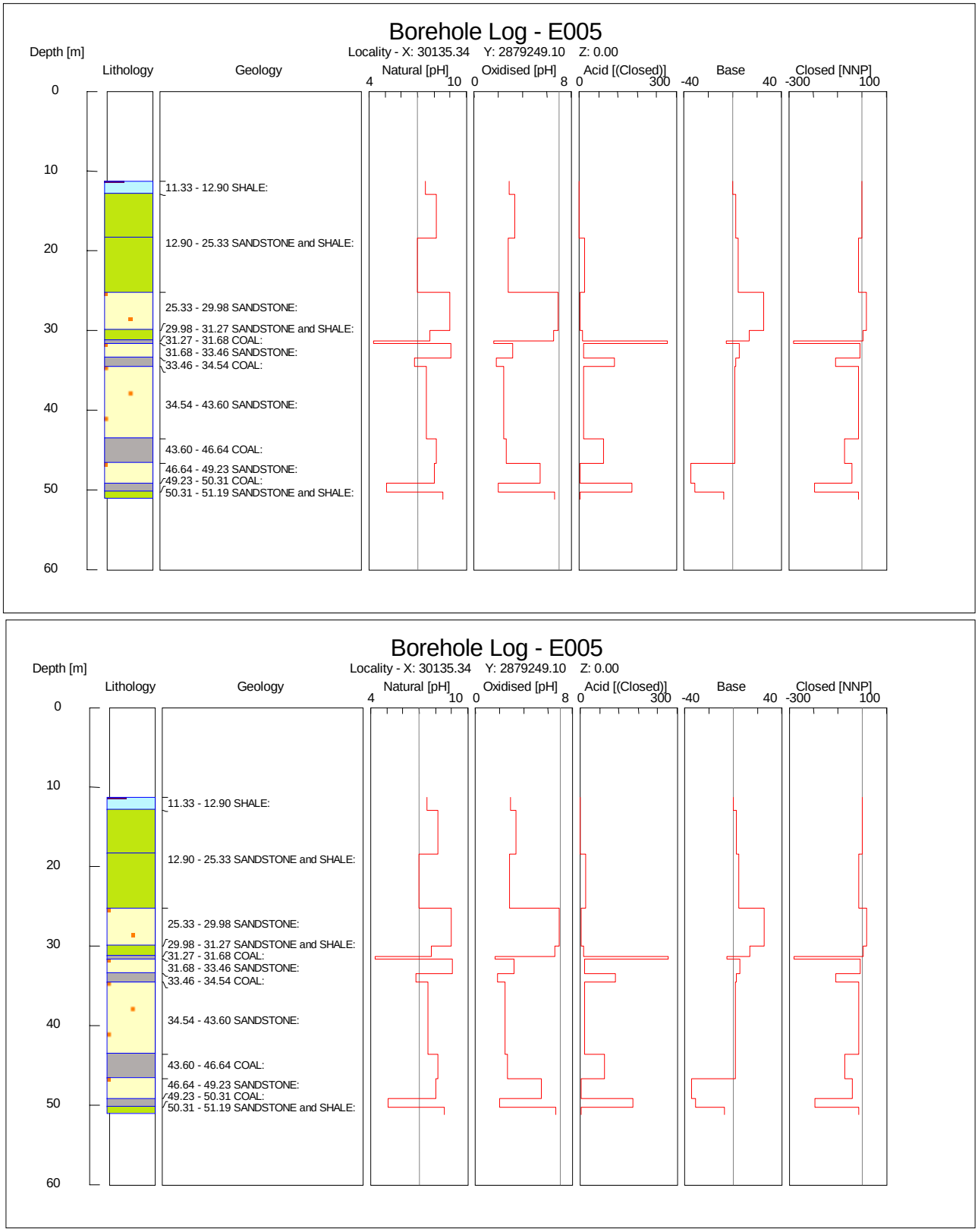


Figure 74. Availability of elements from rock and coal at various pH-levels (continued).

Vertical plots of the mineral availability, including the geology, sometimes yield more information of the distribution of these constituents in the different layers (Figure 75).

Many detailed interpretations might be made from this information, but the one that should be mentioned here is the extent of vertical variations in the parameter values. The conclusion is that is not possible to single out a specific layer in the coalfields and assign unique values to it. At best, the totality of the system can be addressed in a statistical or probabilistic way.



Borehole Log - E005
Locality - X: 30135.34 Y: 2879249.10 Z: 0.00
Depth [m]
Lithology
Geology
Natural [pH]
Oxidised [pH]
Acid [(Closed)]
Base
Closed [NNP]
4
10
0
8
0
300
-40
40
-300
100
0
10
20
30
40
50
60
11.33 - 12.90 SHALE:
12.90 - 25.33 SANDSTONE and SHALE:
25.33 - 29.98 SANDSTONE:
29.98 - 31.27 SANDSTONE and SHALE:
31.27 - 31.68 COAL:
31.68 - 33.46 SANDSTONE:
33.46 - 34.54 COAL:
34.54 - 43.60 SANDSTONE:
43.60 - 46.64 COAL:
46.64 - 49.23 SANDSTONE:
49.23 - 50.31 COAL:
50.31 - 51.19 SANDSTONE and SHALE:

Figure 75. Vertical profiles showing physical and chemical characteristics of the various layers.

The proof of the heterogeneity of the system under field conditions lies in chemistries, as measured in boreholes into opencast spoil and underground mines. Three

examples are included to demonstrate this complex relationship (

Figure 76 - 79).

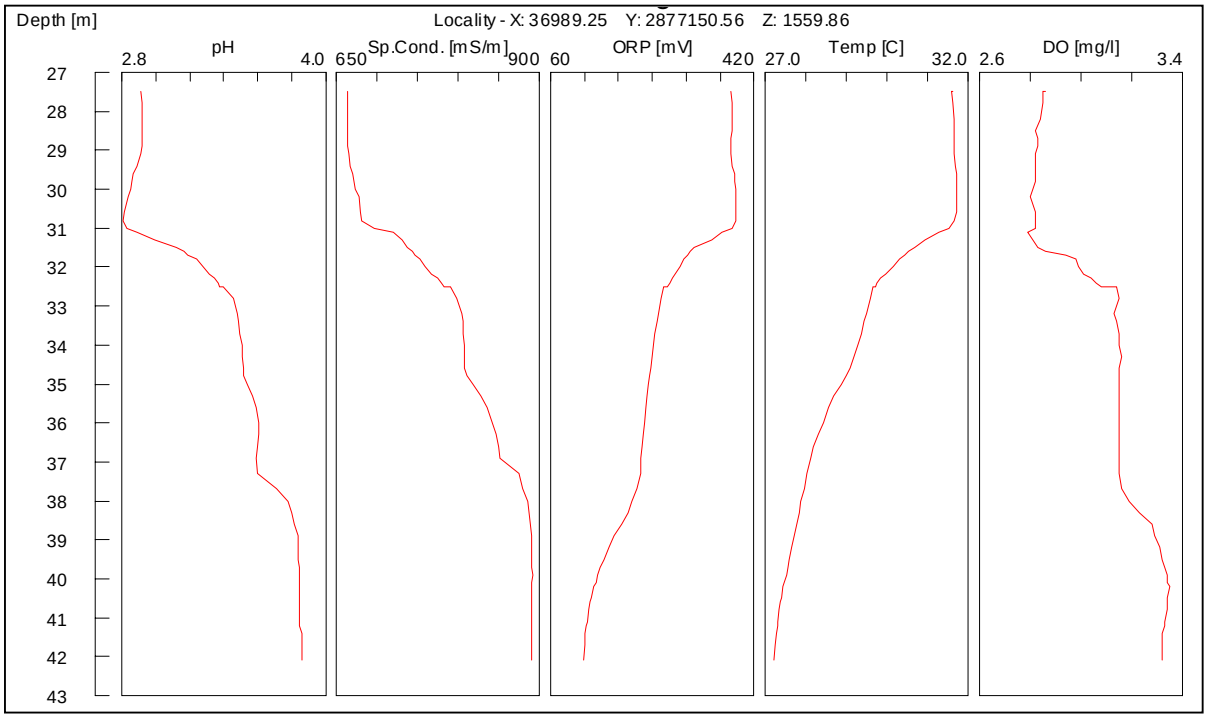


Figure 76. Multiple chemical logs of acid water in an opencast mine.

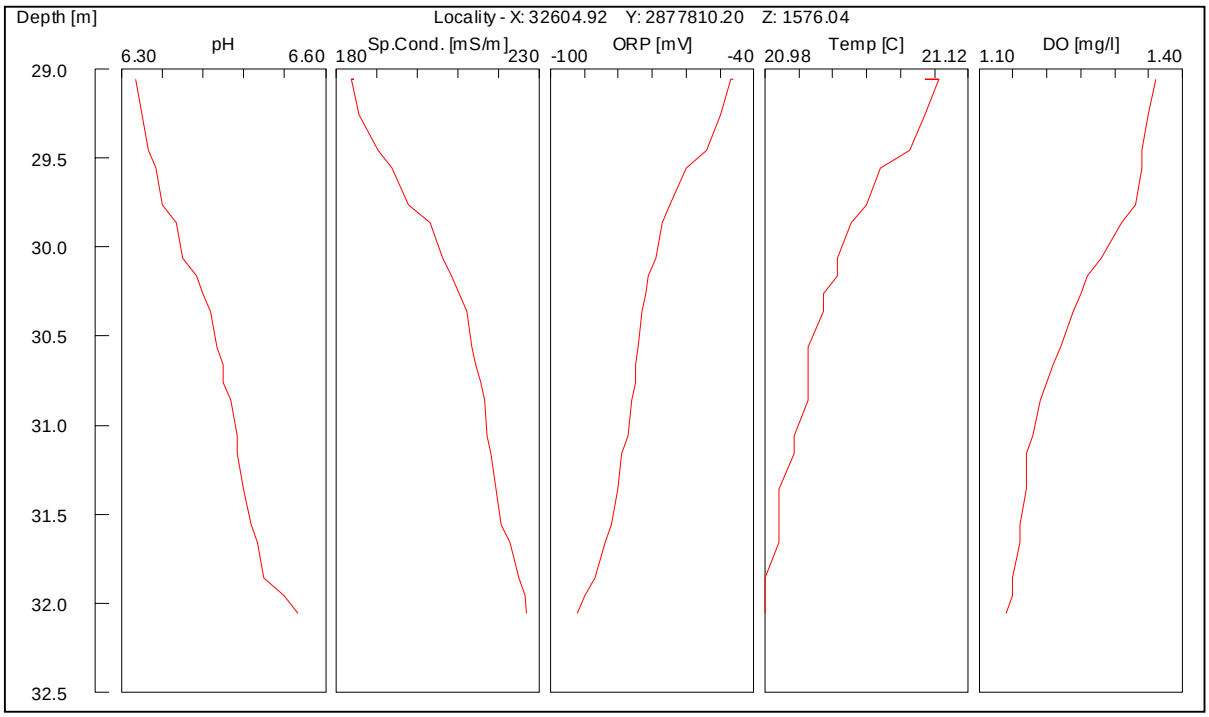


Figure 77. Multiple chemical logs of neutral water in an opencast mine.

