A CONTRIBUTION, THROUGH RESEARCH, TO KNOWLEDGE IN THE FIELD OF MINE SAFETY AND HEALTH

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A thesis submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements of the degree of Doctor of Engineering.

Johannesburg, January 2019

CONTENTS

DECLARATION

I declare that the preparation of this thesis is my own unaided work and in this form has never been submitted for any degree or examination at any other university.

However, parts of Chapter 2.3 and 2.4 draw on research which was also submitted in my Ph.D. at the University of Newcastle upon Tyne in 1976 and was subsequently augmented with additional data and published after the award of the degree. In terms of the Senate Standing Orders this material is not submitted for examination but is included merely as a record of my early research and to contextualise my later work in coal excavation, leading to my final publication in this field in 1995.

Chapters 5.2 and 6.1 draw on research undertaken at the University of New South Wales, Australlia while working there as a lecturer and senior lecturer (1977 – 1984).

A great deal of the remainder of this thesis is based on research undertaken in collaboration with postgraduate students under my supervision and technical direction. Every effort has been made to acknowledge their contribution to the body of knowledge contained in this thesis.

Huw R. Phillips

Huw R Phillips

28th day of January, 2019

ABSTRACT

This thesis has been prepared for examination for the degree of Doctor of Engineering. As such it is a compendium of my research output from the early 1970s to date.

The thesis contains seven chapters, with the first chapter being an expanded version of my CV, explaining my career and research choices. The final chapter highlights the areas where my research output may have made a contribution to mining engineering knowledge. The remaining 5 chapters are based on research output and each covers a discrete topic, as follows

- Chapter 2 covers my early empirical research into fundamental aspects of rock excavation using both picks and disc cutters. This work formed part of my Ph.D. thesis and should not be considered in any current examination. However, it is so intricately interwoven with my post-doctoral work undertaken after October 1975, and which forms the remainder of the chapter, that its inclusion is warranted.
- Chapter 3 records the development of the Wits-EHAC Index used by coal mines in South Africa to evaluate the propensity for spontaneous combustion of their coal. In addition, three examples of applied, industry based projects dealing with spontaneous combustion are included.
- Chapter 4 describes my only venture into gold mining research and deals with the technical difficulties and costs involved in cooling ultra-deep mines. The research evolved from a simple consideration of the quality of insulation of chilled water pipes to the development of a model looking at the cooling requirements and costs for a notional, fully mechanised mine at a depth of 5 km.
- Chapter 5 deals with the monitoring of respirable dust levels, which has been one of my research interests since the 1970s. Research undertaken while at UNSW influenced the decision to adopt gravimetric sampling in New South Wales. Research while at COMRO initiated a similar debate in South Africa. More recent research has yielded a much better understanding of respirable dust patterns in room-and-pillar coal mining.
- Chapter 6 deals with my 40 years of research into coal seam methane, its accumulation in working areas, means of diluting and dispersing it and the prevention of methane and coal dust explosions. It also contains a brief description of my committee work promoting research into this topic.

There are three appendices to this thesis. Appendix A is my CV, while Appendix B contains many of my publications and Appendix C contains relevant publications, theses and dissertations, which could otherwise prove difficult to find.

DEDICATION

This thesis is dedicated to Professor Emeritus Frank F. Roxborough, my lecturer, research supervisor, Head of School and friend for very nearly 50 years. Thank you for being such an influence on my life.

ACKNOWLEDGEMENTS

I wish to thank

- my wife Beatrix for her patience and understanding during the past year. Many days have been spent in front of a computer instead of spending them together, enjoying normal retirement pursuits.

- Professor Ian Jandrell, Dean of the Faculty of Engineering and the Built Environment, for his encouragement to undertake the preparation of this thesis and for his role as my mentor.

- the more than 40 Master's postgraduates and 22 Ph.D. students I have had the privilege to supervise over the past 40 years. At the University of New South Wales I had the luxury of working in the laboratory and at the mines alongside my research students. At the University of the Witwatersrand other duties precluded such hands-on research. However, I believe I made a significant contribution to the research of each and every postgraduate I supervised by maintaining close contact and through the technical direction of their work. I have never had an academically poor student, the vast majority have been very good and a handful have been truly exceptional. Without their contribution and hard work this thesis would be considerably thinner!

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CHAPTER 1 A PERSONAL PROLOGUE

This chapter introduces the body of work submitted in this thesis. It also outlines the development of my involvement in mining and the factors that influenced my choice of research interests. In fact, its main purpose is to flesh out my curriculum vitae which forms Appendix 1 of this thesis.

1.1 My Early Career in the United Kingdom

The earliest and perhaps the most significant influence was the fact that I grew up in a coal mining village within sight of the headgear of Tower #4 Colliery. After my early education in the village primary school, I advanced to the Aberdare County Grammar School for Boys where, after seven years of study, I obtained my A Levels. During this time my parents and teachers impressed on me that education was the route out of mining and so my choice of discipline to study at the University of Bristol was Electrical Engineering.

During the second of my three years of university study an event occurred that significantly altered my career plans. On October 21st, 1966, I was home from university attending a family celebration when a tip of colliery waste slid down a steep mountainside and overwhelmed the primary school in the adjacent village of Aberfan.

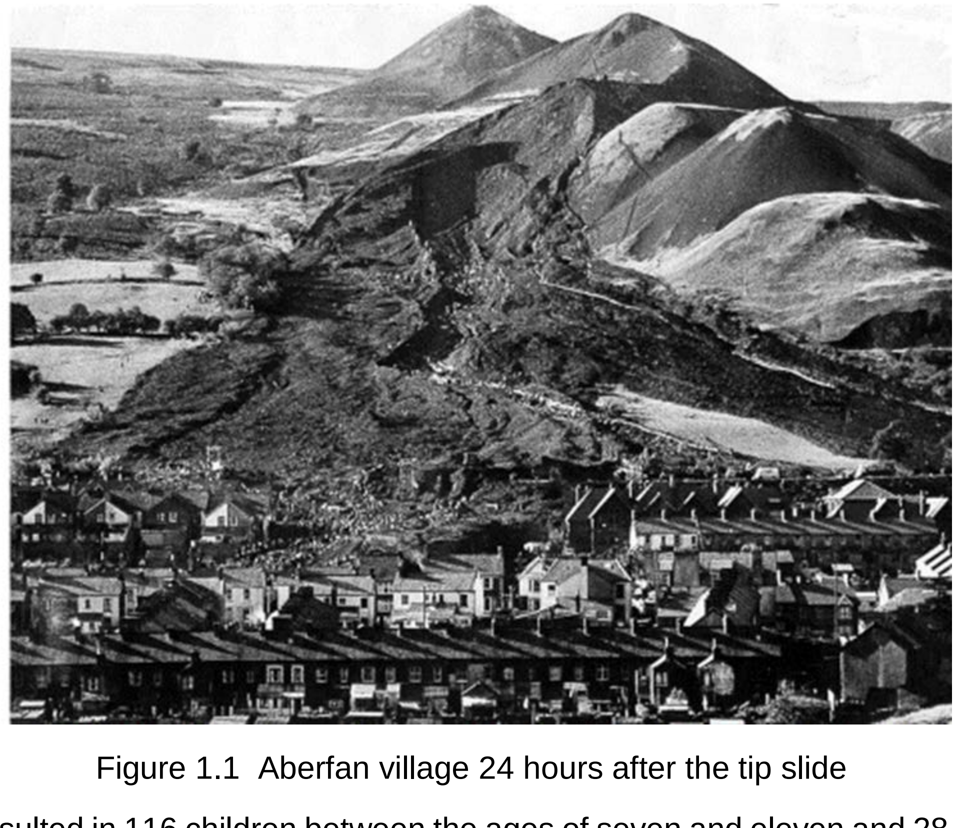


Figure 1.1 Aberfan village 24 hours after the tip slide

This resulted in 116 children between the ages of seven and eleven and 28 adults being fatally buried alive. That afternoon and evening, as a volunteer worker in the attempted rescue operation, I gained first-hand experience of the devastation mining can wreak on mining communities, but this event also sparked my interest in mining as a career (Gooding, 2010). The following year, as a final year student, I chose as my topic for a compulsory technical report “The development of electrical engineering in the mining industry” (Phillips,1968).

Early in 1968, when applying to various companies for employment as a graduate-in-training the best offer, both in terms of remuneration and practical training, came from the National Coal Board. Two weeks after the twin events of my 21st birthday and my graduation, I started work as a Directed Practical Trainee (DPT) in the Number 4 area of the South Wales Division of the NCB. This programme, which was two years in duration, prepared participants to obtain their statutory qualification. In my case this was the Electrical Engineer’s Certificate of Competency which was issued in September 1970. Under normal circumstances I would then have been appointed at a mine as a Deputy Electrical Engineer. However, by this stage I was more interested in the way coal mines were changing due to mechanisation. In the 1960s many coal mines in South Wales were still undercutting, drilling and blasting the longwall faces and hand-loading the broken coal, with face supplies still being delivered using ponies. At this stage I was fortunate in being awarded an NCB scholarship to undertake an M.Sc. in Mining Engineering at the University of Newcastle upon Tyne. There the Department of Mining Engineering had a 12-month programme open to engineering and geology graduates wishing to convert their first degrees to a mining qualification. In addition to a quite strenuous programme of coursework, a research report was required. Since most of my time as a DPT had been spent working in thin-seam mines utilizing coal ploughs, I chose to research the digital simulation of ploughs (Phillips, 1971). This work was supervised by Dr Frank Roxborough, the Reader in Mining Engineering, who was to have a massive influence on my life.

On completion of the M.Sc. in September 1971, as is typical of the vagaries of large companies, I was not sent to a mine with ploughs but was appointed to Markham Colliery as Deputy Unit Engineer with responsibility for the ambitious programme of installing three longwall shearer faces within a single year. This was a mine with no previous experience of power loading equipment or longwall chocks and, together with the rest of the workforce, I was on a steep learning curve and enjoying the challenges.

However, towards the end of 1972, I was approached by Frank Roxborough to return to the University of Newcastle upon Tyne as a Senior Research Associate managing a rock cutting research programme funded by the Transport and Road Research Laboratory, Department of Transport, U.K. and directed by Roxborough. This was a great opportunity to undertake well-funded research in an excellent laboratory backed by extensive workshop facilities. This research led to two major reports (Roxborough and Phillips,1975a), (Phillips,1975b) and my Ph.D. (Phillips, 1975a). In addition, Roxborough and I, sometimes with other co-authors, also published some 8 papers based on the nearly five years of research (1973 to 1977).

In May 1975 Roxborough left the United Kingdom to take up the Chair of Mining Engineering at the University of New South Wales (UNSW). Professor ELJ Potts, the Head of the Department of Mining Engineering, then took on the responsibility for rock cutting research but it slowly lost impetus. Early in 1977 Professor Roxborough again contacted me to ask if I would be interested in applying for a vacant lectureship at UNSW, where he was trying to enhance the research profile of the School of Mining Engineering. My application was successful and I relocated to Sydney in October 1977.