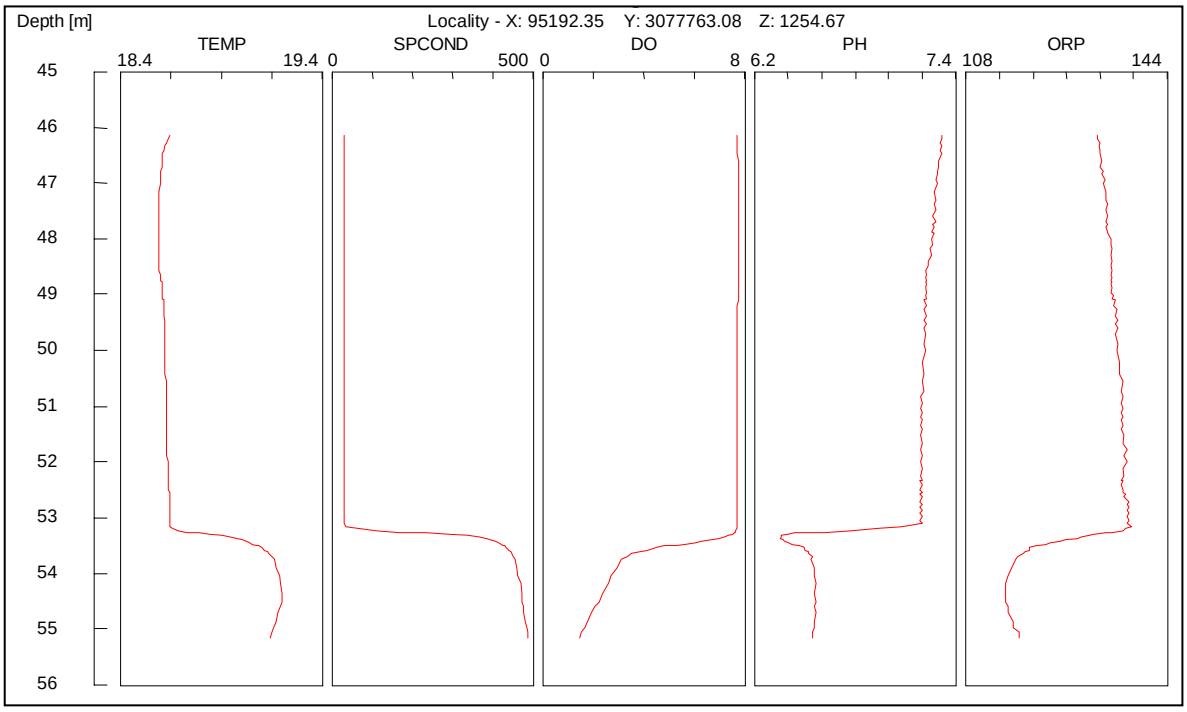


Figure 78. Multiple chemical logs of water above and in an underground mine.

The first example (

Figure 76) is from water in opencast spoil in the northern portion of the Mpumalanga Coalfields:

- The spoil water is acid with pH-levels between 2.8 – 4.0.
- The specific conductance, redox, dissolved oxygen and temperature values reflect corresponding variations with variations in pH-levels.
- All the parameters are in the ideal range for bacterial oxidation of the pyrite. Hence oxidation rates up to 10^6 times higher than those under neutral conditions (Evangelou, 1995) would be expected in the unsaturated spoil from 0 – 27.5 m. Below the latter depth the spoil is flooded.
- The rise in the pH with depth suggests that the oxidation rates in the unsaturated spoil exceed those in the flooded spoil, though oxidation of pyrite is also possible in the flooded spoil at these low pH-values.
- On the surface at this locality, there is no evidence of the acid reactions and spoil water presence deeper down. There is also no way in which this could be predicted in terms of time and chemistries. From static acid-base accounting for this mine, it was inferred that the possibility of acidification exists, though this cannot be predicted on a site-specific basis.
- The complex relationship between the various parameters measured in this borehole is sufficient to demonstrate that it would be impossible to predict site-specific scenarios in an opencast mine. At best, general conclusions in terms of the probability of acidification can be made on a mine or regional basis.

The second example (Figure 77) is from the same mine, though some 500 m away from the first borehole. The following are the main conclusions from this example:

- The pH of the water is slightly below neutral, at the buffering level of the calcium/magnesium carbonate in the rock. The increase in the pH from top to bottom suggests that some of the spoil above the water is at lower pH-levels. This lower source of pH-values is probably from the material in the unsaturated zone.
- The conclusion is that the buffering effect of the sodium minerals in the spoil has been depleted, otherwise the pH of the water would have been in the range of 8.0 – 9.5.
- The specific conductance of the water is moderately high, increasing with depth. This suggests a through-flow system, with rainfall recharge though the unsaturated spoil of 29 m thickness. The stratification in the spoil-water quality is usually more pronounced after the rainfall season than after the winter, if measured over the period of a year.
- Redox conditions in the water are reducing and the temperature of the spoil water is low. These suggest minimal on-site oxidation.
- The vast difference in chemistry of the spoil water at this locality, compared to the preceding one, sufficiently illustrates the lateral variation in spoil water chemistry to be expected in these collieries. There is no way in which this can be predicted on a site-specific basis in a mine.
- General guidelines can, however, be provided, based on detailed static acid-base accounting and working with average values thereafter.

The third example (Figure 78) was drawn from a borehole into an underground mine. Discussion of these results is as follows:

- The mine has been stooped and the roof of the coal has collapsed. The overlying strata are highly permeable and rainwater recharges through the cracks.
- In essence, this system is therefore similar to that in opencast mines, where water from rainfall recharges the mine.
- The essential difference between the two systems lies in the fact that the spoil in an opencast mine has been mixed, whereas the rock in the fractured roof is still in the original sequence. In addition, spoil has material of all sizes while in the underground mine, the fracture frequency decreases from the mine upwards. At the surface, only a few major cracks show. These form preferred pathways for rainwater to infiltrate.
- Chemical oxidation along these cracks is possible, but the flux of rainwater is sufficient to carry all by-products from oxidation into the mine below. The fractured strata are therefore almost completely leached of available salt at any point in time.
- The logs from chemical probing confirm this. The specific conductance

shows a pronounced increase as the probe enters into the mine workings. The other parameters measured show similar trends.

- The high dissolved oxygen concentration in the column above the mine is at surface concentrations, confirming active recharge.
- The pH-levels in the mine are at the buffering level of calcium/magnesium carbonate, confirming that sodium buffering is depleted.
- The temperature of the mine water is low, suggesting that active pyrite oxidation is subdued through flooding of the mine workings.
- The shape of these chemical profiles is typical for almost all underground mines. Very often though, when sampling water from underground mines, water is taken from the top of the water column. This is then reported as water coming from the mine. It is suggested that all mines that report "almost pristine water qualities" in the underground workings, should invest in good sampling equipment that allows stratigraphic sampling. Pressurised pneumatic samplers are the only ones that work.

In conclusion, the system becomes so complicated on a site-specific basis, that it is impossible to predict the short- or long-term water chemistry with any degree of accuracy. In the general sense though, the deterioration of the mine-water quality follows empirical rules that can be listed. Some of these are:

Acid generation

- All the collieries in the Mpumalanga Coalfields have the potential to acidify with time.
- The net acid-generating potential varies from colliery to colliery and within collieries.
- It does not only depend on the mineralogy of the samples but also on the mining method.
- Acidification starts in a patchy fashion with favourable microclimates for acid production establishing themselves in isolated areas.
- In underground mines, these could be moist areas on the outside of coal pillars; particularly where the base potential of the coal and rock is leached by the dynamic flow of mine water.
- In opencast spoil, acidification starts on the outside of boulders, rocks and grains of material. This is referred to as the "skin effect". Many instances have been recorded in the Mpumalanga collieries where the outside of the spoil material would be at a pH of 3.0, with its inside above 7.0.
- The skin effect starts soon after mining has disturbed the rock. This could be within a year of two after mining.
- Water that flows in the unsaturated state across material altered by the skin effect, will report as acid water to the bulk of the mine water. This is why so many of the mines have acid seeps while mining.
- Flooding of areas impacted upon by the skin effect will usually result in a

neutral pH. Flooding saturates the rock and coal within water and the inside of the material, which is at a neutral pH, partakes in the reaction. Many mines, such as New Largo, Minnaar, TNC and Ermelo, can be sited where acid water was encountered during mining, but where the mine water has returned to a neutral pH after partial or complete flooding.

- The degree of flooding in a mine is a controlling factor in terms of the long-term stability of the neutral pH. Some of the mines can be flooded totally, where others will decant through barriers or onto the surface, before total flooding can be reached.
- In mines that are partially flooded, the unflooded portion will continue to produce acid water. The acidic water drains into the flooded areas, where it progressively depletes the base potential.
- Once the water is acidic, acid production is a self-sustaining reaction through electron transfer and does not require the presence of oxygen.
- Great care should therefore be exercised in mines, not to let acidic water enter uncontrolled into areas of neutral water. This can be done through proper planning while mining, thus allowing for natural or artificial barriers to separate unflooded and flooded areas.
- It is currently acceptable practice in the coal industry to drain seepage water from surface discards into underground workings for storage of this water. This practice should be reviewed continuously by monitoring the pH of the water into the mines and that on the coal seam horizon. In the event that the underground water becomes acidic, lime water should be introduced into the underground workings to ensure neutral pH conditions. The seepage water from the discard should also be neutralised on surface before introducing it into the mine.

Base neutralisation

- Base neutralisation is only available in areas inundated with water, since acidic water actually has to come in contact with base minerals before any neutralisation can occur.
- In the unsaturated zone, such as spoil in opencast mining or coal pillars in unflooded underground workings, the base potential included in the material is useless for acid neutralisation because the acid does not reach there.
- Once flooded, the base potential becomes available since the totality of the material becomes part of the overall hydraulic system.
- In instances of insufficient base potentiall, this water will become acidic.

Salt content

- The salt content of the mine water goes through several phases during the evolutionary process mentioned above.
- Initially, mining exposes the rock and coal to water and air. Leaching of the

soluble constituents occurs. This is more obvious in mines south of Kriel Colliery, where the sediments have a high sodium content. North of this mine, circulating groundwater has already leached most of the soluble constituents from the rock. The northern mines therefore do not commonly face a high sodium problem. In the south, sodium levels may exceed 600 mg/L, which renders the mine water useless for most common applications.

- Associated with high sodium levels, is a high alkalinity content. The latter is useful for acid neutralisation. Values between 200 – 500 mg/L alkalinity have been reported in mine water from the southern portion of the coalfields. Most of these mines have excess water and have to discharge it on a periodic basis. Through this action, the alkalinity is flushed from the mines, rendering the underground workings vulnerable to acidification.
- Oxidation of pyrite in the mines with high sodium concentrations is relatively slow. This oxidation process is mainly of a chemical nature, because bacterial oxidation does not occur at pH-levels above 5.5. However, microclimates establish in these mines, and acid generation is locally accelerated. As an exception, spots of acidification have lately been reported in some of the southern mines. It should, however, be realised that these collieries have been mining for more than 20 years and that some acidification may be expected after such a long time. In these mines, sulphate concentrations are usually less than 2 000 mg/L. This is ascribed to the dynamic flow of water through the mine, rather than precipitation of the excess sulphate in the mine. Calcium concentrations are typically in the range of 200 – 300 mg/L.
- Oxidation of pyrite in the northern mines is much faster than in the south. Instances occur where sediments are acid the minute they are exposed to air. In these areas, the natural base potential of the sediments has been depleted before mining, by circulating groundwater. At low pH-levels, sulphate concentrations could be in excess of 10 000 mg/L, while calcium values could be as low as 100 mg/L. The total dissolved solids in acid mine water are often the same as in neutral mine water but a shift in the constituent concentrations has occurred.

Overall chemistry

The above information should be sufficient to provide general guidelines in terms of salt concentrations in the mine water of the Mpumalanga coalfields. In terms of water quality management, many strategies have often been proposed to solve salinity-related problems. Many of these have been so-called “on the spur of the moment solutions” in which the wider picture is not considered. Other solutions have been “because the DWA&F would accept this”. A third kind of solution has been the “future technology solution”. In computer technology, there is a term especially for this type of solution – they call it “vaporware”. In the mining industry, the real solution, namely the “50 years in advance solution” has never been investigated or applied. Secunda Mines and New Denmark Colliery are probably furthest along the line of finding and

applying solutions for the future.

Resistance to the application of long-term and lasting solutions is often found from both the government and mining companies. The government is cautious and the companies would like to minimise current expenditure. It has long been time that they work constructively towards a common goal, namely to find and apply the “50 years in advance solution”. All the information to plan for intermine flow and to provide for future systems through which its impact can be minimised, exists. The section that follows details some of the options available.

9 MANAGEMENT OPTIONS

Management options to cope with intermine flow are applicable on either a catchment basis or mine basis. Those on a catchment basis are:

- The reduction of the number of decanting points through the interconnection of mines.
- The control of decanting positions through interconnection of mine workings.
- The combined treatment of decanting mine water at convenient locations.

On a mine basis, the following actions will markedly reduce the volume of water or the amount of salt to be discharged:

- Design long-term water-management schemes taking cognisance of neighbouring mining activities.
- Design the mine lay-out to retain as much of the mine water as possible in the underground workings whilst mining.
- Investigate coal-barrier characteristics and design coal extraction accordingly.
- Minimise water ingress into mines to reduce water volumes, if required.
- Mix mine water of various qualities to achieve the best possible quality before treated flood discharge.
- Minimise salt loads.
    - Flood mine workings as soon as possible.
    - Flush flooded mines.
    - Utilise the natural neutralisation potential of the coal and rock.
- Some of the options previously considered and that cannot, in the opinion of the authors, be applied successfully in the Mpumalanga collieries, are also discussed below.

9.1 REGIONAL MANAGEMENT OPTIONS

Several major paleo-drainage channels are present on the No. 2 Coal Seam floor. If

no barriers existed, much of the water on this coal seam horizon would drain towards the paleo-channels, where it would dam up. By providing for controlled release systems through barriers, much of the mine water could therefore be drained naturally towards centrally located areas, from where abstraction and treatment can be done in a controlled fashion. An excellent example of such an area where this will be possible, would be in the central Witbank Coalfield, where water from Tavistock, Rietspruit, Douglas, New Clydesdale, TNC and SACE mines could centrally be controlled and, if required, treated before flood discharged. This would typically involve some 100 ML/d.

Another example is the diversion of much of the decant water from New Denmark Colliery into the Waterval River. This would contribute an additional 15 ML/d to the 90 ML/d anticipated from the Secunda mines. Many other such examples, though not on the same scale, can be found on a local basis. It is obvious that significant advances in water-quality management can be made by planning the possible transfer of salt to centralised treatment facilities. Here acid water could be blended with alkaline water, for instance. Settlement of iron from the mine water should be allowed, before release of the water during flood events.

9.2 LOCALISED WATER MANAGEMENT OPTIONS AND NON-OPTIONS

9.2.1 Neighbouring mining activities

None of the mines visited during this investigation were sufficiently informed on mining activities in neighbouring mines. This is a serious shortcoming in the overall planning of intermine flow management. Much more emphasis should be placed on information exchange with adjacent mines. Joint water management committees should be established. This should be extended to the regional level, where corporate houses should contemplate management strategies for catchments.

9.2.2 Containment of mine water for flooding and neutralisation purposes

Mine water has historically been pumped from active workings to allow unhindered coal production. Almost no consideration has been given to the best management strategy for water while mining. Yet, this is simple: Mine from deep to shallow and leave water behind in the mined-out workings. This strategy has, for the past few years, been applied in several of the larger collieries with significant success. The advantage of this mining sequence does not only lie in managing water volumes, but also in water quality management. Mined-out areas are flooded, thus excluding oxygen. Furthermore, the natural alkalinity of the water is not flushed from the rock. This counteracts acidification.

9.2.3 Design of barrier pillars

Barrier pillars between mines or compartments should be designed for the specific

purpose of keeping water from transgressing through it; or to release water at a specific rate through controlled flow mechanisms. Only one of the mines has ever done permeability testing in a barrier pillar. It is essential that barrier pillars be tested and that monitoring systems be installed before intermine flow takes place in an uncontrolled fashion. Issues such as liability and accountability can only be addressed when these systems are in place.

9.2.4 Minimising water volumes

The minimisation of water volumes in mines leads to high salt concentrations and smaller volumes of water that need to be handled. Up to now, minimisation strategies on the scale of the problem have not been applied to any significant extent as a management tool. This is largely due to the vast scale of coal mining operations. It is almost impossible to change a mining strategy, for instance, from underground high extraction to bord-and-pillar once mining has commenced. Other water minimisation strategies include the artificial flooding of closed mines; improved rehabilitation of surface and the planting of trees. The conclusion is that water influx minimisation should be weighed carefully against active flushing, before a decision is made.

9.2.5 Mixing of mine water

At most of the larger mines, the opportunity exists for mine water of different qualities to be mixed, thus improving the overall water quality. Typical benefits of doing this would lie in pH adjustment and iron precipitation. For the latter, retention of the mine water in a surface-holding facility where aeration is possible, is necessary. Such a facility could also be used for quick release of the water during flood discharge. Very few other chemical benefits would be forthcoming from mine water mixing, because most of the constituents are undersaturated in this water.

9.2.6 Minimising salt loads

Several options for minimising the salt loads from mines need to be exploited for future use. These typically are: (1) Flooding of mines as soon as possible after closure; (2) Active flushing of flooded mines and (3) Greater utilisation of the natural base potential in the coal and rock for acid water neutralisation. These are all concepts with great potential, but will necessitate a change in the direction of thinking by the controlling authorities. Active flushing would imply the controlled release of salt into a catchment, with the specific purpose of improving the mine water quality to the extent that it would become a useable resource. Computer simulations have suggested that through flushing, the sulphate concentration in a mine could be dropped by 50% every 20 – 40 years, depending on the rate of flushing. Considering that many of the mines will have holding capacities far exceeding that of the Witbank Dam, this is a management option that, at the very least, should be pursued in a couple of the mines, thus establishing a number of field trials.

9.2.7 Isolation of mine water – not an option

Isolation of dirty mine water from its surroundings (zero discharge) has been advocated by the DWA&F for many years. In principle, this is a sound idea. However, in reality it would never be possible to isolate mine workings completely from its surroundings. There will always be some through flow of water, no matter how minute. In an opencast situation, for instance, the pit hydrology could be designed to evaporate excess water in low-lying areas, which are specifically created for this purpose while mining. Such evaporation areas have to cater for a specific rainfall event and anything exceeding such an event will then spill into the stream below. Such systems are currently in place at several collieries, such as Optimum and TNC. These work well, except that they are far from "true containment systems".

Isolation of mine water will lead to concentration of the salt in the water. The chemical reactions that will take place under these conditions can be simulated in the laboratory or through computer modelling. An output from such a simulation is provided in Figure 79.

This shows the increase of all constituents, except that of calcium because of gypsum precipitation. The original water contained calcium in excess of magnesium, but the final water has magnesium levels in excess of 3 000 mg/L. This, in conjunction with the sulphate values of 17 000 mg/L would render the mine water toxic by any standard. It should be emphasised that the chemical reactions in mine spoil cannot be stopped by merely preventing the pit water from spilling into the stream below.

Temporary isolation of mine water, to be released during excessive floods should, in theory be a sound management principle. However, on the scale of mining in Mpumalanga, these well-isolated systems will not be easy to maintain. Spills from them will be more frequent than anticipated. This will defeat the object of isolated and send plumes of toxic mine water into the environment below.

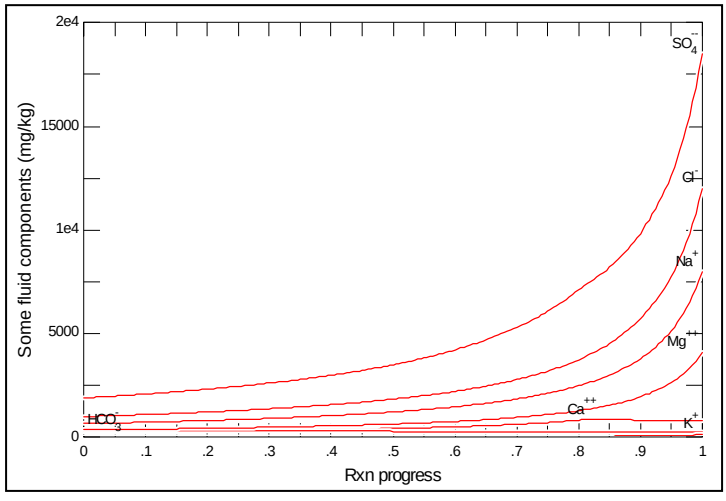


Figure 79. Typical concentration curves form mine water in a closed system.

9.2.8 Wetland treatment of mine water – not an option

The use of wetland for the treatment of mine water has been a failure under South African conditions. Nowhere in the Mpumalanga coalfields do wetlands of any significance exist in terms of mine-water treatment. This is, after more than 25 years of research into the matter. Wetlands, as described in literature are:

- Well-constructed structures kept under anaerobic conditions and harvested periodically to remove heavy metal build-up.
- If not well-maintained, they are nothing but toxic waste sites – the South African wetlands fall into this category.

The information in this document should be sufficient to confirm that the scale of potential mine-water interflow and decanting onto the surface is so great that there is simply not enough space and money to provide for sufficient wetland capacity to make a difference in the long-term mine-water quality.

9.2.9 Future technology – not an option

Many technologies are available for mine water control. These would include prevention, minimisation, desalination and brine disposal technologies. These may, or may not, play an important role in the future. However, before they are fully developed, tested and incorporated into the long-term water-management plan for the Mpumalanga Coalfields, they should not figure in the equation. Mines and authorities should not bargain on future technologies to solve problems. Current problems should be addressed using current solutions. As future improvements become available, these may be implemented.

10 CONCLUSIONS

The following overall conclusions are drawn from this investigation:

- This has been an extensive study in terms of its aerial extent and information gathered. The GIS consists of very detailed mining plans for most of the mines in the Mpumalanga area. This is accompanied by mine-specific information on coal-floor contours, water qualities and quantities. The database is available as part of this document. This should be used for reference to the plans rather than the figures in this document. The zoom facility in the software allows much closer inspection of the data, than what would be possible in the diagrams of the report.
- The GIS software that has been opted for is the Windows Interpretation

System for Hydrogeologists (WISH). It is a well-established software package that has been developed with partial sponsorship by the WRC. It allows simultaneous processing of datasets and maps, followed by the extraction of processed information to well-known word-processing packages. WISH can be acquired from its Web Site: www.uovs.ac.za/igs/wish

- Examples of intermine flow already exist on an alarming scale in the Mpumalanga collieries. This document shows its impact on surface and also at barrier pillars in individual collieries. The conclusion is that the time for action is now. Mines should incorporate mine-water interflow into their planning phase and ensure that this fits in with activities of their neighbouring mines.
- Management options to cope with intermine flow are applicable on either a mine basis or catchment basis. Those on a catchment basis are:
    - The reduction of the number of decanting points through the interconnection of mines.
    - The control of decanting positions through interconnection of mine workings.
    - The combined treatment of decanting mine water at convenient locations.
- On a mine basis, the following actions will markedly reduce the volume of water or the amount of salt to be discharged:
    - Design long-term water management schemes taking cognisance of neighbouring mining activities.
    - Design the mine lay-out to retain as much of the mine water as possible in the underground workings whilst mining.
    - Investigate coal-barrier characteristics and design coal extraction accordingly.
    - Minimise water ingress into mines to reduce water volumes, if required.
    - Mix mine water of various qualities to achieve the best possible quality before flood discharge.
    - Minimise salt loads.
        - Flood mine workings as soon as possible to limit oxidation reactions.
        - Flush flooded mines and use flushed areas for water storage.
        - Utilise the natural neutralisation potential of the coal and rock through proper design.

11 RECOMMENDATION FOR FURTHER WORK

The following are the recommendations for further work in connection with the planning and management of future intermine flow in the Mpumalanga coalfields:

- Technology transfer should happen on an industry level to mining houses,

DWA&F and the DME.

- This should be followed by technology transfer to groups of mines on a catchment basis. Three groups are identified: Olifants River and mines to its west; Klein Olifants River and mines to its east; and the Vaal River. This should be organised through the various Forums for each of these drainage systems.
- On a third level, groups of mines with common interests should be appointed with the responsibility of tackling interrelated issues on mine-water interflow. This would typically constitute the Secunda Mines including New Denmark Colliery; the central collieries of the Witbank area and so forth.
- The fourth level would be assignment of actions on a mine level, such as the design of mine lay-outs and pillars to achieve the necessary goals, including testing the integrity of the barrier structures.
- In the overall sense, the GIS database has to be kept up to date. This in itself is a mammoth task. It should be initiated and financed through the mine corporate offices. Instruction for the submission of updated mine plans and associated information should come from the corporate head offices. This is the only way in which sufficient reaction will be received from the mines.
- Controlling bodies should be formed on three levels, namely the:
    - Inter-corporate level including the DWA&F and the DME, to decide on overall policy matters.
    - Intermine level to decide on, research and implement local water-management strategies in line with those from the inter-corporate level.
    - Mine level, to perform investigations and ensure proper design.