1.2 My Research at the University of New South Wales (Oct.1977 - Dec.1984)

During the 7 years I spent at UNSW my research was concentrated in 5 areas, as summarised in Table 1

While the "Fundamental studies of rock cutting" project was a continuation of the work Professor Roxborough and I were involved with at the University of Newcastle upon Tyne, the work on seam gas content and respirable dust were my new areas of research activity. Because of this Smits, Savidis and Price were jointly supervised by Roxborough and myself, while for Dawood and Hayes I was the sole supervisor.

Table 1.1 Summary of my research activity at UNSW

	Project	Source of Funding	Supervision of Postgraduates
1	Fundamental studies of rock cutting	ARGC	Smits (PhD, 1980) Savidis (PhD, 1982)
2	Some design aspects of longwall shearers using scale models	NERDDC	Price (PhD, 1985)
3	Respirable dust generated by longwall shearers	NERDDC	Dawood (PhD,1984)
4	Coal seam gas content, sorption characteristics and permeability	ARGC KCC	Hayes (MSc, 1983)
5	Introduction of gravimetric monitoring of respirable dust	JCB	

ARGC = Australian Research Grants Committee
NERDDC = National Energy Research, Development and Demonstration Council
KCC = Kembla Coal and Coke (Pty) (Ltd)
JCB = Joint Coal Board of NSW

In 1979 a large multi-year research contract aimed at improving longwall shearer performance was awarded to Professor Roxborough and myself. It quickly became obvious that the workload involved was beyond the commitment full-time academics could devote to the project. While at Newcastle upon Tyne I had supervised the M.Sc. of Mr D.L. Price, an Australian post-graduate student, and recruited him to manage the day-to-day operations. Together we completed the design of a $^1/_8$ scale longwall shearer, operating on an armoured face conveyor and cutting a 6 metre long face cast from small coal and cement. Figures 1.2 and 1.3 show the model face and $^1/_8$ scale longwall shearer.

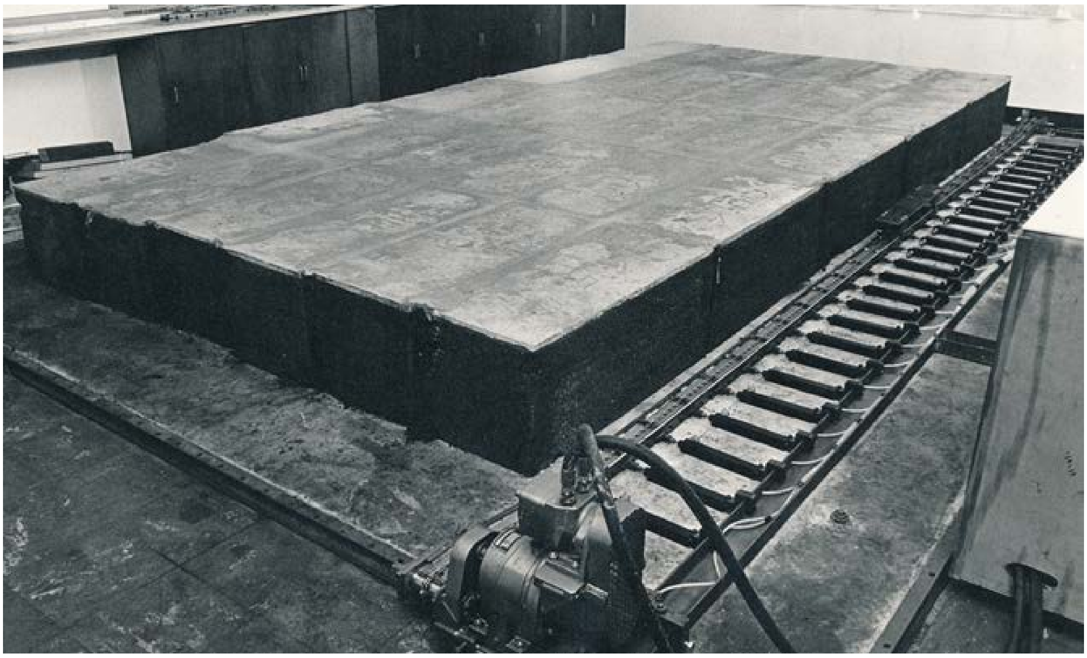


Figure 1.2 View of the $^{1}/_{8}$ scale face, pushing rams and conveyor

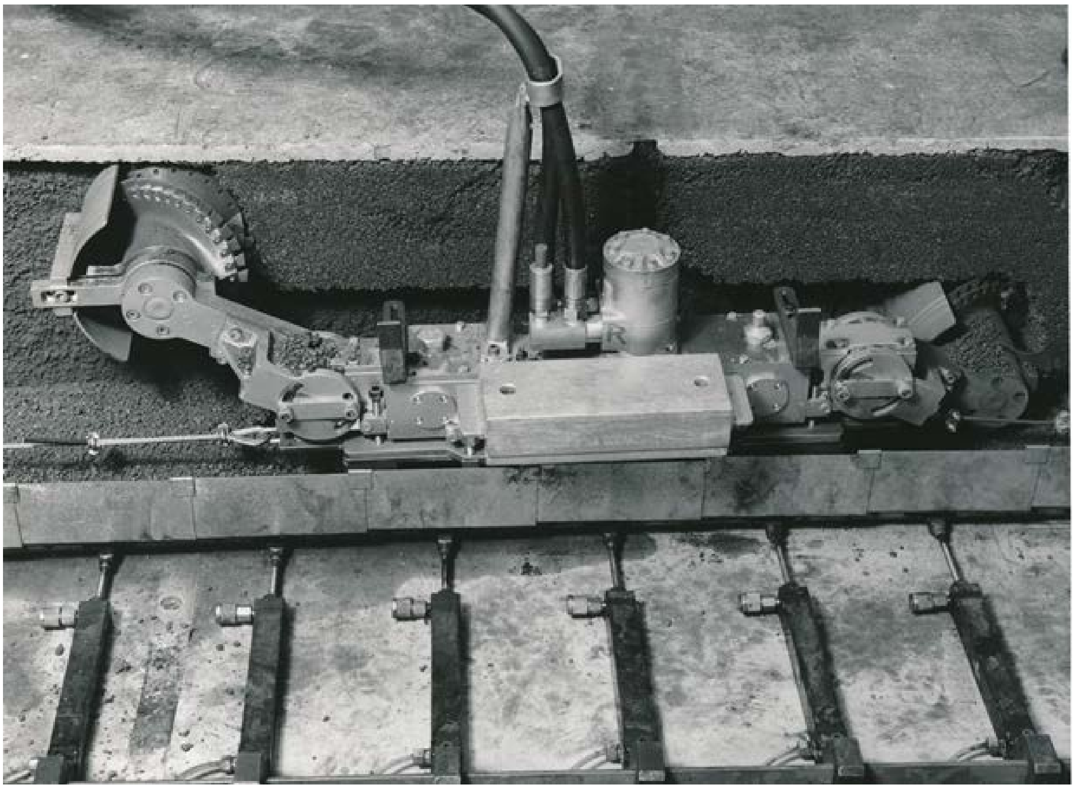


Figure 1.3 The $^{1}/_{8}$ scale double drum shearer cutting

With a competent project manager in place, my contribution to the research into the loading characteristics of different drum designs was minimal. The one area of personal research was the performance of armoured face conveyors. It was obvious from studying the performance of longwalls in the early 1980s, in New

South Wales, that machines, when deployed in the thicker seams, could cut and load more coal than face conveyors could accommodate. A mathematical model was developed, together with a set of design charts. This work demonstrated conclusively that increasing conveyor speed was ineffective and that the key to better longwall performance lay with increasing the conveyor width (Phillips and Drummond, 1980).

The introduction of highly productive mechanised longwall coalfaces was also revealing problems in diluting both the methane released during coal production and the large quantities of respirable dust generated. I already had an interest in quantifying seam gas content because of the severe outburst problems being experienced at the new Westcliff Colliery, in the southern coalfield of New South Wales. Similar problems were being experienced in Queensland and the risk of fatalities and the premature closure of certain mines was such that the “Australian Government Advisory Committee on Gas Related Problems in Coal Mining” was established and I was invited to become a member in 1980. In addition, the problem of adequately diluting the methane released during coal production required a better understanding of the relationship between seam gas content, permeability and sorption rates under realistic triaxial stress conditions and research was undertaken in this area (Lingard, Phillips & Doig, 1982; Lingard, Phillips & Doig, 1983 and Lingard, Doig & Phillips, 1984).

Up until the introduction of mechanised longwalls the underground coal mines in New South Wales had an exemplary record in combating pneumoconiosis, together with other related respiratory diseases. This was in stark contrast to the scourge of pneumoconiosis amongst retired colliers I had witnessed as a child growing up in South Wales. There was a fear, however, that exposure to respirable dust was increasing in NSW and that monitoring of the situation using particle counting techniques was inadequate. As a member of the Joint Coal Board’s Standing Committee on Dust Control I was aware of these concerns and was encouraged to research the possible introduction of gravimetric sampling (Phillips, 1984a and Phillips, 1984b). On the basis of my research activities, I was promoted to the rank of Senior Lecturer effective from 1st January 1981.

After three and a half years at UNSW I was eligible to apply for 6 months of study leave and my application to spend this time at the Chamber of Mines Research Organisation (COMRO) in Johannesburg was approved. The South African coal industry had been slow to mechanize due primarily to the availability of inexpensive labour. However, continuous miners were introduced in the mid-1970s but experienced difficulties in cutting coal seams considerably harder than those in either the United States or Australia (Roxborough et al, 1981). During my time at COMRO I completed two laboratory studies on the cutting of block samples from the No 4 seam in the Witbank/Highveld coalfield (Grant, Pieterse & Phillips, 1981) and (Phillips et al, 1983). In addition, I undertook an investigation into dust suppression techniques for continuous miners, which also involved the first serious use of gravimetric sampling in the South African coal industry (Phillips et al, 1981).

Because of staff shortages at the Department of Mining Engineering at the University of the Witwatersrand (Wits), the Chamber of Mines was asked for assistance and, for the duration of the second semester 1981, I was seconded for 6 hours each week to teach the course in Excavation Engineering. I returned to UNSW in February 1982 in time for my teaching duties in first semester.

In July 1983 I was approached and invited to apply for the vacant Chamber of Mines Chair of Mining Engineering at Wits. Following an interview in December 1983 I was offered the post. However, while in Johannesburg for the interview I met with the Gold Producers Committee of the Chamber of Mines. This powerful group of individuals stressed the need to build capacity at Wits as it was becoming more difficult to source and retain mining engineering graduates from overseas. The only disagreement I had with them was their view that the Department of Mining Engineering at Wits should only teach, as research was the preserve of the Chamber of Mines Research Organisation. Even then I knew that it would be impossible to attract and promote quality staff if they were discouraged from research activities in parallel with their teaching duties.

After much thought I accepted the position and I arrived in Johannesburg in January 1985.

1.3 My Years at The University of the Witwatersrand (1985 - 2015)

At the start of 1985 the Department of Mining Engineering was small and for some time there had been as few as 4 academic staff. When I arrived the staff consisted of Professor Alex Budavari, the Head of Department, Professor Danie Krige (on a 50% appointment) and four Senior Lecturers. During 1985 I spent as much time as possible visiting mines, speaking at Mine Managers Association events and discussing industry requirement with senior executives.