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APPENDIX A – GROUNDWATER FLOW MODELLING

1 PRINCIPLES OF GROUNDWATER FLOW MODELLING

1.1 INTRODUCTION

A groundwater flow model is intended to calculate the bulk, average, rate and direction of the movement of groundwater through aquifers and confining units in the subsurface. These calculations are referred to as simulations.

The simulation of groundwater flow requires a thorough understanding of the hydrogeological characteristics of the site (Bear, 1987). The hydrogeological investigation should include a complete characterisation of the following:

- Subsurface extent and thickness of aquifers and confining units (hydrogeological framework).
- Hydrologic boundaries (also referred to as boundary conditions) which control the rate and direction of movement of groundwater.
- Hydraulic properties of the aquifers and confining units.
- A description of the horizontal and vertical distribution of the hydraulic head throughout the modelled area for both beginning conditions (initial conditions) and later conditions that may vary with time (transient conditions).
- Distribution and magnitude of groundwater recharge, pumping or injection of groundwater, leakage to or from surface water bodies, etc. (sources or sinks, also referred to as stresses). These stresses may be constant (unvarying with time) or may change with time (transient).

The output from the model simulations is the hydraulic heads and groundwater flow rates which are in equilibrium with hydrogeological conditions (hydrogeological framework, hydrologic boundaries, initial and transient conditions, hydraulic properties and sources or sinks) defined for the modelled area (Anderson *et al.,* 1992). Through the processes of model calibration and verification, the values of the different hydrogeological conditions are varied to reduce any disparity between the model simulations and field data, and to improve the accuracy of the model (Bennett, 1976). The model can also be used to simulate possible future changes to the hydraulic head or groundwater flow rates because of future changes in stresses on the aquifer system.

To select an appropriate flow code, one should carefully consider the geology, physics, and, if appropriate, chemistry of the problem to be solved (Freeze *et al.,* 1966). Careful consideration of the conceptual model of fluid flow in a fractured rock setting is critical to successful fracture flow modelling. Inherent in this conceptualisation is the determination of spatial and temporal scales relevant to the problem. Near-field problems may require explicit discrete fracture flow models, while far-field problems may appropriately employ a single continuum model (Bear *et al.,* 1991). For the large number of intermediate situations, dual-continuum models may be used. For cases of chemical or radio-isotope transport, pertinent physical and chemical processes must be identified and considered as to their relative importance in describing mass transport of the constituents of concern. These processes may include advection, dispersion, adsorption/desorption, diffusion, retardation, chemical reactions, phase partitioning, decay and radioactive decay product series.

Research into the nature of fluid flow in fractures and multiphase fluid flow has been conducted by petroleum engineers, mining engineers and hydrogeologists. In the past, economic resource evaluation and production were the motivation for these studies. Today, that motivation is fuelled by world-wide concerns about waste isolation and migration.

Research on fluid flow in fractures and in fractured porous media has a history that spans nearly four decades (Barenblatt *et al.* 1960 and Warren and Root, 1963). This research has focused on four principal aspects of fracture flow:

- Development of conceptual models.
- Development of analytical and numerical solution schemes.
- Description of fracture hydraulic characteristics in static and deforming media.
- Development of stochastic techniques to describe fracture flow and hydrogeological parameter distributions. Bear *et al.* (1991) extensively review the research on fracture flow phenomena.

Researchers have developed several conceptual models describing fluid flow in fractured porous media. Fundamentally, each method can be distinguished on the basis of the storage and flow capabilities of the porous medium and the fracture. The storage characteristics are associated with porosity, and the flow characteristics with permeability. Four conceptual models have dominated the research:

- Explicit discrete fracture.
- Dual continuum.
- Discrete fracture network.

- Single equivalent continuum.

In addition, multiple interacting continua and multi-porosity/multi-permeability conceptual models have recently been introduced in the literature. Further distinctions can be drawn on the basis of the spatial and temporal scales of integration or averaging of the flow regime. Bear and Berkowitz (1987) describe four scales of concern in fracture flow:

- The very near field, where flow occurs in a single fracture and porous medium exchange is possible.
- The near field, where flow occurs in a fractured porous medium and each fracture is described in detail.
- The far field, where flow occurs in two overlapping continua with mass exchanged through coupling parameters.
- The very far field, where fracture flow occurs, on average, in an equivalent porous medium. Figure 80 shows the representation of averaging volumes (REV) in the hydrological system (Diersch, 1980 and Kolditz *et al.,* 1994).

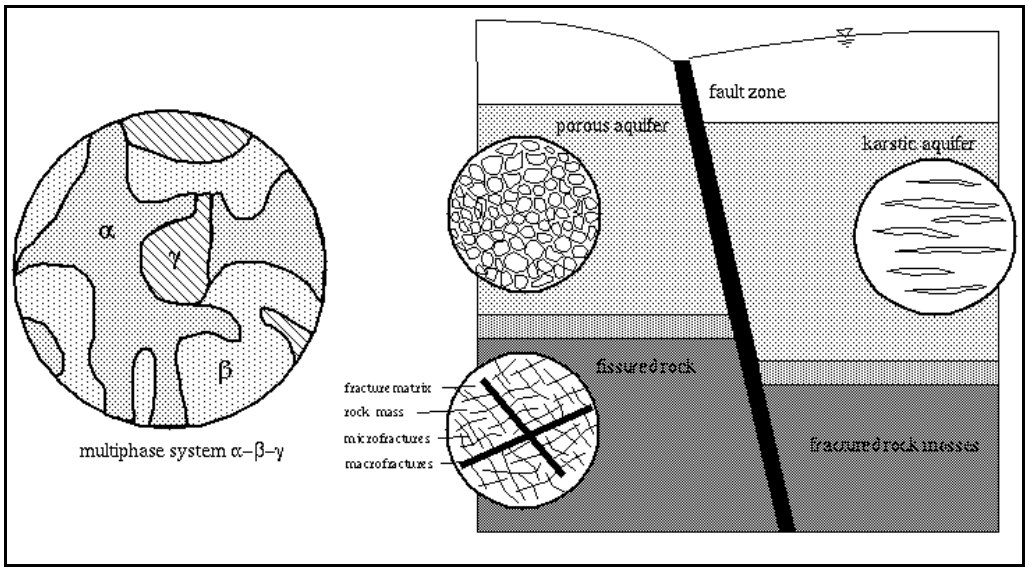

Figure 80. Representation of averaging volumes (REV) in the hydrological system (modified from Diersch, 1980 and Kolditz and Zielke, 1994).

Groundwater modelling is a tool that can analyse many groundwater problems. A model is designed to represent a simplified version of reality. Models are useful for reconnaissance studies preceding field investigations, for interpretative studies following the field program and for predictive studies to estimate future field behaviour. In addition to these applications, models are useful for studying various types of flow behaviour by examining hypothetical aquifer problems. Creating numerical groundwater models of

field problems requires careful attention to describing the problem domain, selecting boundary conditions, assigning model parameters and calibrating the model.

A model is based on physical principles, as well as on the hydrological setting and available measured data. The validity of the predictions depends on how well the model approximates field conditions. Reliable results can only be obtained from a model which is based on detailed input. Wang and Anderson (1982), however, state that a system modelled with inadequate field data may still be useful to identify areas where more detailed field data are necessary. Before attempting such studies, however, one must be familiar with groundwater modelling concepts and model usage.

1.2 HYDRODYNAMIC EQUATIONS

1.2.1 GROUNDWATER FLOW - FLOW EQUATION

The two-dimensional equation governing the flow of groundwater is derived by combining appropriate forms of Darcy's law and the continuity equation. In 1856, Henry Darcy investigated the flow of water in vertical homogeneous sand filters in connection with the fountains of the city of Dijon (France).

$$Q = \frac{KA(h1 - h2)}{L} \qquad \textit{(Equation 1-1)}$$

From his experiments, Darcy concluded that the rate of flow, Q, is proportional to the cross-sectional area A, proportional to (h_1 - h_2), and inversely proportional to the length L, and K is a coefficient of proportionality.

The length h_1 and h_2 are measured with respect to some arbitrary datum level. One can easily recognise that here h is the piezometric head and h_1 - h_2 is the difference in piezometric head across the filter of length L (Figure 81).

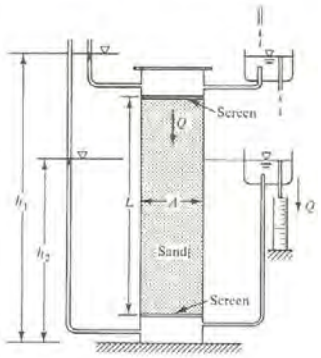

Figure 81. Darcy's experiment (after Bear, 1979).

1.2.1.1 THE CONTINUITY EQUATION

$$\nabla(mv) + q = s\frac{\partial h}{\partial t} \qquad \textit{(Equation 1-2)}$$

Combining these two equations (7-1 and 7-2) gives the flow equation:

$$\nabla(mK\nabla h) + q = S\frac{\partial h}{\partial t} \qquad \textit{(Equation 1-3)}$$

or

$$\nabla(T\nabla h) + q = S\frac{\partial h}{\partial t} \qquad \textit{(Equation 1-4)}$$

where:

V (m/day) = (v_x,v_y) the two-dimensional vector of specific flow rate.

∇ (1/m) = $(\frac{\partial}{\partial x}, \frac{\partial}{\partial y})$ gradient operator.

K (m/day) = the hydraulic conductivity tensor.

h (m) = piezometric head or water table elevation.

m (m) = thickness of saturated flow.

q (m3/day/m2)= discharge/recharge (- , +) per unit surface area.
T (m2/day) = transmissivity tensor of the aquifer.

S = storage coefficient.

1.2.1.2 MASS TRANSFER EQUATION

This equation can be derived in a similar way to the flow equation from an expression governing dispersion and an equation governing the conservation of mass. The common solute transport equation has the following form:

(For steady state flow, i.e. h and m are constants with time.)

$$\nabla(\frac{D}{R}\nabla c) - \frac{u}{R}\nabla c + \frac{q}{n_e mR}(c_{in} - c) + \frac{S_{\text{int}}}{n_e mR} - \lambda c = \frac{\partial c}{\partial t} \qquad \textit{(Equation 1-5)}$$

Additional terms:

u	(m/day)	= two-dimensional vector of average pore velocity.
c	(mg/m^3)	= concentration of pollutant in groundwater.
c_{in}	(mg/m^3)	= pollution concentration in aquifer inflows.
D	(m^2/day)	= tensor of dispersion in two dimensions.
λ	(l/day)	= degradation/reaction rate of pollutant.
n_e		= effective porosity.
R		= retardation factor = $1 + \rho K(1 - n_e)/n_e$
Si_{nt}	($mg/day/m^2$)	= internal pollution source/sink rate related to unit horizontal area, not to recharge.

1.3 SOLUTION OF THE HYDRODYNAMIC EQUATIONS

Both the flow and the mass transport equations are linear, parabolic, partial differential equations. Both have solutions, as long as certain initial and boundary conditions are specified.

1.3.1 INITIAL AND BOUNDARY CONDITIONS

1.3.2 FLOW EQUATION

The initial conditions are the water levels at a specific time, usually the beginning of the simulation period (t = 0). Water levels should be specified over the whole modelled domain.

The boundary conditions needed are:

- Dirichlet – constants head boundary, which implies that water levels do not change with time.

- Neuman – prescribed flux, stating that flow across a boundary occurs at a specific rate. The simplest case is no-flow, for example, at an impervious dyke. A non-zero specified flux is easier to incorporate as a recharge point.

1.3.3 MASS TRANSPORT EQUATION

Initial conditions:

- These specified the concentration of the chemical species that is going to be traced through the simulation at the starting time. This must be specified over the whole modelled domain.

Boundary conditions:

- Dirichlet: concentrations remain as specified and do not change with time. Inner boundaries or points may be specified as Dirichlet boundaries to preserve pollution sources.
- Neuman boundary conditions specify the concentration gradient normal to the boundary, where they control only the dispersive flux. The advective flux on the boundary is the maximum rate at which a contaminant may move, and is an interplay between the concentrations and velocities which are calculated from head values generated by the flow equation and Darcy's law (Wood *et al.,* 1984).