In January 1986 I was appointed Head of Department and was able to action some of my plans. The Department became a School in 2000 and its trajectory over the years has been well documented (Phillips, 1988), (Phillips, 1994), (Budavari and Phillips, 1996), (Karmis et al, 2009) and (Musingwini, Cruise and Phillips, 2013)

Table 1.2 Development of the School of Mining Engineering at Wits

	1985	2009
Undergraduate registrations	74	473
Postgraduate registrations	49	329
Number of academic staff	6	15
Research income	R106 000	*Several $R10^6$
Mining industry funding	R51 000	R3 635 000

*In addition to direct funding, collaborative research with industry provided support in kind

At the end of 2009 I stepped down as Head of the School of Mining Engineering after completing 24 years in that position. The changes that took place between 1985 and 2009 are summarised in Table 1.2.

Discussions with the Government Mining Engineer (GME) in 1985 and 1986 were key to the development of research within the Department as they revealed the dreadful state of mine safety, with 710 mining fatalities in 1985 (540 in gold and 93 in coal mining). Historical data regarding coal mine safety will be discussed further in Chapter 6 but the outcome of the discussions with the GME and with the Research Advisor of the Chamber of Mines, Professor M D G Salamon, was that Wits Mining should be involved in rock engineering and health and safety research, with an emphasis on coal mining. This was because the Chamber of Mines devoted enormous resources to solving the problems of the gold industry i.e. preventing rock bursts and falls of ground, mine cooling to provide acceptable working conditions and the difficult task of mechanisation in hard rock, tabular mines. Coal on the other hand was very much the Cinderella of the mining industry, attracting a very small proportion of COMRO's budget. In March 1985 I was invited by the GME to join the Explosion Hazards Advisory Committee, a sub-committee of the Coal Mining Research Controlling Council (CMRCC), a body set up after the Coalbrook disaster when, in January 1960, 437 of the underground workforce at Coalbrook Colliery were killed by the collapse of pillars over a wide area (Phillips, 1999). Through research contracts with the CMRCC and COMRO approximately R100 000 was raised in 1985 to begin re-building the Department's research profile. This was sorely needed since at the start of 1985 there were only 49 postgraduates, registered as follows :- 6 Ph.D., 14 M.Sc. and 29 Graduate Diploma students. Only three of these postgraduate students were full time in the Department.

It took time and effort to both increase undergraduate numbers and to establish a really viable postgraduate activity in the School but this was assisted by the appointment of additional staff. As part of this process the mining industry honoured its commitment to support the Department and, in 1987, agreed to fully fund the appointment of four Senior Lecturer positions and an Administrative Manager. In 2000 this commitment to supporting staff positions passed over to the Minerals Education Trust Fund.

However, this thesis is concerned with demonstrating my contribution to mining engineering knowledge through research. While at the University of Newcastle upon Tyne I was able to undertake hands-on research. During my time at the University of New South Wales it was a little more difficult but still possible to find time for laboratory and field research, which was usually undertaken together with one of the postgraduate students. Once appointed as a Head of School at Wits, I recognised that it would be very difficult to be involved at the same level and that my contribution would be as a supervisor of research students. Over the period 1985- 2015, I successfully supervised to completion 41 Master's dissertations and 19 Ph.D. students. In addition, I have acted as mentor for the preparation of two successful D.Eng. submissions.

My research effort at Wits can conveniently be divided into five areas :-

1. Mechanised mining systems, which can be viewed as a continuation of my early work on the simulation of coal ploughs and the design of pick and disc cutting arrays. During my time at Wits the research was mainly concerned with attempts to replace conventional narrow reef mining with trackless mining. However, this area of research does not form part of this thesis.

2. The early work on spontaneous combustion involved the development of a laboratory for determining crossing-point temperatures, as well as an adiabatic calorimeter. Using this equipment a standard test, the Wits-EHAC test, was developed and there is still a requirement for all underground coal mines in South Africa to use this test to report the propensity of their coal to spontaneously combust. The knowledge gained from the early work was the basis for solving a really massive environmental problem at the Grootegeluk Open Pit coal mine.

3. As computing power became more readily available on the gold mines, two M.Sc. students worked on developing software, based largely on previous research by COMRO, that would provide mine ventilation engineers with the tools to design mine specific cooling strategies. By the 1980s deep level gold mines were maintaining acceptable working temperatures using foam insulated chilled water reticulation systems. However, the Kinross gold mine fire in September 1986 in which 177 workers died from inhaling toxic fumes (hydrogen cyanide) when polyurethane insulation caught fire resulted in the need to re-evaluate heat loads and cooling strategies in deep mines.

4. As described in Section 1.2 I had an interest in monitoring and controlling respirable dust in coal mines from as early as 1980. When a young engineer working at CSIR Miningtek applied to register for a Ph.D. in this field I was delighted to resume my work in this field and to supervise his thesis (Belle, 2004). In addition to the thesis, Belle and I have jointly published 11 papers.

5. My research into methane content and release rates, which started at UNSW, resumed almost immediately after I arrived at Wits. In the same general area I became active in administering research in this field, as a member of the Explosion Hazard Advisory Committee (EHAC) and subsequently the SIMRAC Collieries Environmental Engineering Advisory Group (COLEEAG) and as Chair of the Special Interest Group on Explosion Hazards (SIGEH). As the research moved forward the focus changed to the prevention of methane ignitions becoming coal dust explosions by active suppression systems and the prevention of coal dust explosions from propagating throughout a mine by the design of the bagged

stone dust barrier. This area of research has probably been the most rewarding of my career.

The five research areas discussed above and the number of postgraduates supervised are summarised in Table 1.3

Table 1.3 Areas of research activity and postgraduate supervision (1985 – 2015)

	Research Area	Postgraduates Supervised
1	Mechanised mining systems	11 M.Sc.,1 D.Eng.
2	Spontaneous combustion of coal	3 M.Sc., 4 Ph.D.
3	Heat loads and cooling in deep mines	4 M.Sc., 1Ph.D.
4	Monitoring of respirable dust in coal mines	1 Ph.D.
5	Prevention of methane and coal dust explosions	8 M.Sc., 2 Ph.D.

Following my departure from the University of the Witwatersrand, most of my time has been devoted to consultancy in the area of explosion prevention. It has involved audits of coal mines in South Africa and Australia to ensure compliance with Codes of Practice and Principal Hazard Management Plans. Research activity has been in two areas. Firstly, in the implementation of active barriers to protect conveyor roadways (Spaeth et al, 2017) and awareness of explosion prevention (du Plessis et al, 2017). Other work has involved developing the correct procedures for reporting Green House Gas emissions from working coal mines.

1.4 Structure of this thesis

The remainder of this thesis is presented in five chapters each dealing with one aspect of my research output, together with a chapter of concluding remarks. The research chapters are listed in Table 1.4 below

Table 1.4 Chapters based on research output

Chapter	Title
2	Mechanised excavation of coal and rock
3	Spontaneous combustion research
4	Heat loads and cooling of deep gold mines
5	Measurement and control of respirable dust levels
6	Prevention of methane and coal dust explosions.

1.5 The central theme of this thesis

As detailed in Table 1.4, the research described in this thesis is presented in 5 Chapters. While the titles of Chapters 3 to 6 clearly indicate the research is related to mine health and safety, at first sight Chapter 2 appears to be an anomaly. In fact, it describes fundamental research related to the mechanical breakage of rock. This was undertaken because of the extent of proposed tunnelling activity in the United Kingdom during the 1970s and 1980s.

1.5.1 Research into mechanised excavation

From the earliest days of tunnelling, it had a safety record at least as bad as that of coal mining. One notable example of a tunnelling accident nearly ended the life of the young Isambard Kingdom Brunel while he was working on the first tunnel under the river Thames in London. “On 12 January 1828, a second flood proved even more calamitous than the first. Six men were killed and Brunel, who had been working on one of the frames with two of the men who drowned, was saved only by sheer luck. His leg was badly crushed and he was knocked half unconscious, but the wave of flood water swept him towards an exit shaft and into the arms of his assistant Beamish, who was rushing to the scene” Anon, 2018. Despite this personal experience of the dangers of tunnelling Brunel continued to use tunnels while constructing the Great Western Railway from London to Bristol. His Box Tunnel was less than 3 km long but reportedly cost more than 100 lives in its construction between 1838 and 1841.

During the 15 months I spent at Markham Colliery (see Chapter 1.1), I was well aware that coal face safety improved significantly once the three working longwall faces were equipped with self-advancing powered supports. Although at that time I was unaware of the fatality statistics, Professor Horst Wagner referred to them in his Presidential Address to the South African Institute of Mining and Metallurgy “I believe that, in the long term, substantial increases in the rate of face advance will result only from changes in mining technology. In this connection, I refer to experiences in the British coal-mining industry. In 1955 more than 140 persons were killed by falls of ground at the coal face in British collieries. This was at a time when face operations were largely labour-intensive hand-filling systems and strata control was achieved by hand-set steel props and bars, and by hand-erected pack-and-waste systems. The introduction of hydraulic props, hydraulic chocks, and ultimately fully mechanized self-advancing hydraulic support systems reduced this number to 3 in 1983” (Wagner, 1986).

My rock cutting research was primarily intended to improve the rock cutting efficiency of tool arrays. At the time I didn’t think too deeply about it but I did realise that encouraging the use of Tunnel Boring Machines (TBMs) would not only increase the speed of tunnelling and reduce costs but would also provide the same degree of safety as coal miners enjoyed once self-advancing chocks and shields were introduced.

1.5.2 Chapters 3 to 6.

The spontaneous combustion of coal (Chapter 3) is a phenomenon that has been extensively researched. When I was approached by the Coal Mining Research Controlling Council to develop a laboratory test to determine the susceptibility of South African coal seams to spontaneously combust, it was because of the number of spontaneous heatings being experienced in underground coal mines in South Africa. The development of continuous sampling and analysis of the underground atmosphere has reduced the significance of this problem through early detection of heatings but the results of undetected spontaneous combustion in gassy underground mines can still be catastrophic (Windridge, et al, 1995). The more recent work reported in Chapter 3 deals with spontaneous combustion when surface mines extract the coal pillars left by earlier underground mining. Under these circumstances uncontrolled spontaneous combustion results in the exposure of workers in the open-cut to unhealthy atmospheric conditions and also leads to a serious safety issue when hot blast holes are charged ready for detonation.

Chapter 4 describes my one excursion into gold mining research. As a member of the Board of the Deepmine Collaborative Research Programme, which investigated technical, social and safety aspects of extending the working depths of gold mines from 3000m to 5000m below surface, I saw the possibility of researching the technologies and costs involved in cooling the workings. Research conducted almost continuously since the 1920s had investigated the effects of heat exhaustion and heat stroke, together with the decline in productivity at elevated temperatures. Refrigeration techniques were already well established but the challenge in ultra-deep mining was to control costs by preventing heat from the surrounding strata entering the workings through insulating intake airways and backfilling stopes. It was envisaged that ultra-deep mining would be highly mechanised and, since it had been shown conclusively that a worker’s concentration decreases and accident rates increase in seriously hot conditions, this chapter concludes with a technical and economic evaluation of cooling mechanised workings at depth.