1.3.4 SOLUTION METHODOLOGY

Exact solutions for these equations may be obtained analytically in regions of simple or regular geometry (Huyakorn and Pinder, 1983). Analytical solutions for the flow equation, with many possible initial and boundary conditions, are described by Kruseman and De Ridder (1991), while an excellent summary of the analytical equations that can be used for solving the flow or mass transport equations are given by Willis and Yeh (1987).

Application of analytical solutions is limited, because:

- They require homogeneity of the aquifer.
- They cannot accommodate complex boundaries.
- Distributed recharge cannot be consistently incorporated into the analytical solution (Wang and Anderson, 1982).

Analytical solutions are useful for analysing pumping test data or for making rough estimates, as the results are obtained quickly. For detailed modelling, a numerical solution is more appropriate. The latter transforms the hydrodynamic partial differential equation into systems of ordinary differential or algebraic equations.

A number of numerical techniques are available. Among them are:

- Finite difference methods.
- Galerkin or finite element methods.
- Collocation methods.
- The method of characteristics.
- Boundary element methods.

These are described in detail in Lapidus and Pinder (1982).

2 THE FINITE ELEMENT METHOD

The method of solution chosen for this investigation is the finite element method, which has the following advantages over other numeric techniques:

- It can accommodate irregular boundary outlines.
- It can accommodate irregularly spaced grids, for example detailed spacing in certain areas to accommodate steep gradients.
- Observation points can occupy any position in the grid.
- Heterogeneity and anisotropy can be accommodated (Segerlind, 1976).

The finite element method employs a "piecewise approximation" over the modelled area, which is divided into a series of finite elements that are connected at a discrete number of nodal points. This process is called discretisation. Nodes can be constructed, so that they coincide with the physical boundaries and with points within the area.

In the Galerkin approach, a matrix expression is developed to relate the nodal variables via a linearly independent interpolating function (w) over the entire surface domain of the element. An approximate solution (~) for each variable is expressed as a series summation, with each term in the series being the product of the nodal variable (h_1) and the associated nodal interpolation function.

$$h_{(x,y,t)} \approx \hat{h}_{(x,y,t)} = \sum_j h_{i(t)} w_{i(x,y)} \quad i = 1,...no\ of\ nodes \qquad (Equation\ 2\text{-}1)$$

The difference (usually referred to as residual) between the approximate solution ($\hat{h}$) and the true solution (h) is orthogonal to the functions used in the approximation, i.e. at each nodal point, the spatial integral of the residual weighted by the basis function is zero.

In the above discussion, the variable head (h_i) has been used to describe the development, as if for the flow model. The same procedure may be followed with the variable, concentration (c_i), to develop similar matrix expressions for the mass transport model. The latter is, however, more complex as additional terms sometimes referred to as 'convective terms', covering dispersion, diffusion, reaction and degradation of chemical species, must be introduced.

The element matrices are combined in a set of algebraic equations that describes the entire global system. The coefficient matrix of this set of equations is referred to as the global matrix. Boundary conditions are incorporated into the global matrix. The resulting set of simultaneous algebraic equation forms a symmetrical matrix in the case of the flow equation, which can be solved by Cholesky's square root method. This method is efficient in that only half the number of elements have to be stored during the solution of the equations. The mass transport problems require more computer memory. Because of the non-symmetry introduced by the convective terms, this means that all the non-zero coefficients must be stored during the execution of the finite element method (Willis and Yeh, 1987).

The Galerkin approach gives rise to the following equations:

Flow Model

$$\int_{\Omega}\left(-T\frac{\delta}{\delta x}\left(\sum_j h_j w_j\right)\frac{\delta w_i}{\delta x}-T\frac{\delta}{\delta y}\left(\sum_j h_j w_j\right)\frac{\delta w_i}{\delta y}-S\left(\sum_j \frac{\delta h}{\delta t} w_j\right)w_i+qw_i\right)dxdy \qquad \textit{(Equation 2-2)}$$

$$+\int_{\Gamma} q_n w_i ds = 0\left(i=1,.....N\right)$$

where the last term on the left-hand side represents the boundary integral if node i lies on the boundary Γ, and q_n is the flow across the boundary per unit length of boundary (Kinzelbach, 1986).

The equation is in the form;

$$\sum_j G_{ij}h_j+\sum_j P_{ij}\frac{\delta h_j}{\delta t}=f_i \qquad \textit{(Equation 2-3)}$$

$$or\left[G\right]\{h\}+\left[P\right]\left\{\frac{\delta h}{\delta t}\right\}=\{f\} \qquad \textit{(Equation 2-4)}$$

where:

$$G_{ij} = \int_{\Omega}\left(T\frac{\delta w_i}{\delta x}\frac{\delta wj}{\delta x} + T\frac{\delta w_i}{\delta y}\frac{\delta w_j}{\delta y}\right)dxdy \qquad \textit{(Equation 2-5)}$$

$$P_{ij} = \int_{\Omega} sw_i w_j dxdy \qquad \textit{(Equation 2-6)}$$

$$f_i = \int_{\Omega} qw_i dxdy + \int_{\Omega} q_n w_i ds \qquad \textit{(Equation 2-7)}$$

These elemental matrices can be assembled into a global matrix by summation.

Ω = The modelled domain.

Γ= The boundary of the modelled domain.

$\int_{\Omega}$ = Summation over index j always runs from j=1 to j = N

Mass transport equations can be given in a similar way:

2.1 DISCRETISATION

2.1.1 TIME DISCRETISATION

Equations 8 and 11 are systems of ordinary differential equations with respect to time. To solve them, the time derivative is usually discretisised with respect to time, allowing the optimal time step to be chosen (Muller and Botha, 1986).

2.1.2 DOMAIN DISCRETISATION

The continuous domain (fl) is divided into a number of adjacent subareas or elements. Triangular elements with straight sides are the simplest form, which permit accurate representation of an irregular domain.

Discretisation can be divided into two processes:

Division of the body into elements (numbering the nodes)

Division into triangular elements follows a method of dividing the area into quadrilateral and triangular regions. The quadrilaterals are divided into triangular elements by joining the diagonals. Elements with an equilateral shape produce more accurate simulations, thus long narrow triangles should be avoided (Botha and Pinder, 1983). Elements can be varied in shape and size to accommodate irregular outlines. Refined areas with smaller

elements should be introduced in regions where rapid changes in hydraulic values are expected.

Triangular elements are joined by nodes, which have to be numbered, so that specific elements can be identified. The node numbering has an influence on the computational efficiency, since the coefficient matrices which must be solved have a diagonal band of non-zero coefficients. Proper computer programming for their solution uses only the coefficients within this band, thus using computer memory efficiently and speeding up the computations. The width of the band is the horizontal distance from the main diagonal running through the matrix to the outer edge of the band. This is the half-band width. The matrix computations are simplified by keeping the half-band width as small as possible.

The half-band width (B) may also be calculated as $B = (R+I)f^0$

where:

- B = the half-band width.
- R = the largest difference between node numbers of a single element.
- f^0= the number of unknowns at each node (degrees of freedom).
- Labelling the elements is trivial and has no effect on the computational aspects.

Numerical errors are introduced by the solution. They may be controlled to a certain extent, if criteria regarding the discretisation are adhered to (Huyakorn and Pinder,1983).

Oscillations or over- and undershoot occur near the concentration front. Overshoot describes erroneously high concentrations encountered upstream of the moving front. Undershoot relates to erroneous values on the downstream side. Oscillations are more pronounced as the front becomes sharper, i.e. a steep gradient exists and advection dominates over dispersion. If dispersion is specified (i.e. not = 0), then oscillations can be eliminated by the correct selection of element size. This can be determined by the Peclet Number (Pe).

For Pe < 2, oscillations are almost eliminated, but Huyakorn and Pinder (1983) show that for Pe up to 10, only mild oscillations are detectable. Thus, the element size must be selected so that the nodal spacing criteria limit numerical dispersion. This has been investigated by Piver and Lindstrom (1991) and is given as $0< Al <- 2D/|u| < {}^2aL$. Luckner and Schestakow (1991) confirm this approach. However, Kinzelbach (1986) states that Al < aL and that when aT is significant, discretisation should be of the same order of magnitude as aT.

For the above discussion, the following definitions hold:

u (miday) = pore velocity.

Al (m) = node spacing.

D (m^2/day) = dispersion.

aL (m) = longitudinal hydrodynamic dispersivity.

aT (m) = transverse hydrodynamic dispersivity.

The time step size in the time discretisation can be used to control overshoot. This is defined by the Courant criterion (C_o) (Kinzelbach, 1986), which ensures that the contaminant cannot be transported further than 1 grid spacing in a single time step.

3 THE FEFLOW MODEL

"FEFLOW was created in about 1979 as a research-orientated special purpose finite-element program at the Institute for Mechanics of the former Academy of Sciences of the GDR, for modelling two-dimensional nonlinear flow and mass transport processes in saturated porous media" (Diersch, 1996). Originally, it was written in FORTRAN IV, but has been developed further and improved continuously; a process which is still going on today (Diersch, 1996). The program has been rewritten in C++ and can currently run under both UNIX and Windows 95/NT platforms. Today, FEFLOW is an interactive, graphics-based groundwater modelling system which can be used to solve both three- and two-dimensional problems (Diersch, 1996).

According to the WASY Home Page, the structure of the program can be presented as indicated in Figure 82.

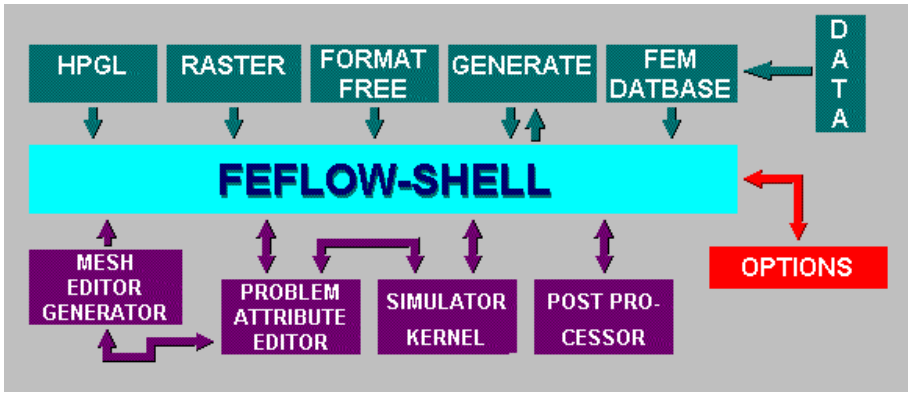

Figure 82. Properties of the program are outlined as follows by the WASY Home Page (http://www.wasy.de).

FEFLOW offers the following integrated properties:

•Galerkin Finite Element Method.

•Streamline, Shock Capturing and Full Upwind Technique.

•Automatic time step control (optional) by means of the Predictor Corrector methods (1st and 2nd order).

•Newton and Picard Iterative Procedures for non-linear problems.

•Fast and direct solvers such as the PCG and Restarted or THOMIN methods with performance enhancing-preconditioning.

•Macroscopic mixed problems by using the Scheidegger Bear Dispersion Model.

•Henry/Freundlich/Langmuir Sorption Isotherms and 1st order chemical non-equilibrium reactions.

•Time-dependent boundary conditions and material data.

•Particle tracking (77K) and isochrone calculations.

•Balancing volume and mass fluxes as well as time integrated mass balances in subregions and boundary sections.

File formats that can be imported and applied in FEFLOW include the following (Diersch, 1996):

- Line, point, polygon and triplet files in ASCII format.
- Shape files of ArcView.
- Raster databases (TIFF).
- Hewlett-Packard Graphics Language (HPGL).

It is thus clear that the GIS (Geographical Information System) interface is an integral component of the system. FEFLOW has been built with an open architecture concept and can thus be modified or extended at any time at relatively low cost (Diersch, 1996).

APPENDIX B - CHEMISTRY OF COLLIERIES

1 INTRODUCTION

Coal mine drainage ranges widely in composition, from acidic to alkaline, typically with elevated concentrations of sulphate (SO_4), iron (Fe), manganese (Mn) and aluminum (Al), as well as common elements such as calcium, sodium, potassium and magnesium. Acid Mine Drainage (AMD), in which mineral acidity exceeds alkalinity, typically contains elevated concentrations of SO_4, Fe, Mn, Al and other ions. AMD may or may not have a low pH, since the presence of dissolved Fe, Al and Mn can generate hydrogen ions by hydrolysis. The major source of acidity is oxidation of pyrite (FeS_2) in freshly broken rock that is exposed by mining. Pyrite oxidation can be rapid upon exposure to humid air or aerated water, particularly above the water table (Rose and Cravotta, 1998).