Chapter 5 is based on my research into the measurement of respirable coal dust. In the late 1970s I was awarded a research contract to investigate the introduction of gravimetric respirable dust sampling in the coal industry of New South Wales. This research interest continued when I moved to South Africa and was motivated by the scourge of pneumoconiosis I had witnessed as a child growing up in a mining village in South Wales in the 1950s and 1960s. The connection between this research interest and the desire to contribute to an understanding of where respirable dust is created, how it can be minimized and the best means and location for monitoring is obvious. Since monitoring is a fundamental precursor to the control of respirable dust and ensuring exposure targets are met, Chapter 5 is all about limiting respirable dust exposure and maintaining a healthy environment for underground coal miners.

Finally, Chapter 6 contains a record of the research that has given me the greatest satisfaction. In the 1980s and early 1990s colliery explosions in South Africa occurred, on average, once each year and, between 1984 and 1993, 130 coal miners were killed in explosions. During this time I was a member of the Explosion Hazard Advisory Committee and the Kloppersbos Research Centre Advisory Committee. In addition to these administrative roles, I supervised the research of 7 M.Sc. and 3 Ph.D. candidates working in the field of explosion prevention. This culminated in the development of the bagged stone dust barrier, now used extensively in South Africa, Australia and several other countries. Since 1994 there has been no major coal mine explosion in South Africa and the development of the Kloppersbos facility and the research conducted there must have contributed to this success.

1.5.3 Conclusion

It is obvious from Sections 1.5.1 and 1.5.2 above that the "Central Theme" of the research presented in this thesis is work undertaken to improve health and safety in the underground environment. This is a legitimate field of study for mining engineers, particularly in South Africa, since the health and safety record of the country was, and still is, unacceptable. Although the connection between Chapter 2 and the Central Theme is a little tenuous, it does exist, while Chapters 3 to 6 are solidly in the area of health and safety research.

1.6 Personal contribution to the research in this thesis

The Senate Standing Orders require applicants for Senior Doctorates to present a "copy of their published work, together with a document co-ordinating the material and providing a framework on which his or her application is to be assessed". In addition to this the Faculty of Engineering and the Built Environment has its own requirements set out in Rule 5.2 dealing with the degree of Doctor of Engineering. Eight Exit Level Outcomes appropriate for this level of an engineering degree are listed and the Associated Assessment Criteria require that the work must be "a record of engineering development ….. carried out under the technical direction of the candidate". It could be argued that the supervisor of any research undertaken by a postgraduate student exercises "technical direction". While it is relatively easy to estimate the contribution of co-authors to a single publication (this has been done in the list of publications at the end of my CV in Appendix A), it is impossible to distinguish, in percentages, the relative contributions of student and supervisor to the outcome of three years or more of close collaboration.

Since my first M.Sc. student graduated in 1983, I have supervised a further 41 Master's dissertations and 22 doctoral theses. However, the theme of this thesis deals almost exclusively with health and safety issues and so reference has been restricted to my technical direction of 9 Ph.D. theses and 12 M.Sc. dissertations. Other postgraduates have worked in a variety of field ranging from blasting, surveying and mining methods to mineral economics. In many cases I felt that I was supervising the thesis or dissertation rather than the research. I would not consider submitting any of the work conducted by these students in support of my current application.

Tables 1.5 and 1.6 list those Ph.D. theses and M.Sc. dissertations to which reference has been made in the course of this thesis. In each case I have attempted to give an honest appraisal of the "technical direction" I provided.

Two of the Ph.D. candidates were full-time postgraduate students and 4 were lecturers/senior lecturers in the School. I had meetings with these 6 candidates at least once a week. Du Plessis and Belle were experienced researchers working for CSIR Miningtek and registered as part-time candidates. Because of their research maturity meetings were at approximately monthly intervals.

Adamski was the pit superintendent at the Grootegeluk open-pit coal mine, excavating more than 50 million tons of material a year in a truck and shovel operation working on 15 benches. Because of his distance from the University we mainly corresponded by telephone and email, with infrequent face to face meetings. However, it was always a pleasure to hold our meetings, to review the results from results obtained at the ISCOR laboratories and in the open-pit and be involved in a spontaneous combustion experiment involving 20 million tons of reactive material!

Table 1.5 Contribution through technical direction of 9 Ph.D. studies

Name	Contribution	Name	Contribution
Wade, 1988	a,c,d,e,f,h	Du Plessis, 2000	(partial c),d,e,f,h
Stripp, 1989	b,c,d,e,f,h	Rawlins, 2002	c,d,e,f,h
Landman, 1992	b,c,d,e,f,h	Belle, 2004	d,e,f,h
Eroglu, 1992	c,d,e,f,g,h	Adamski, 2004	c,d,e,f,g
Gouws, 1993	a,d,e,f,h		

a = design and supervision of laboratory equipment construction in the School's workshop
b = design and supervision of construction of field equipment
c = assistance with design of programme of experiments
d = discussion as to the significance of research results
e = assistance with structuring the thesis
f = reading and commenting on draft chapters
g = correction of language errors
h = collaboration on the preparation of papers

Table 1.6 Contribution through technical direction of 12 M.Sc. studies

Name	**Contribution**	**Name**	**Contribution**
Hayes, 1983	a,b,c,d,e.f	Cook, P., 1992	b,d,e,f
Von Glehn,1987	d,e,f	Rawlins, 1998	b,d,e,f,g,h
Gouws, 1987	b,d,e,f	Cook, A., 1999	b,d,e,f
Billenkamp, 1988	a,b,c,d,e,f	Van Dijk, 2000	d,e,f
Flint, 1990	e,f	Genc, 2000	d,e,f,
Searra, 1990	a,b,d,e,f	Uludag, 2001	b,d,e,f,g,h

a = design and supervision of laboratory equipment construction in the School's workshop
b = assistance with design of programme of experiments
c = physical assistance with laboratory work
d = discussion as to the significance of research results
e = assistance with structuring the thesis
f = reading and commenting on draft chapters
g = correction of language errors
h = collaboration on the preparation of papers

I have felt uneasy with compiling the tables detailing my contribution. It has always been a pleasure supervising postgraduates and giving technical direction to their work. The ideal relationship is based on the experience of the supervisor and the enthusiasm of the student. With doctoral students you develop a particularly strong relationship and a friendship lasting for many years after they have graduated. If any of my past postgraduate students feel I have been unfair and not given them sufficient credit, then I apologise unreservedly.

CHAPTER 2 THE MECHANISED EXCAVATION OF COAL AND ROCK

The term "mechanised excavation" is generally used to describe the use of machines for the bulk excavation of rock and minerals from their natural in-situ condition. The principal feature of such a mining machine is the cutting head, which in most cases uses a number of tools disposed in an array. The two most commonly used tools are picks, which are widely used in the coal mining industry, and disc cutters for the excavation of harder and more abrasive rock.

2.1 Brief historical review

The first concerted efforts to mechanise the excavation process occurred during the 19th century in underground coal mining. This is not surprising with so many circumstances favourable to the development of mechanisation. Coal mining was already a big industry and was proving to be a dangerous occupation for the large numbers of men working at the coal face. Coal is a comparatively weak and non-abrasive material occurring in seams of regular thickness and could therefore be extracted by the application of relatively low forces using machines made of wrought and cast iron and which could perform a simple repetitive task. The machines in use from roughly 1850 to 1950 were not designed for bulk excavation but were used to excavate slots in the coalface and provided a "second free face" resulting in enhanced blasting efficiency. This improvement allowed the use of lower power explosives and this significantly reduced the risk of methane ignitions during blasting.

The availability of power in suitable form was crucial to the development of mechanical coal mining. Water (in the form of a water wheel) was the first recorded means for supplying power to a colliery and this was as early as 1680. The first mention of steam power relates to its use at a Durham colliery (England) in 1705. Compressed air, which proved to be the touchstone for the development of coal face mechanisation was first introduced in a Scottish colliery in 1850. The introduction of electrical power underground started in 1882 at the Trafalgar Colliery located in the Forest of Dean (United Kingdom) coalfield (Phillips, 1968). However, electricity was only used in intake airways well away from the coalface because of fears that the sparking of switchgear could ignite methane explosions. Low voltage signalling equipment powered by wet cell batteries was considered safe and was permitted in haulage roadways, even in the face areas. Following the Senghenydd Colliery disaster in October 1913, in which 439 men were killed in an explosion caused by low voltage sparking, there was a demand to remove all electrical apparatus from underground. Research into making electrical equipment safe resulted in the design of flameproof apparatus, which had become mandatory in German coal mines from 1912 and from 1922 in the United Kingdom. The design requirements to make low voltage apparatus intrinsically safe followed shortly afterwards and these designs were based primarily on research undertaken at the University of Sheffield (Phillips, 1968).

Between 1900 and 1939 many attempts were made to develop machines which could eliminate the cyclic nature of drilling and blasting operations. It was the Second World War which added urgency to these attempts. The war resulted in an unprecedented increase in the demand for coal to be used in power generation, transport and the manufacture of steel. This was at a time when manpower for mining activities was severely limited due to recruitment into the armed forces. Solutions were sought in Germany, the United States and United Kingdom and although successful, in reality they were not brought to fruition until the post-war period. The coal plough for use on longwall coalfaces was developed in Germany, an early example of which is shown in Figure 2.1

Figure 2.1 An early coal plough with friction props and bars

In the United States the need was for machines to be used in room and pillar mining. An early example was installed in a Consolidated Coal and Coke Company mine in 1943 but its success was limited. However, it came to the attention of Joy Mining Machinery, a company founded in 1920 by Joseph Joy for the manufacture of coal loading equipment. After trials in early 1948 of a prototype, the JCM 2, the first successful ripper type continuous miner, the Joy JCM3, was deployed in December 1948 and subsequently over 200 machines of this model were manufactured. Figure 2.2 is a photograph of an early example of the JCM3 prior to delivery.

In the United Kingdom attempts were made to achieve mechanised longwall excavation using the techniques perfected for cutting slots in the face. Partial success was achieved using machines such as the Anderson Boyes Meco-Moore cutter loader, shown in Figure 2.3. However, it was the shearer, conceptualised and developed by James Anderton, who was the UK National Coal Board (St Helens area) production manager, which proved successful. The Anderton shearer was patented in 1953 and the first production machine was commissioned in 1954 (see Figure 2.4). Nowadays shearers totally dominate the world-wide market for longwall mining machines.

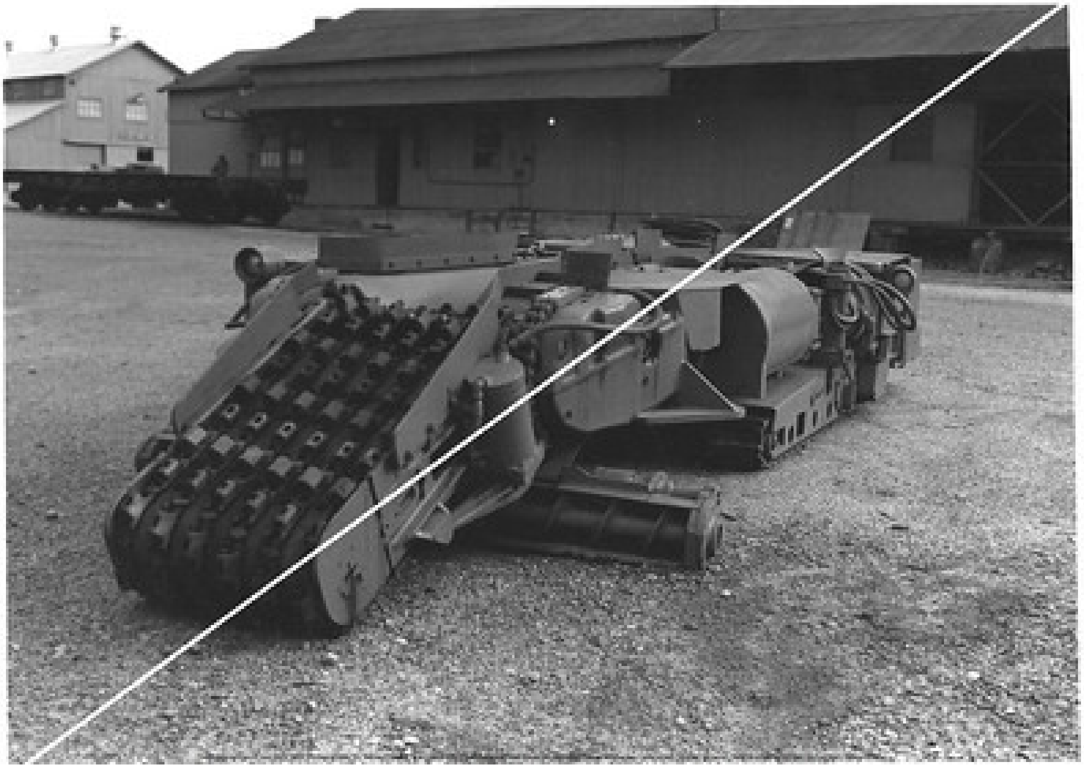


Figure 2.2 The First Joy Mining Machinery Continuous Miner – JCM3

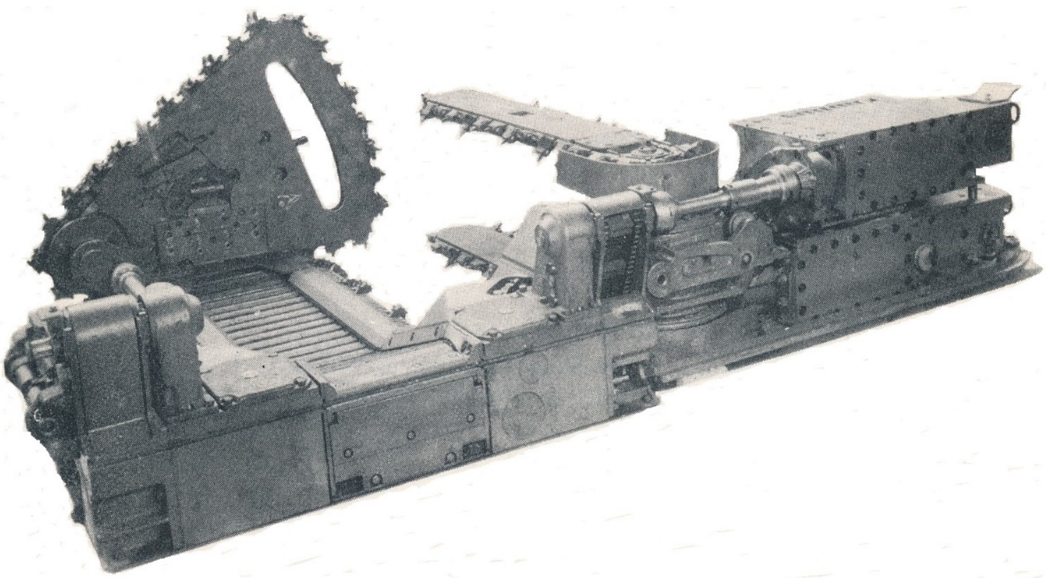


Figure 2.3 The Anderson Boyes Meco-Moore cutter loader

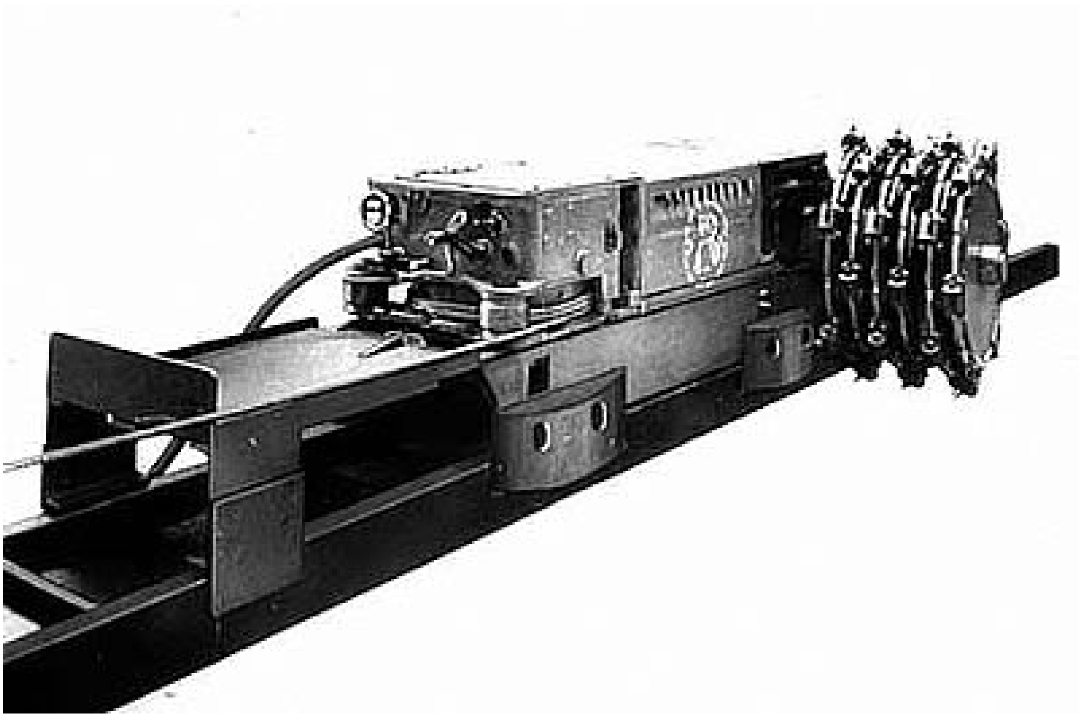


Figure 2.4 An early Anderton single drum longwall shearer

All the machines discussed so far rely for their action on the successive and sequential penetration into the face of a large number of cutter picks disposed in array along the periphery of, or across the face of, the machine's cutting element.

However, conventional picks are not suitable tools for excavation in hard and abrasive rock formations. The first machine to use disc cutters in an attempt to tackle hard rock was invented in 1856 in the United States. This machine undertook a trial in granite near Silver Plume in Colorado, where a 2.1m diameter tunnel only managed to advance 10m with much difficulty.

The free rolling disc cutter was first used successfully in the early 1950s when the Goodman Manufacturing Company developed a tunnel boring machine based on the work of J S Robbins . Three machines were built in 1954 which successfully excavated tunnels for the Oahe Dam in the United States. These were the fore-runners of the present day tunnel and raise boring machines. Following the untimely death of J S Robbins, the Robbins Company under the direction of his son Richard (Dick) Robbins became a leader in the field of tunnel boring as well as being the dominant supplier of raise boring machines. A large diameter tunnelling machine equipped with disk cutters is shown in Figure 2.5 below.

2.2 Rock cutting mechanics research in the UK prior to 1973

Rock cutting mechanics, as a serious subject of study and research, owes its origins to the British National Coal Board with the formation of the Mining Research Establishment's coal cutting group in 1954. This was the time when new longwall techniques and the newly available Anderton shearer were being applied. There was considerable trepidation regarding shearers since experience with coal ploughs had not been good.

Figure 2.5 Disc cutters disposed on the head of a tunnel boring machine

It was, in fact, the relatively poor performance in Britain of the otherwise very successful coal plough in Germany that stimulated research into the cutting of coal. The first British plough installation, at Morrison Busty Colliery in 1948, was very successful but subsequent attempts at applying the technique were often abortive (Foreman and Adams, 1953). It was recognised that the performance of this machine was very sensitive to coal hardness and also variations in hardness both along the face and within the height of the seam. In addition to the National Coal Board's research, the Department of Mining Engineering at the University of Newcastle upon Tyne also conducted research into the ploughing of coal (Potts and Roxborough, 1961), (Potts et al, 1967).

By 1970 the effects of coal strength and the results of changing the pick geometry were reasonably well understood. However, plough faces were still not achieving production targets due to component failures, particularly blown fluid couplings between the drive motors and the chain sprockets at either end of the face and also jammed haulage chains due to the generation of slack chain at the non-driving sprocket. Any variation in face length or size of drive motors resulted in weeks and sometimes months of trial and error on production coal faces before success was achieved. It was obvious that some form of simulation could give valuable insight and assist in mine mechanisation.

In 1968 NASA declassified many of their simulation programs used for the space programme. Among this software was the Continuous Simulation Modelling Program (CSMP) written in Fortran IV. IBM saw the commercial of advantage of this software and released it for use with IBM mainframe computers. Although cumbersome to use (Fortran programming skills were not sufficient to input system data and a simulation language, S/360 CSMP had to be learned), the results provided valuable insight into the practical application of coal ploughs (Phillips, 1971). In particular, understanding the effects of changing chain pretension, different face lengths and the required size of the plough drive motors greatly assisted mechanisation teams new to this system.

Since mechanised faces using ploughs, cutter loaders and shearers all increased the rate at which longwall faces advanced, it became obvious there was a need to excavate access roadways at a similar rate creating the need for machines capable of cutting through rock. It was a logical step to extend the coal cutting research to embrace other rock formations. The 1960s was also a period of world-wide major civil construction in areas such as road and rail transport, water supply, sewerage, power stations, hydro-electric schemes and nuclear waste disposal.

In the United Kingdom there was great interest in the early 1970s in the construction of a tunnel under the English Channel. For geological reasons the rock formation the tunnel would follow was the Lower Chalk, while for geotechnical reasons it was decided that a smooth bored, concrete lined tunnel would be constructed. The UK government agency charged with evaluating the feasibility of a machine driven Channel Tunnel was the Transport and Road Research Laboratory (TRRL) and, because of the University of Newcastle upon Tyne's reputation for coal cutting research, a contract to determine "The Mechanical Cutting Characteristics of the Lower Chalk" was secured.

The Lower Chalk consists of 7 or 8 different vertical zones and samples from two different levels of the Pitstone quarry were sourced. The characteristics of chalk vary enormously with moisture content, varying from being a brittle rock when dry to behaving as a plastic material when wet. The experimental work proved more difficult than expected and indicated that a prohibitive number of block samples would be required, with many failing as shown in Figure 2.6. For this reason the pick cutting programme concentrated on only two levels of rake angle and tool clearance angles were not considered. Similarly the performance of only three roller cutters was investigated (110mm diameter discs with edge angles of 60º and 90º and a 110mm 12 tooth roller cutter).

Despite these limitations the research programme was considered a success and yielded invaluable data (Roxborough and Rispin, 1972) and (Phillips and Fowell, 1976). This data was used in the design of the 5 metre diameter tunnelling machine used in the Chinnor, Oxfordshire trials and the subsequent Channel Tunnel machine in 1974 (O'Reilly et al, 1976)

The research conducted in both coal and chalk was made difficult by the variation in properties between successive samples and also by the innate weaknesses in the samples. The chalk samples were generally fractured before delivery while coal is a notoriously anisotropic and heterogeneous material.