Upon exposure to oxidising conditions and in the absence of alkaline materials, some sulphate minerals are oxidised in the presence of water and oxygen to form highly acidic, sulphate-rich drainage. Acidity levels, metal composition and concentration depends on the type and amount of sulphate mineral and the presence or absence of alkaline materials. In the Coalfield, when sulphates are present, the oxidation of Fe disulphates and subsequent conversion to acid occurs through several reactions, as discussed in the section below (Skousen *et al.,* 1998).

Acid Mine Drainage (AMD) occurs when sulphide-bearing mineral in rocks is exposed to air and water, changing the sulphide sulphur to sulphuric acid (BC Mining Watch, 1997, Durkin, 1996, Durkin and Herrmann, 1996, Evangelou, 1995, Evangelou and Zhang, 1995, Hadley and Snow, 1974, Moses *et al.,* 1987 and Robertson, 1996). This acid dissolves heavy metals such as lead, zinc, copper, arsenic, selenium, mercury and cadmium into ground and surface water. Certain bacteria, naturally present, can significantly increase the rate of this reaction. The contaminated water is often reddish-brown in colour, indicating high levels of oxidised iron (Hadley and Snow, 1974). AMD and heavy metals pollution can poison ground- and drinking water and can destroy aquatic life and habitat. It can develop at several points throughout the mining process, in underground workings, open pit mine faces, waste rock dumps, tailings deposits and ore stockpiles. Acid generation can last for decades, centuries, or longer, and its impacts can travel many miles downstream. Roman mine sites in Great Britain continue to generate acid drainage, 2000 years after mining ceased (BC Mining Watch, 1997).

Industry, labour, government and environmentalists agree on one issue, that AMD is the number one environmental problem facing the mining industry (BC Mining Watch, 1997, Domvile *et al.,* 1994, Lawrence and Wang, 1997 and Schafer Laboratory, 1997). Waste rock and tailings are the most important sources of AMD. There is no dispute that AMD:

- Devastates fish and aquatic habitat,
- Is virtually impossible to reverse with existing technology,
- Once started, costs millions of dollars annually to treat and can continue for centuries (BC Mining Watch, 1997) and
- Is very complex to control and treat (Lawrence and Day, 1997a and Evangelou, 1995).

AMD is a severe environmental pollution problem associated with coal and other sulphide-containing ore mining operations and results mainly from the oxidation of pyrite. It is estimated that each year in the USA, wet cleaning of coal alone leaves behind aqueous slurries of refuse containing at least 10 million tons of pyrite (Evangelou and Zhang, 1995).

It is only relatively recently that the full implications of its impact have been acknowledged in mine planning. The Equity Silver Mine, in northern BC, for example, received permits to operate in 1979 without proper consideration for the potential of AMD. Within months, it became an infamous site of contamination, due to the generation of metal contaminated and acidic drainage from waste rock, tailings and many structures around the mine site which had been constructed from acid-generating rock (Lawrence, 1997a).

Bright orange-coloured water and stained rocks are usually tell-tale signs of Acid Mine Drainage. The orange colour is due to ferric hydroxide ($Fe(OH)_3$) (yellow boy) precipitating out of the water. The precipitate forms as the Acid Mine Drainage becomes neutralised. At low pH-values, the metal ions remain soluble. When the pH rises, the iron oxidises and precipitates out. Depending on the conditions, the orange-coloured precipitates may form inside the mine or several miles downstream. The precipitates can be harmful to aquatic life in several ways. The clumps reduce the amount of light that can penetrate the water, which affects photosynthesis and the vision of animal life. When the precipitate settles, it blankets the streambed smothering the bottom dwellers and their food resources (Acid Mine Drainage Chemistry, 1997). Heavy metals kill because they prevent the energy that is essential for life from being produced by binding to –SH groups (Stevenson, 1997).

2 STAGES IN THE DEVELOPMENT OF MINE SITE DRAINAGE

Morin and Hutt (1997) present a section dealing with the evolution of drainage chemistry from typical mine sites. They stress the importance of primary dissolution and the formation and subsequent reaction of secondary minerals to describe the hydrochemical evolution at most mines.

According to Morin and Hutt (as illustrated in Figure 83), the evolution is divided into three stages, with the first stage involving the rapid and often kinetically driven reaction of sulphates (Rate 1) and the subsequent neutralisation reactions. These rates decrease over time as minerals become depleted and equilibrium concentrations are approached, but the secondary minerals are so rapidly formed that a state of supersaturation exists. In time, these minerals will precipitate and the remaining components of the system create the observed chemistry from the mine (Rate 2).

The second stage starts when all the primary minerals are exhausted or the reaction rates are almost negligible. The precipitated secondary minerals will now typically begin dissolving, usually under equilibrium conditions, resulting in the observed chemistry at this stage. Due to this equilibrium dependence, the Rate 2 reactions are fairly predictable and constant throughout the first two stages.

The final stage begins when only the very slow reacting primary and secondary minerals remain in abundance. The observed chemistry under these conditions shows typically low loadings and concentrations.

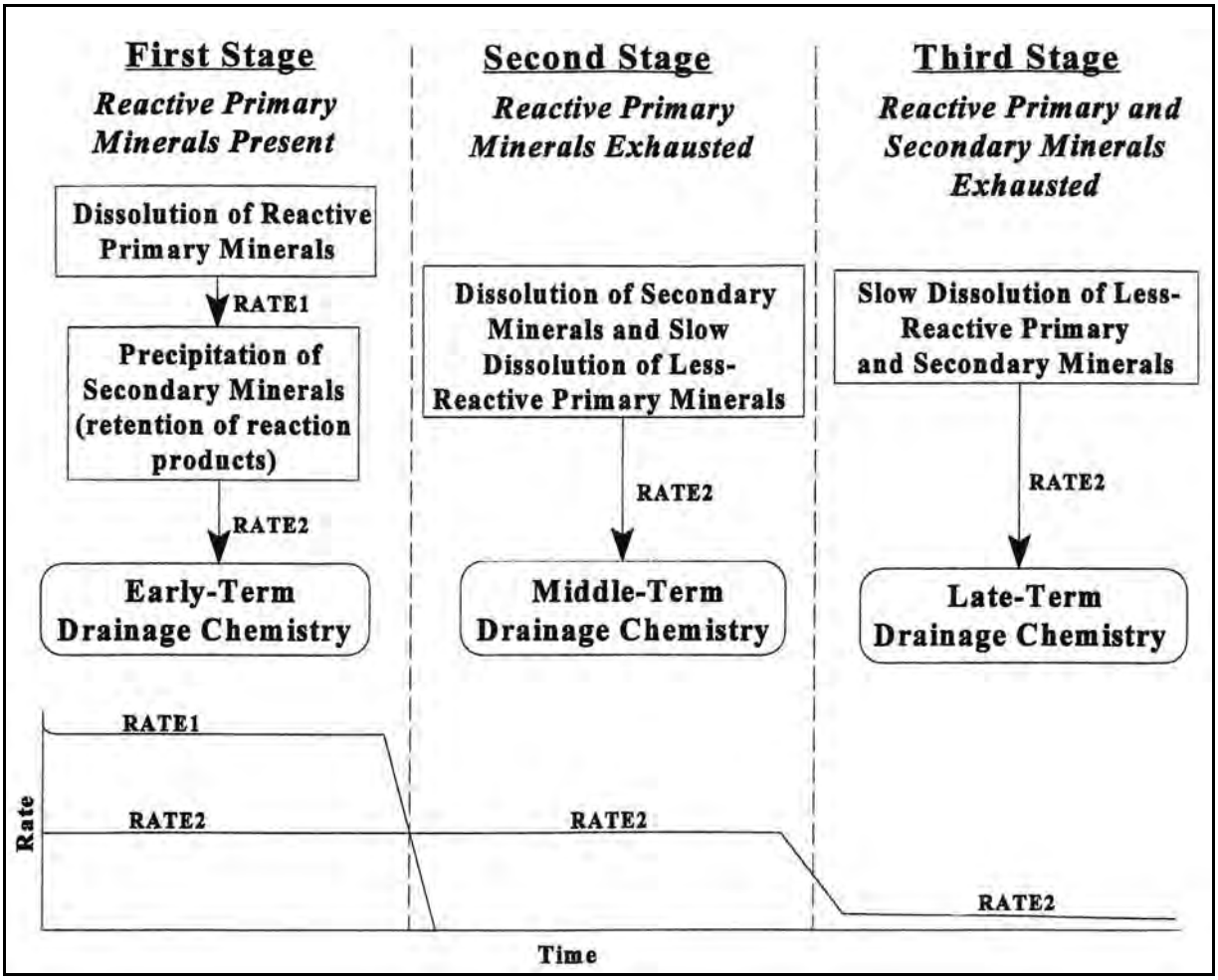

Figure 83. The three stages in the evolution of drainage chemistry (Morin and Hutt, 1997).

2.1 STAGES IN DEVELOPMENT OF AMD

The development of AMD involves a complex combination of organic and sometimes inorganic processes and reactions. To produce severe acid drainage, where the pH of the system drops below 3, sulphide minerals must create an optimum micro environment for rapid oxidation and must continue to oxidise for a sufficiently long time to exhaust all the neutralisation potential of the rock (Figure 84). The potential of sulphide rock to generate acid is strongly related to the amount of alkaline, often calcareous, material in the rock. For example, a rock containing 5% sulphide minerals may not generate acid due to an over-abundance of calcite in the rock that is available for acid neutralisation. Another rock, containing less than 2 percent sulphide minerals, might generate a considerable amount of acid if no neutralising minerals are present within it.

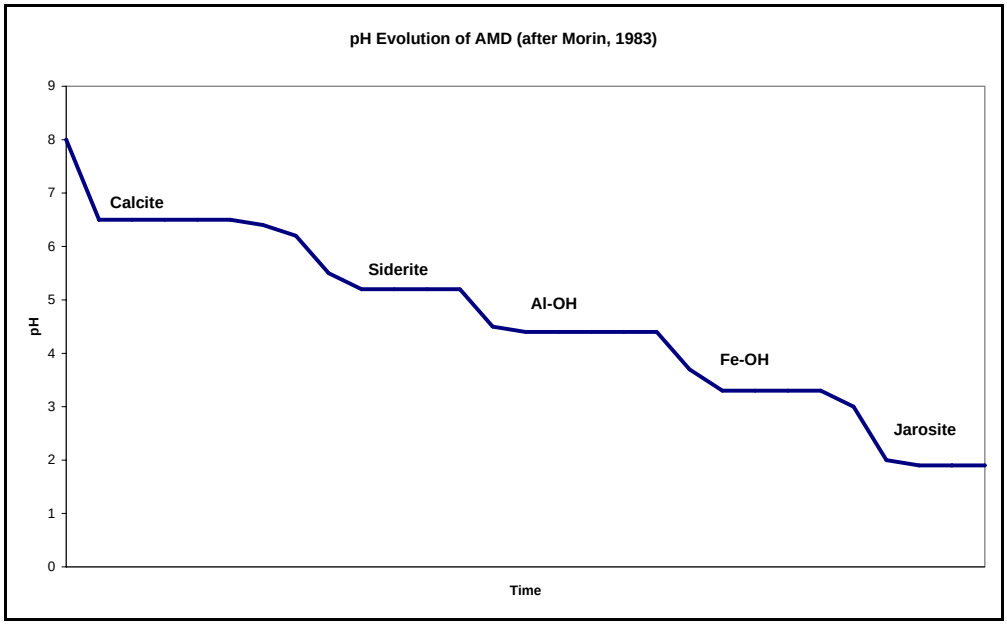

Figure 84. Schematic evolution of pH plateaux (after Morin, 1983 and Morin, 1988).

When reactive sulphide rock is initially exposed to flowing water and oxygen, sulphide oxidation and acid generation begins. Any calcium-based carbonate in the rock immediately neutralises this small amount of acidity and maintains neutral to alkaline conditions in water passing over the rock (Broughton *et al.,* 1992). As acid generation continues and the neutralising agent is consumed or is rendered ineffective in further neutralisation, the pH of the water decreases, which, in turn, enhances the conditions for further acid generation. As the rate of acid generation accelerates, the pH progressively decreases in a step-like manner. Each plateau of relatively steady pH represents the dissolution of a neutralising mineral that becomes soluble at that pH. If the rate of acid generation remains high enough to remove all of the neutralisation potential in the rock, the pH-values will drop below 3 and AMD will become severe. The various stages can last for weeks, months, or centuries, until the sulphide minerals completely oxidise and the rock becomes inert or until special waste management and AMD control actions are taken (Durkin and Herrmann, 1996).

3 BASIC CHEMISTRY OF AMD GENERATION

3.1 REACTIONS

The geochemistry of AMD has been the subject of numerous investigations. Some general references on the subject include publications by Temple and Koehler (1954), Singer and Stumm (1970), Kleinmann *et al.* (1981), Nordstrom (1982), Williams *et al.*

(1982), Hornberger *et al.* (1990), Alpers and Blowes (1994), Blowes and Jambor (1994), Evangelou (1995) and Nordstrom and Alpers (1996).

Acid Mine Drainage impacts stream and river ecosystems in several ways through acidity, ferric ion (Fe^{3+}) precipitation, oxygen depletion and release of heavy metals associated with coal and metal mining, such as aluminium (Al^{3+}), zinc (Zn^{2+}) and manganese (Mn^{2+}) (AMD Chemistry, 1997).

When mineral deposits that contain sulphides are mined, they have the potential to produce Acid Mine Drainage. The mineral pyrite, more commonly known as "fool's gold", is iron disulphide (FeS_2). Pyrite is one of the most important sulphides found in the waste rock of mines. Much of the pyrite present in rocks and overburdens of coal mines occurs as very small crystalline grains intimately mixed in sandstones and shales (Temple and Koehler, 1954). When exposed to water and oxygen, it can react to form sulphuric acid (H_2SO_4). Because of its wide distribution in coal and overburden rocks, especially in shales of marine and brackish water origin, pyrite is recognised as the major source of acidic drainages (Rose and Cravotta, 1999).

The following oxidation and reduction reactions express the breakdown of pyrite that leads to Acid Mine Drainage.

The following reactions characterise various stages in the complete reaction (Stumm and Morgan, 1981, p. 470):

$(1)\ FeS_2 + 7/2\ O_2 + H_2O \Rightarrow Fe^{2+} + 2SO_4^{2-} + 2H^+$ *(Equation 3-1)*

$(2)\ Fe^{2+} + 1/4O_2 + H+ \Rightarrow Fe^{3+} + 1/2\ H_2O$ *(rate limiting step)* *(Equation 3-2)*

$(3)\ Fe^{3+} + 3H_2O \Rightarrow Fe(OH)_3$ *(yellow boy)* $+ 3H^+$ *(Equation 3-3)*

$(4)\ FeS_2 + 14Fe^{3+} + 8H_2O \Rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+$ *(Equation 3-4)*

Reaction 1 shows oxidation of the disulphide, thus releasing ferrous iron (Fe^{2+}) and two protons. In Reaction 2, the ferrous iron is oxidised to ferric (Fe^{3+}) which hydrolyses to form ferric hydroxide (an insoluble compound at pH greater than 3,5) and in the process as shown in Reaction 3, three more protons are released. Thus, for every mole of pyrite, five protons are released. However, since one proton is consumed for the oxidation of ferrous to ferric, only four protons are actually produced (Evangelou, 1995). Upon initiation of pyrite oxidation, the ferric iron can be reduced by the pyrite itself, as shown in Reaction 4 (Stumm and Morgan, 1970 and Singer and Stumm, 1970).

Therefore, pyrite continues to oxidise as long as ferric iron is generated. The conversion of ferrous to ferric is also the rate limiting step in the oxidation of pyrite. However, since

oxidation of ferrous to ferric in the pH-range of 3 is extremely slow (half-life in the order of 100 days), it appears that pyrite oxidation in this pH-range would be extremely slow, unless oxidation of ferrous at low pH is catalysed by micro-organisms (Singer and Stumm, 1970). It turns out that in the pH-range of 2.5 – 3.5 (Jaynes *et al.,* 1984), *Thiobacillus ferrooxidans* rapidly oxidise ferrous iron to ferric iron. Iron-oxidising bacteria can accelerate the rate of Fe^{2+} oxidation by a factor of 10^6 (Singer and Stumm, 1970 and Dugan, 1975). Also, sulphur oxidising bacteria such as *T. thiooxidans and T. ferrooxidans* can eliminate the need for ferric iron when in the presence of oxygen and some organic substrates (Evangelou, 1995). These bacteria, which require only dissolved CO_2, O_2, a reduced form of Fe or S and minor N and P for their metabolism, produce enzymes which catalyse the oxidation reactions and use the energy released to transform inorganic carbon into cellular matter (Temple and Delchamps, 1953, Kleinmann *et al.,* 1981, Nordstrom, 1982 and Ehrlich, 1990). At low pH (<4.5), pyrite is oxidised by Fe^{3+} much more rapidly (Appelo and Postma, 1993) than by O_2 and more rapidly than dissolved Fe^{2+} is oxidised by O_2 to Fe^{3+}(Evangelou, 1995). This process rapidly consumes all Fe^{3+} and pyrite oxidation would cease unless Fe^{3+} is replenished by the process of oxidation of Fe^{2+} by oxygen (Apello and Postma, 1993). For this reason, Reaction 2 is known to be the rate limiting step in abiotic pyrite oxidation (Singer and Stumm, 1970).

T. ferrooxidans is an acidophilic chemolithotrophic organism that is ubiquitous in geologic environments containing pyrite (Nordstrom, 1982). Thus, in the presence of *T. ferrooxidans* and under low pH-conditions, pyrite oxidation can be described by Reactions 2 and 4. Reaction 3, taking place at pH-values as low as 3, is a readily reversible dissolution/precipitation reaction that serves as a source as well as a sink of solution Fe^{3+}, and is a major step in the release of acid to the environment (Evangelou, 1995).

Acidity is caused by the liberation of hydrogen ions (H^+) in three of the four reaction steps. Production of Acid Mine Drainage can occur long after mines have been abandoned if piles of waste rock are in contact with air and water. The red colour often seen in streams receiving Acid Mine Drainage is actually a stain on the rocks called "Yellow-Boy," or Ferrous Hydroxide ($Fe(OH)_3$) formed during Reaction 3 above (AMD Chemistry, 1997).

The overall chemical reaction can be simplified to:

$$FeS_{2(s)} + 15/4O_2 + 7/2\ H_2O \Rightarrow Fe(OH)_3 + 2SO_4^{2-} + 4H^+ \qquad \textit{(Equation 3-5)}$$

In the above reaction, every mole of pyrite yields four moles of acidity (Cohen, 1996, Durkin and Herrmann, 1996, Kempton *et al.,* 1997 and Morin and Hutt, 1997). Notice the necessity for air and water (although the process can occur in a dry environment)

(Cohen, 1996). Reactions 1 - 4 can also be seen in the model (Figure 85) by Stumm and Morgan (1981).

Acidity is commonly measured by pH-values, which are easy to collect and compare. pH is not always a good indicator of Acid Mine Drainage, because it only indicates the concentration of hydrogen ions present. When evaluating the extent of Acid Mine Drainage, it is important to know the amount of hydrogen ions remaining in solution after the natural buffering of the stream is used up. For example, pH-measurements may not detect heavy Acid Mine Drainage in a stream because of the high alkalinity due to dissolved carbonates (Evangelou, 1995 and Evangelou and Zhang, 1995).

Most iron released during the initial stages of pyrite oxidation ends up as iron hydroxide due to the relatively high pH on pyrite surfaces (Fornasiero *et al.,* 1992 and Ivanov, 1962). However, when pH in the vicinity of the pyrite surface drops below 3,5, the activity of free Fe^{3+} in solution increases and oxidation of pyrite by Fe^{3+} becomes the main mechanism for acid production. Singer and Stumm (1970) reported that Fe^{3+} can oxidise pyrite at a much higher rate than O_2. This can be seen in the model for the oxidation of pyrite (Stumm and Morgan, 1981) and the following reaction (Evangelou and Zhang, 1995).

$$FeS_2 + 14Fe^{3+} + 8H_2O \Leftrightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \qquad \textit{(Equation 3-6)}$$

Iron-oxidising bacteria

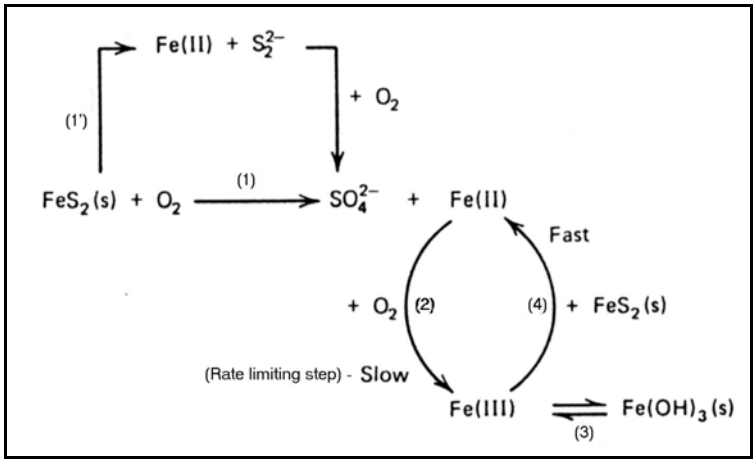

Figure 85. Model for the oxidation of pyrite (from Stumm and Morgan, 1981).

It has been reported that pyrite in mining waste or coal overburden is initially oxidised by the atmospheric O_2, producing H^+, SO_4^{2-} and Fe^{2+}. The Fe^{2+} can be further oxidised by O_2 into Fe^{3+} which in turn hydrolyses and precipitates as amorphous iron hydroxide releasing additional amounts of acid (Nordstrom, 1982). During this initial stage, pyrite

oxidation is a relatively slow process (Ivanov, 1962). However, as acid production continues and the pH in the vicinity of the pyrite surface drops below 3,5, formation of ferric hydroxide is hindered and activity of Fe^{3+} in solution increases. Under these conditions, oxidation of pyrite by Fe^{3+} becomes the main mechanism for acid production in mining waste (Singer and Stumm, 1970 and Moses *et al.,* 1987).

It may be seen that in addition to pyrite, the presence of both oxygen and water is required for process progression (Evangelou, 1995 and Mills, 1998). The process is complex because it involves chemical, biological and electrochemical reactions and varies with environmental conditions. Factors such as pH, pO_2, specific surface and morphology of pyrite, presence or absence of bacteria and/or clay minerals, as well as hydrological factors, determine the rate of oxidation. There is, therefore, no single rate law available to describe the overall kinetics of pyrite oxidation for all cases (Evangelou, 1995). This has important ramifications in that removal of the oxygen source (e.g. by total submersion under water) or the water source (e.g. conditions of aridity) will halt AMD production. If any of the processes represented by the equations were slowed or altogether stopped, the generation of AMD would also slow or cease. Removal of air and/or water, two of the three principal reactants, from the system would stop pyrite from being oxidised. An almost complete absence of oxygen occurs in nature when pyrite is found beneath the water table where oxidising conditions are limited. Under these conditions, the pyrite remains almost completely unreacted. When pyrite is enclosed within massive rock, only minimal amounts of pyrite are oxidised through natural weathering, thereby generating only small amounts of acid, and this acid is sometimes naturally diluted or neutralised by surrounding alkaline rocks. However, when large volumes of pyritic material are fractured and exposed to oxidising conditions, which can occur in mining or other major land disturbances, the pyrite reacts and water dissolves and moves the reaction products (Fe and other metals, sulphate, and acid) into ground and surface water sources.

AMD production would also be considerably slowed or halted by the termination of *T. ferrooxidans* reproduction by a bactericidal agent. The end products are sulphuric acid and ferric sulphate. Also, sulphuric acid is an important intermediate product. From the onset of pyrite oxidation, pH falls (acidity increases) quickly and then stabilises, typically at values around pH 2.5 - 3.0. The pH of stabilisation is normally determined by the optimal habitat requirement of the site-specific strain of bacteria.