Figure 2.6 Collapse of a Lower Chalk sample
(Source: Roxborough and Rispin, 1972)

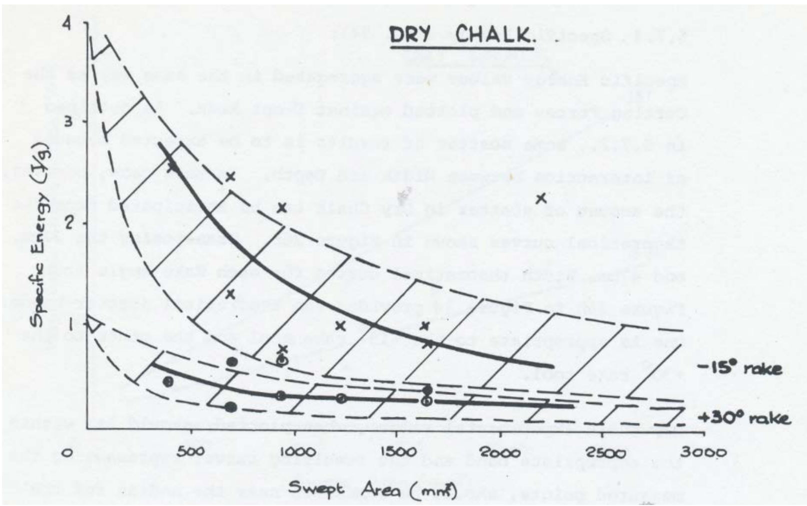


Figure 2.7 Typical scatter in chalk cutting results
(Source: Roxborough and Rispin, 1972)

2.3 Personal research into cutting rock with picks

Much of the Midlands and North of England is underlain by two major formations – the New Red Sandstones and Magnesium Limestone. With considerable tunnelling activities planned for the 1970s and beyond (the 2nd and 3rd Mersey tunnels, the Liverpool Loop railway tunnels, several interceptor sewerage tunnels to prevent raw sewerage entering rivers and the decline tunnels for the Selby coalfield), mechanised excavation in these two formations was of interest to the Transport and Road research Laboratory. Following completion of the Chalk cutting contract, TRRL issued a new 3 year contract to investigate the mechanical properties and cutting characteristics of the Bunter Sandstone and Magnesian Limestone.

In January 1973 I took up my appointment as a Senior Research Associate in charge of this new contract under the direction of Dr Frank Roxborough. It was indeed a propitious moment to begin a research career in this field, since a number of advantages coincided at this time.

1. A considerable body of knowledge had already been accumulated and this allowed the choice of appropriate levels for each of the pick variables being investigated
2. Excellent linear rock cutting rigs were already available at the Department of Mining Engineering, University of Newcastle upon Tyne and these were complete with both steel and aluminium triaxial dynamometers and data recording systems. This meant meaningful research could commence without delay
3. The TRRL contract provided adequate funding for salaries and student bursaries, together with overhead costs (technician time, workshop consumables, travel, etc.)
4. It was possible to obtain good quality block samples of both rock types from two quarries. The variations between different blocks were very small allowing multiple blocks to be used in experiments with an absolutely minimal influence on results
5. A six week visit to the University of Newcastle upon Tyne in January 1973 by the renowned researcher Professor M.M. Protodyakonov(Jr) coincided with the planning of the large factorial experiments. Protodyakonov was a member of the Academy of Sciences of the USSR and, in addition to his reputation as a mathematician and rock mechanics researcher, he had also been involved in the planning of very large agricultural experiments which, because of the impact of differences in weather conditions, all needed to be completed in one growing season. It was possible with his help to plan the work required under the TRRL contract.

The research undertaken under this contract led to two major reports, my own PhD thesis and a number of journal and peer reviewed conference papers. These outputs are discussed in Section 2.3.2

2.3.1 Planning of the pick cutting experiments

The main programme of experiments was planned to provide data through which relationships could be established between criteria of cutting performance and the several variables available in pick cutter design and operation, as shown in Figure 2.8 below.

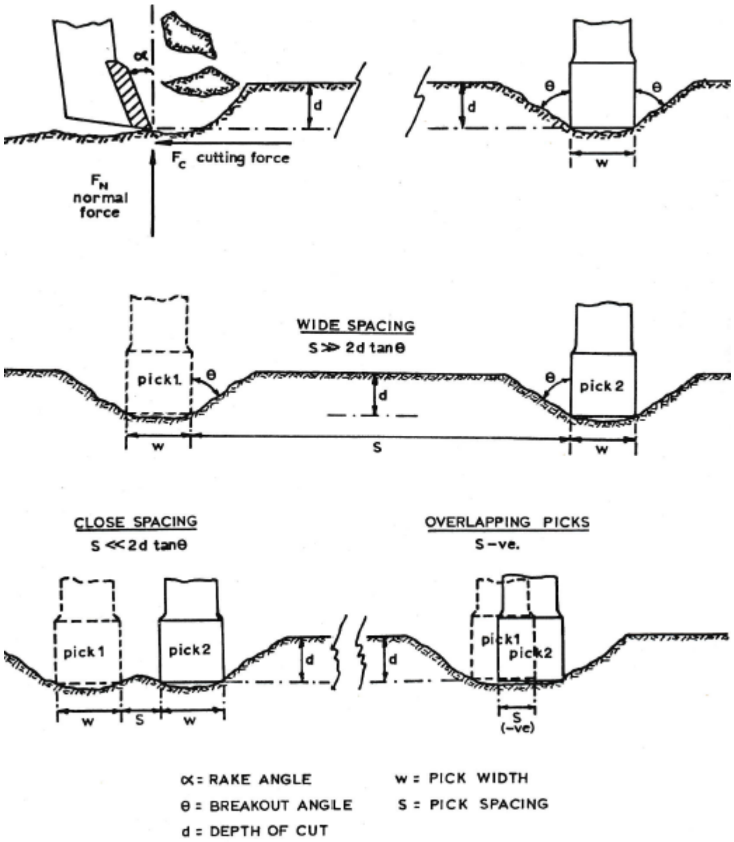


Figure 2.8 Geometry of cutting situations
(Source: Roxborough and Phillips, 1975a)

In order to establish a reliable relationship between one of the principal variables with each performance parameter (cutting and normal forces, volume of rock excavated and the energy used), previous research (Roxborough and Rispin, 1972) had indicated the need to study its effect at no less than 5 levels and that, if possible, the levels should form an arithmetic progression.

Table 2.1 Comparison of Chalk and Bunter Sandstone test programmes

Variable	Symbol	Levels	
		Chalk	Bunter Sandstone
Front Rake Angle	α	2	5
Width	w	3	5
Depth of Cut	d	3	5
Spacing	s	6	5
Moisture Content	m.c.	2	2
Cutting Speed	v_p	-	-

This indicated immediately that the workload for the Bunter Sandstone and Magnesian Limestone contract would far exceed that completed in Chalk.

A great deal of previous work on the effect of cutting speed had shown that all the parameters, except possibly tool wear rate, are unaffected by speeds well beyond the range used by machines. Moisture content was considered at the two extreme values of dry and saturated. However, it can be seen from Table 2.1, that a full factorial experiment involving 4 variables at 5 levels and 1 variable at 2 levels would lead to

$$(5)^4 \times (2)^1 = 1250 \text{ cutting tests}$$

Furthermore, these 1250 tests would have provided only 1 result for each condition. Analyses of previous rock cutting data had shown that since every type or rock exhibits some heterogeneity, several replications are necessary. A preliminary series of cutting tests in Bunter Sandstone showed that the coefficient of variance of cutting and normal forces was usually about 10% and always below 15%. It was decided therefore that each test should be carried out 4 times and represented as a mean value. Four replication for each of 1250 tests using picks, together with a considerably larger number for discs, would have created a workload of equivalent to 10 man years. This was clearly unrealistic and the use of partial factorial planning as advocated by Professor Protodyakonov was a necessity.

The technique used to develop partial factorial matrices in both the main pick cutting and the main disc cutting experiments was the well-known Latin Squares method (Fisher, 1970). A pre-requisite of Latin Squares is that all variables should appear at the same number of levels, i.e. for this rock cutting contract the decision was 5 levels. The process followed has been fully explained (Protodyakonov and Teder, 1971) (Roxborough and Phillips, 1975a) and (Phillips, 1975a). The resulting partial factorial matrix, showed that each level of a variable appears in only one combination with each level of the other variables. The “master plan” for the main pick cutting experiment is shown in Figure 2.9. The need for statistical replications was satisfied by executing the same planned programme four times.

In addition to the programme of tests with simple chisel picks, additional experimental work was performed using picks of complex shape (vee bottom, ridge front and side rake angle). The effects of groove deepening, corner cutting and cutting with picks under simulated rock stress conditions were all studied in subsidiary experiments. The effects of changing the composition of the tungsten carbide tips used in pick cutters were also explored (Phillips and Roxborough, 1981).