Stable pH developed (2.5 - 3.0), and the products of sulphuric acid and ferric sulphate create conditions where the ferric iron ion itself can act as an oxidant. In the absence of ferric iron at pH 2.5 - 3.0, sulphuric acid will dissolve some heavy metal carbonate and oxide minerals, but has little reactive effect on heavy metal sulphides. However, the ferric iron ion is capable of dissolving many heavy metal sulphide minerals, including those of lead, copper, zinc and cadmium, by the general reaction:

$MS + nFe^{3+} \Leftrightarrow Mn^{+} + S + nFe^{2+}$ (7) *(Equation 3-7)*

where: MS = solid heavy metal sulphide; Fe^{3+} = aqueous ferric iron ion; Mn^{+} = aqueous heavy metal ion; S = sulphur and Fe^{2+} = aqueous ferrous iron ion.

It is by this process that significant amounts of heavy metals may be solubilised by AMD. In addition, many metallic elements are often present at trace levels within the minerals pyrite and pyrrhotite. Oxidation of these minerals can therefore release and mobilise these trace elements.

The chemical processes of AMD are directly related to metal transport and the formation of metallic mineral deposits via the natural generation and neutralisation of AMD (Mills, 1998).

Untreated (not neutralised) AMD creates two quite distinct environmental problems - the acidity from sulphuric acid (which is invariably a product by definition) and the heavy metal solubilisation caused by ferric iron (which may occur under the conditions described above). It is important that these two effects be recognised as separate, since their consequences to ecosystems are distinct, and because AMD generation and heavy metal transport are separate processes (Mills, 1998).

3.2 REACTION MECHANISMS

Fe^{3+} is the major pyrite oxidant in the acidic pH-region, while O_2 is expected to be the direct pyrite oxidant at neutral to alkaline pH (Stumm and Singer, 1970). Moses *et al.* (1987) demonstrated that Fe^{3+} may be a very effective oxidant at circum neutral pH. Pyrite oxidation over the pH-range 2 - 9 was favoured in the presence of Fe^{3+}, as opposed to dissolved O_2, and a low concentration of Fe^{3+} was very effective in oxidising pyrite. Luther (1987) explained that the faster rate of pyrite oxidation by Fe^{3+} was due to the fact that Fe^{3+} can bind chemically to the pyrite surface, whereas O_2 cannot.

4 BASIC CHEMISTRY OF AMD NEUTRALISATION

Neutral or alkaline mine drainage (NAMD) has alkalinity that equals or exceeds acidity, but can still have elevated concentrations of SO_4, Fe, Mn and other solutes. NAMD can originate as AMD that has been neutralised by a reaction with carbonate minerals, such as calcite and dolomite, or can form from rock that contains little pyrite. Dissolution of carbonate minerals produces alkalinity, which promotes the removal of Fe, Al and other metal ions from solution and neutralises acidity. However, neutralisation of AMD does not usually affect concentrations of SO_4 (Rose and Cravotta, 1999).

Acid Mine Drainage depletes the buffering ability of water by neutralising carbonate and bicarbonate ions to form carbonic acid (H_2CO_3) (Drever, 1997).

$$H^+ + CO_3^{2-} \Leftrightarrow HCO_3^- \quad \textit{(Equation 4-1)}$$

$$H^+ + HCO_3^- \Leftrightarrow H2CO_3 \quad \textit{(Equation 4-2)}$$

Once exposed to the Acid Mine Drainage, the affected carbonate buffering system is not able to control changes in pH as well. The buffering system is completely destroyed below a pH of 4.2 where all carbonate and bicarbonate ions have been converted to carbonic acid. The carbonic acid readily breaks down into water and carbon dioxide (AMD Chemistry, 1997).

$H_2CO_3 \Rightarrow H_2O + CO_2$ *(AMD Chemistry, 1997 and Todd and Reddick, 1997) (Equation 4-3)*

If the pH of AMD is increased, as would happen with contact with basic minerals such as calcite ($CaCO_3$) or dolomite ($CaMg(CO_3)_2$) or entry into a water system of higher pH (e.g. a saltwater fjord such as Howe Sound), then metallic ions such as Fe^{3+}, Cu^{2+}, Zn^{2+}, Pb^{2+} and As^{3+} will react to eventually form hydroxides as precipitates by the general reaction: -

$$Mn^+ + nOH^- \Leftrightarrow M(OH)_n \quad \textit{(Equation 4-4)}$$

where: OH^- = hydroxyl ion and $M(OH)_n$ = metal hydroxide.

This over-simplification represents chemical neutralisation as it occurs by human intervention rather than an accurate portrayal of natural occurrence, where the precipitation products are usually carbonates and sulphates and their hydrated and/or hydroxy-complex forms. In nature, acid-generating minerals such as pyrite often occur in close association with acid-neutralising minerals such as calcite ($CaCO_3$) and dolomite ($CaMg(CO_3)_2$), and acid produced from pyrite is neutralised *in situ* by these minerals. The sulphate most commonly formed is gypsum ($CaSO_4.2H_2O$), which is sparingly soluble in water, and which therefore contributes to elevated sulphate levels in ground- and surface waters (Shaw and Mills, 1998).

Factors that determine the neutralisation rate by carbonate and silicate minerals include pH, PCO_2, equilibrium conditions, temperature and the presence of “foreign” ions. Similar factors affect sulphide oxidation. Comparison of rates shows that sulphides react fastest, followed by carbonates and silicates (Sherlock *et al.,* 1995).

If water flowing into pyritic materials is alkaline or alkaline conditions can be maintained in the pyritic material, the acid-generating reactions may be inhibited so that little or no AMD forms (i.e. bacterial oxidation of Fe^{2+} is minimal). Alternatively, once AMD has

formed, its interaction with alkaline materials may neutralise the acidity and promote the removal of Fe, Al and other metals. Hence, water with high SO_4 and low Fe may be indicative of earlier AMD generation.

The carbonate minerals, calcite ($CaCO_3$) and dolomite ($CaMg(CO_3)_2$) are the main minerals providing alkalinity. Siderite ($FeCO_3$) is also a possible source, with qualifications discussed later. The carbonate minerals may occur as layers of limestone or dolostone in the overburden above coal, as cement in sandstone or shale, or as small veins cutting the rock. The initial reaction with an acid solution (using calcite as an example) is:

$$CaCO_3 + 2\,H^+ = Ca^{2+} + H_2CO_3(aq) \qquad \textit{(Equation 4-5)}$$

If a gas phase is present, the H_2CO_3 may partly decompose and exsolve into the gas phase, i.e.:

$$H_2CO_3(aq) = CO_2(g) + H_2O_{(l)} \qquad \textit{(Equation 4-6)}$$

Upon further neutralisation of AMD with carbonate to pH-values greater than 6.3, the product is bicarbonate (HCO_3^-):

$$CaCO_3 + H^+ = Ca^{2+} + HCO_3^- \qquad \textit{(Equation 4-7)}$$

In contrast with oxidation reactions, which are mainly significant under unsaturated conditions, carbonate dissolution and production of alkalinity are significant under both.

For a dilute water encountering limestone, Figure 86 indicates the approximate equilibrium concentration of HCO_3^- (alkalinity) in the aqueous phase as a function of the pH and P_{CO_2}, in the presence of calcite. Waters containing significant concentrations of other elements (Fe, Mg, SO_4) may deviate from the concentrations in this diagram. This diagram also indicates the approximate maximum amounts of dissolved alkalinity that may be carried into pyritic spoils by groundwater that has contacted carbonates, and the amounts of alkalinity that may be generated in systems such as anoxic limestone drains.

If one is concerned about the amount of calcite required to neutralise AMD of a given quality, as in the calculation of Neutralisation Potential (Sobek *et al.,* 1978) or the addition of alkaline materials to pyritic spoil in order to prevent AMD formation, Reactions 1.12 and 1.14 are also relevant. The amount of calcite required to neutralise a given amount of AMD depends on the behaviour of CO_2 during neutralisation and on the pH reached. If the AMD is to be neutralised to pH 6.3 or above (i.e. HCO_3^- is the main carbonate species produced) and no CO_2 is allowed to exsolve to the gas phase, then the reaction may be written (Cravotta *et al.,* 1990):

$FeS_2 + 4\ CaCO_3 + 3.75\ O_2 + 3.5\ H_2O = Fe(OH)_3 + 2\ SO_4^{2-} + 4\ Ca^{2+} + 4\ HCO_3$ (Eq. *4-8)*

Under these conditions, neutralisation of the products of oxidising 1 mole of pyrite requires 4 moles of $CaCO_3$, or 400 g of $CaCO_3$ to 64 g of pyritic sulphur, or 62.5 t of $CaCO_3$ per 1 000 t of material with 1% S as pyrite.

In contrast, if all CO_2 escapes to the gas phase and/or the AMD is only neutralised to about pH 5, then the reaction may be written:

$FeS_2 + 2\ CaCO_3 + 3.75\ O_2 + 1.5\ H_2O = Fe(OH)_3 + 2SO_4^{2-} + 2Ca^{2+} + 2CO_2$ *(Equation 4-16)*

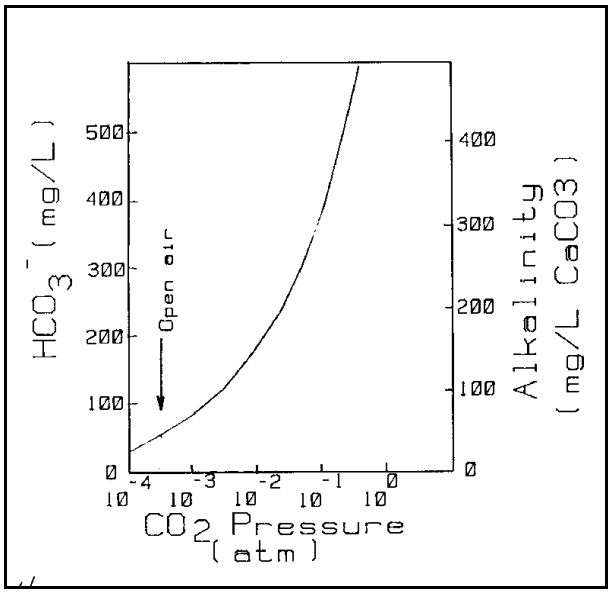

Figure 86. Equilibrium concentration of HCO_3- (alkalinity) from dissolution of calcite ($CaCO_3$) by pure water at various partial pressures of carbon dioxide (P_{CO2}) at 25°C. (calculated after Garrels and Christ, 1965).

Under these conditions, neutralisation of AMD generated by oxidation of 1 mole of pyrite requires 2 moles of $CaCO_3$, or 200 g of $CaCO_3$, or 31,25 t of $CaCO_3$ per 1 000 t containing 1% pyritic sulphur.

Most natural situations probably fall between these two extremes. Air within most strip mine spoil can contain significant amounts of CO_2 (Lusardi and Erickson, 1985 and Cravotta *et al.,* 1994), so that some CO_2 is clearly exsolving. If O_2 can get into the spoil to drive the pyrite oxidation reaction, then some CO_2 can escape into the open air. On the other hand, in order to provide detectable alkalinity in the effluent, some HCO_3^- must

be present. Thus, the theoretical amount of carbonate required to neutralise AMD generally falls intermediately between the two end member cases.

If the neutralising material is lime, composed mainly of CaO or $Ca(OH)_2$, then the neutralisation reaction is

$$Ca(OH)_2 + 2\,H^+ = Ca^{2+} + 2\,H_2O \qquad \textit{(Equation 4-17)}$$

5 CONCLUSIONS

Although, in principle, the formation of AMD by pyrite oxidation is simple, the preceding discussion indicates that the possible processes are many and complex. The water quality is clearly dependent on geologic and hydrologic conditions at a given site and thus careful field observation and laboratory analysis, combined with an understanding of possible processes, are necessary to draw conclusions for a particular location (Rose and Cravotta, 1999).

The rate of water quality deterioration in opencast pits is a function of the:

- Method and rate of mining.
- Method and rate of rehabilitation.
- State of weathering of the rock and coal.
- Acid and base potential of the rock and coal.
- Rate of water ingress from groundwater and surface water sources.
- Residence time of water within the pits.