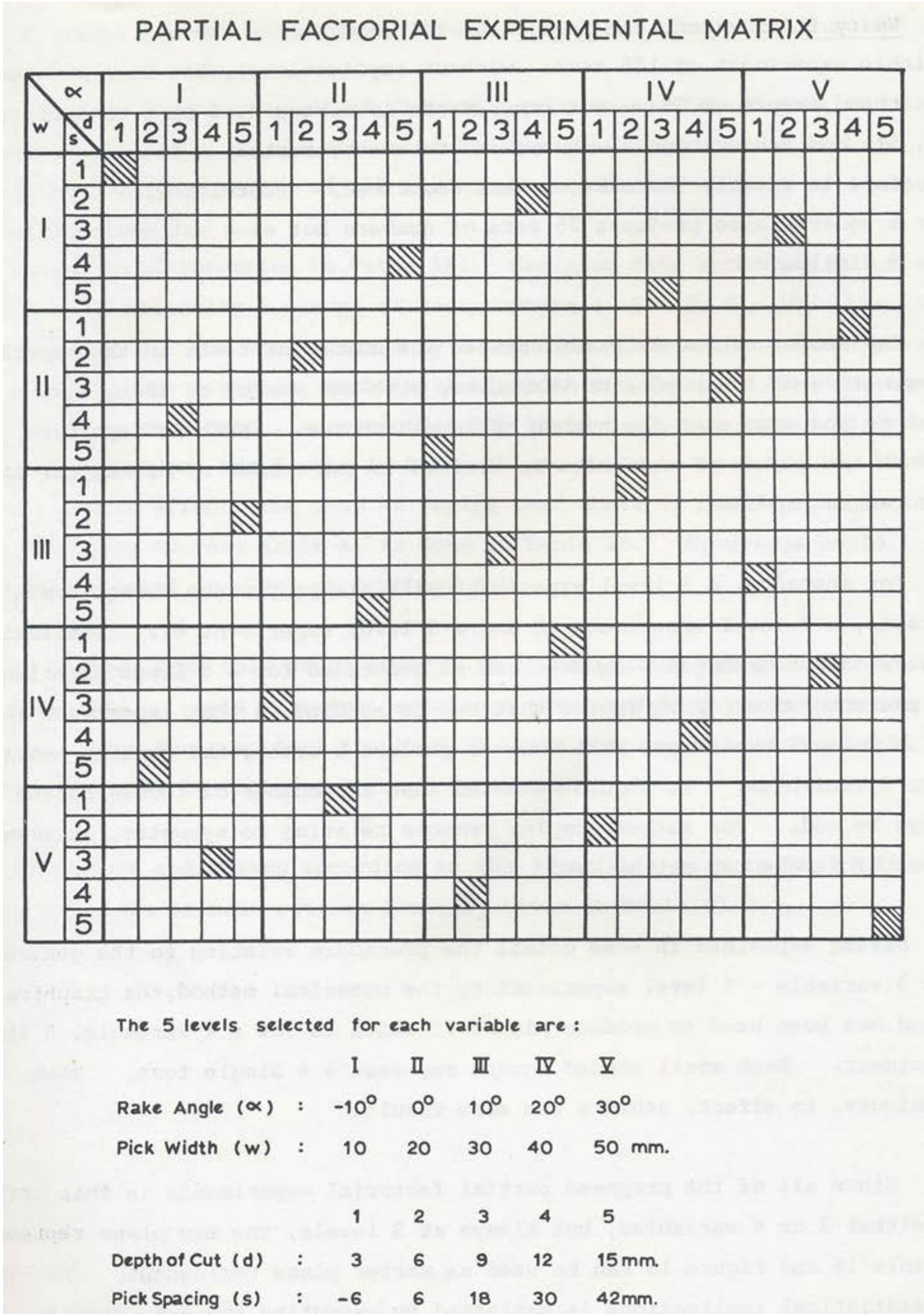


Figure 2.9 Matrix for partial factorial pick cutting experiments
(Source: Phillips, 1975a)

2.3.2 Key findings of my pick cutting research

The results of the pick cutting experiments in Bunter Sandstone were reported to the Transport and Road Research Laboratory (Roxborough and Phillips, 1975a) while the end of contract report contained both the Bunter Sandstone and Magnesian Limestone results (Phillips, 1975b). The final report also formed part of my Ph.D. thesis (Phillips, 1975a). In addition 4 papers dealing with pick cutting were published in peer reviewed journals or at prestigious peer reviewed conferences.

The first of these papers (Roxborough and Phillips, 1974) presented the findings of the main pick cutting experiment in dry Bunter Sandstone. The paper explained the need to undertake partial factorial experiments, the technique adopted in planning the experiment and then went on to show how unambiguous data could be obtained when experiments are conducted in uniform rock samples. To illustrate this Figure 2.10 shows the scatter of results in the Lower Chalk and the need to make informed decisions based on theoretical calculations. In fact, the results in coal cutting experiments (Evans and Pomeroy, 1966) show cleats and bedding planes can complicate the analysis even further.

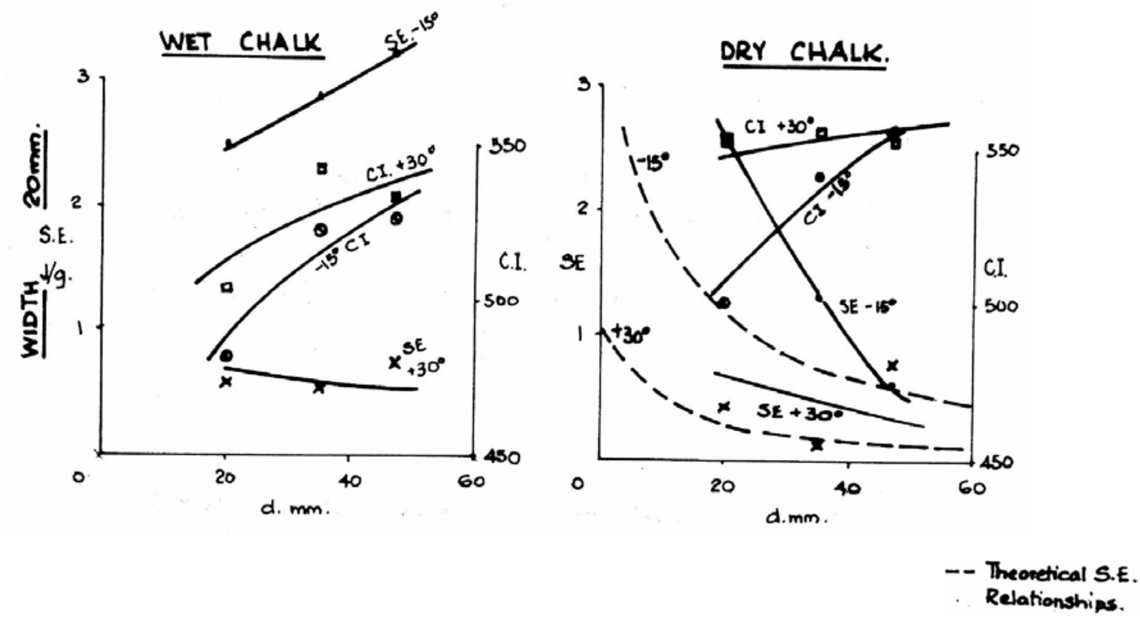


Figure 2.10 Specific energy and coarseness index results in Chalk
(front rake angles of +30º and -15º)
(Source: Roxborough and Rispin,1972)

Figure 2.11 shows the equivalent graphs for cutting tests in Bunter Sandstone. Here the trends are obvious. While these results did not change the conclusions of previous work they did unconditionally confirmed the effects of changing the design parameters of chisel picks (rake angle and width), together with effects depth of cut has on the forces, yield and specific energy (Roxborough and Phillips, 1974). Similar graphs were produced for forces, yield and specific energy against both the pick width and pick rake angle.

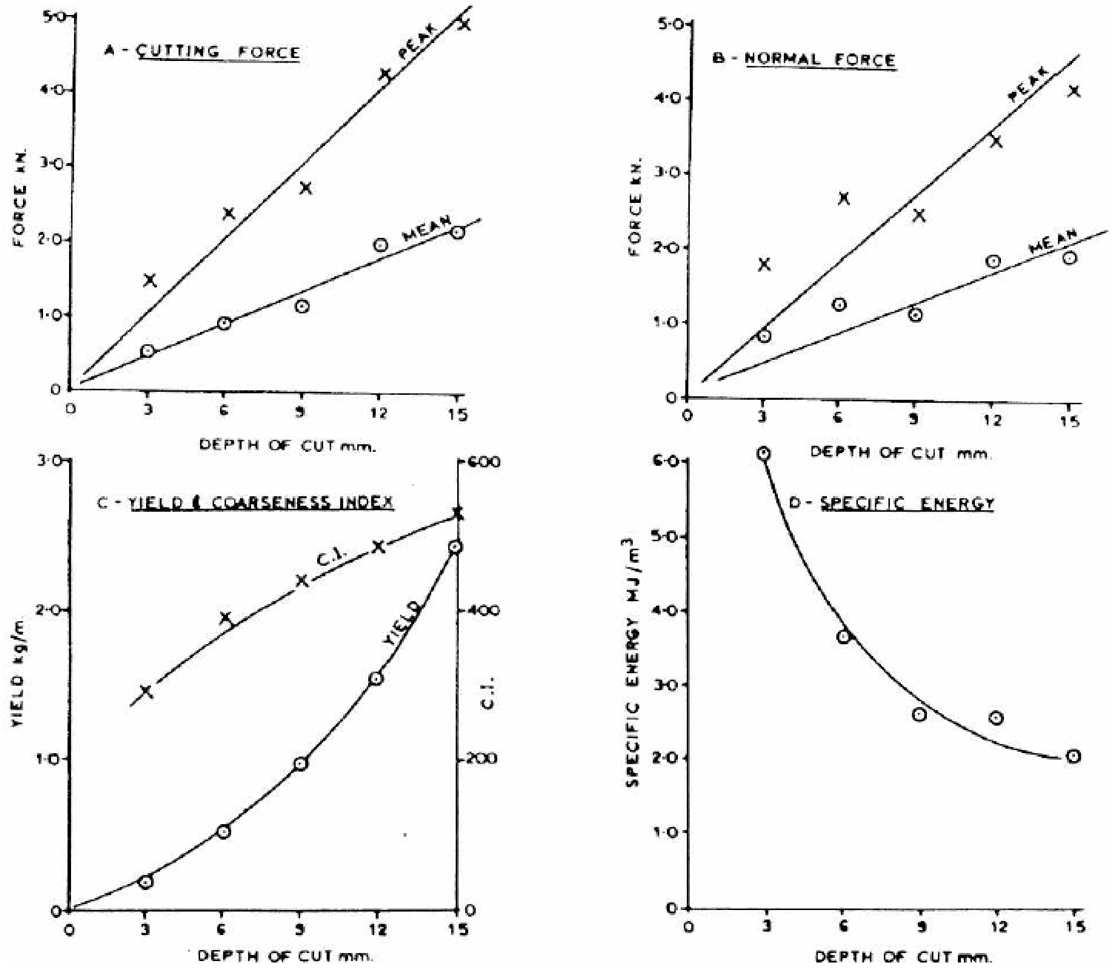


Figure 2.11 Specific energy and yield results in Bunter Sandstone
(front rake angle of +10º and width of tool 30 mm)
(Source: Roxborough and Phillips, 1974)

The specific energy relationships are presented in two ways in Figure 2.12. To the left the direct effects of spacing are shown with the minimum specific energy occurring at close spacings for the shallow cuts and increasing spacings as the depth of cut becomes larger. The locus of minimum specific energy was of a similar form to the unrelieved relationship between specific energy and depth as shown in Figure 2.11. This led to the realisation that when spacing is expressed as a multiple of cutting depth, as shown on the right in Figure 2.12, the minimum specific energy values all occur at the same s/d ratio of between 2 and 2.5. Since the same range of s/d values was also found in Magnesian Limestone, this provided a practical guideline for manufacturers wishing to optimise the cutting efficiency of pick arrays on the cutting heads of their machines.

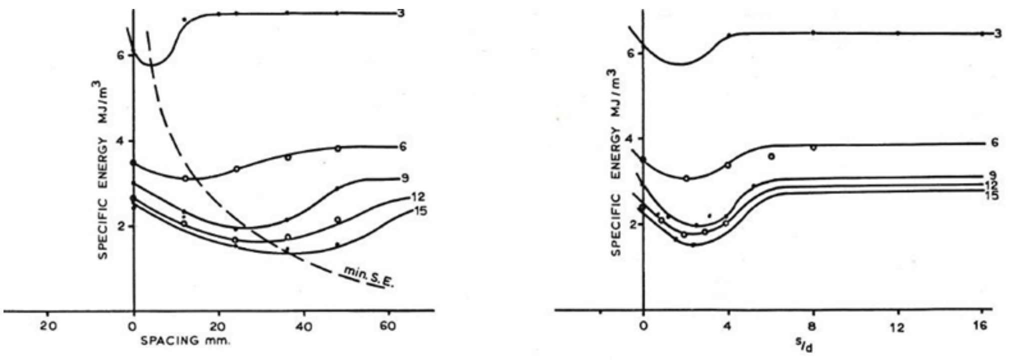


Figure 2.12 The relationship between specific energy, spacing and depth of cut in Bunter Sandstone (Roxborough and Phillips, 1975a)

In the 1970s coal mining equipment manufacturers saw the potential to market their machines for excavating soft to medium strength rock. On occasions when these machines had insufficient thrust, and excavated through a series of shallow cuts, two phenomena occurred. If the ridges of rock were strong enough pick boxes could be damaged or even torn off the cutting head. Occasionally before this could happen the torque required by the cutting head would increase to the point where the head would stall. Both phenomena indicated the need to examine the effects of taking a series of shallow cuts at wide spacing (see Figure 2.8). This particular experiment involved deepening grooves up to a maximum of 18mm in 6 successive cutting depths increments of 3mm. Only one tool configuration was used: a simple chisel of rake +10º and width 10mm.

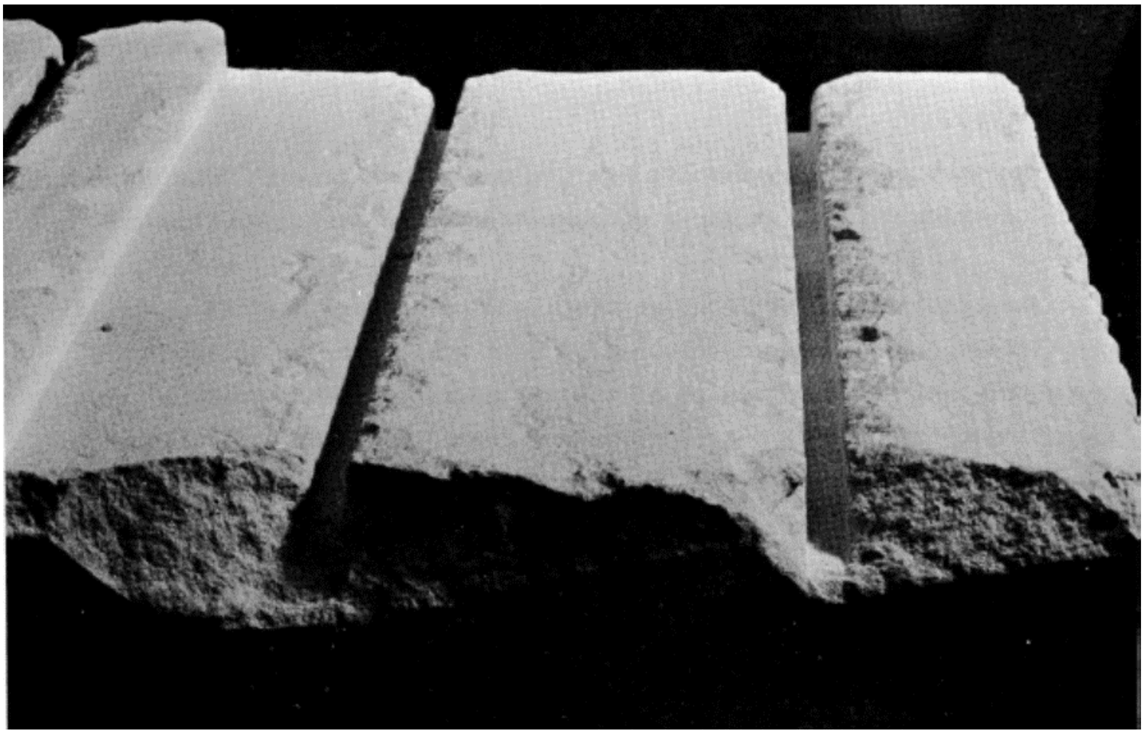


Figure 2.13 Incrementally deepened grooves
(Source: Roxborough and Phillips, 1975a)

Progressive deepening of a cut was found to produce a groove of fundamentally different shape to that produced when cutting to the same total depth in one pass of the tool. Figure 2.13 shows that lateral breakout is inhibited as the tool moves deeper eventually reaching a situation where the cut and swept volumes are the same. Figure 2.14 illustrates the difference between a succession of shallow cuts and a single cut to the same depth.

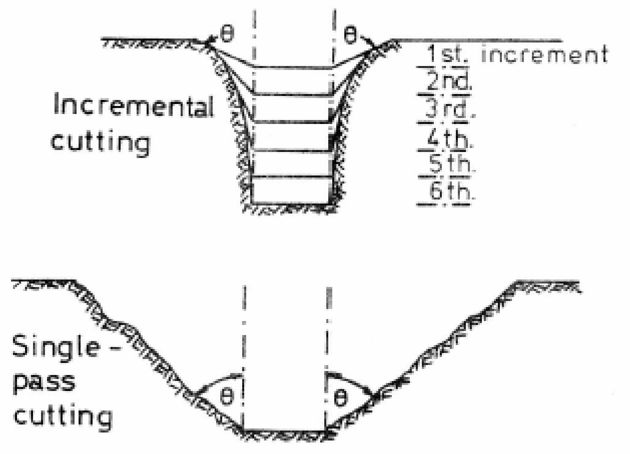


Figure 2.14 Effect of incremental cutting on breakout angle
(Source: Phillips and Roxborough, 1979a)

The specific energy for a 3mm depth of cut was found to increase rapidly over the first few increments, thereafter to approach a constant value after the 5th. Although the results highlighted a dramatic fall in cutting efficiency associated with deepening a groove, they did not provide a full indication of the pernicious consequences of this mode of cutting. What is necessary is to sum the work done in excavating each increment and also measure the total volume of rock excavated. Since the definition of specific energy is the work done in excavating unit volume of rock, Figure 2.15 can be constructed. In this example it can be seen that the cumulative specific energy is at least 7 times that required for a single cut.

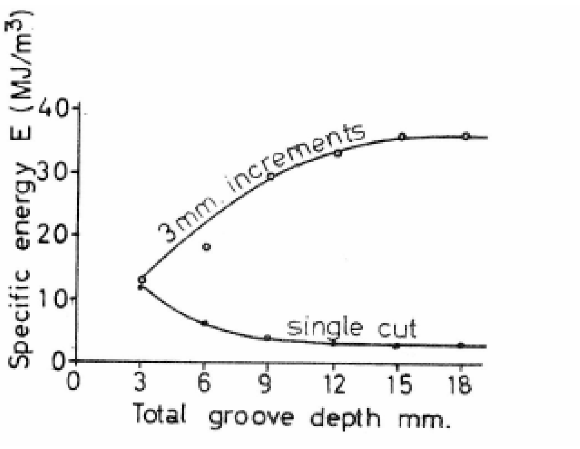


Figure 2.15 Specific Energy for single-pass and incremental cutting
(Phillips and Roxborough, 1979a)

Early longwall shearers pulled themselves along a static chain anchored at either end of the coalface, while continuous miners relied on their mass and the co-efficient of friction between their caterpillar tracks and the floor to provide forward traction. In both cases the forces available to cause the machine to advance were limited. It was this lack of “thrust’ that limited the penetration/ revolution of the cutting drum and was the cause of groove deepening with its significant loss of efficiency in coal mining machines. Once machines were deployed in hard rock, groove deepening often became a “show stopper”. Despite reservations concerning the rigidity of a road heading machine a trial was undertaken in 1978 in a cut-and-fill operation at Mount Isa Mines. Shallow cuts led to groove deepening and Figure 2.16 shows Professor Frank Roxborough and myself underground at the test site in a Mount Isa cut-and-fill stope where the machine trials proved unsuccessful.

Figure 2.16 Underground trials of a road header at Mount Isa, Queensland

During the 1970s machine manufacturers and their pick suppliers moved away from utilizing simple chisel picks, despite the fact that all the research to date had indicated that chisel picks were significantly more efficient than other, more complex shapes. To understand the reasons behind this, the performance of complex shapes was researched using Bunter Sandstone.

The picks most commonly used on the cutting drums of longwall shearers were “radial” picks as shown in Figure 2.17, while continuous miners had moved from being chain type machines, as shown in Figure 2.2, to much more robust drum

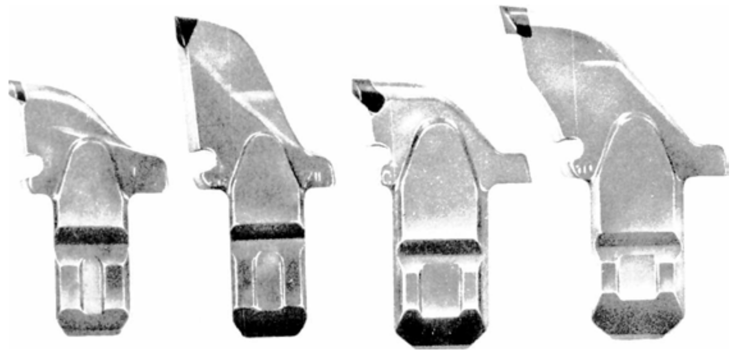
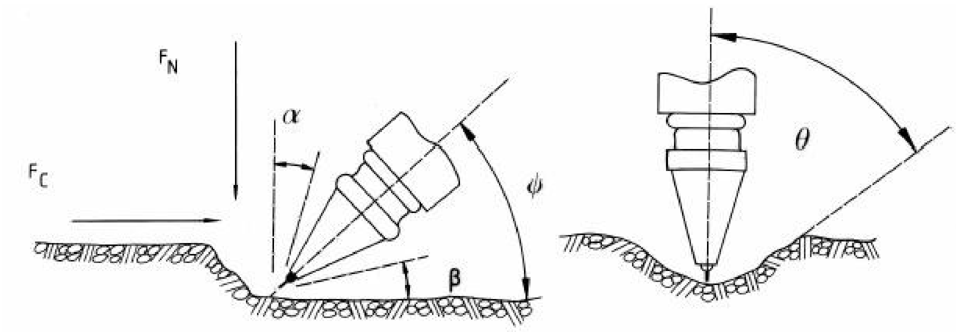


Figure 2.17 Radial picks used on shearers

machines using rotating “pencil point” conical picks, as shown diagrammatically in Figure 2.18



α = rake angle, β = back clearance angle, ϕ = angle of attack, θ = breakout angle

Figure 2.18 Drawing of a conical pick

In order to understand why conical picks, capable of rotating in the pick boxes, were finding favour in industry, a test programme was undertaken in dry Bunter Sandstone using the levels of front-ridge and vee-bottom listed in Table 2.2. All tools were of width 20mm, front rake +10º and 10º back clearance. Obviously a front ridge angle of 180º and/or a vee-bottom angle of 180º represents a simple wedge shape of tool.

The results of the programme of cutting tests using ridged front and vee-bottom picks finally explained why operators preferred complex shaped tools, as well as providing useful data on the rock breakage patterns created by vee-bottom picks (Phillips and Roxborough, 1979a). The results for the ridged front tests are shown graphically in Figure 2.20. The clear implication is that, although a ridged angle reduces the pick forces, it cuts less rock and is thus less efficient than a simple wedge (ridge angle =180º). The disadvantage of a front-ridge angle in cutting efficiency shows most clearly at a shallow depth of cut; at the largest depth, however, the dominating effect of depth on efficiency almost wholly overshadows the influence of front-ridge i.e. at a depth of 15mm a sacrifice of 14% in cutting efficiency leads to savings in cutting and normal forces of more than 30%

Table 2.2 Angles used in the complex shape experiments

Front Ridge Angle	-	90º	120º	150º	180º
Vee- bottom Angle	60º	90º	120º	150º	180º

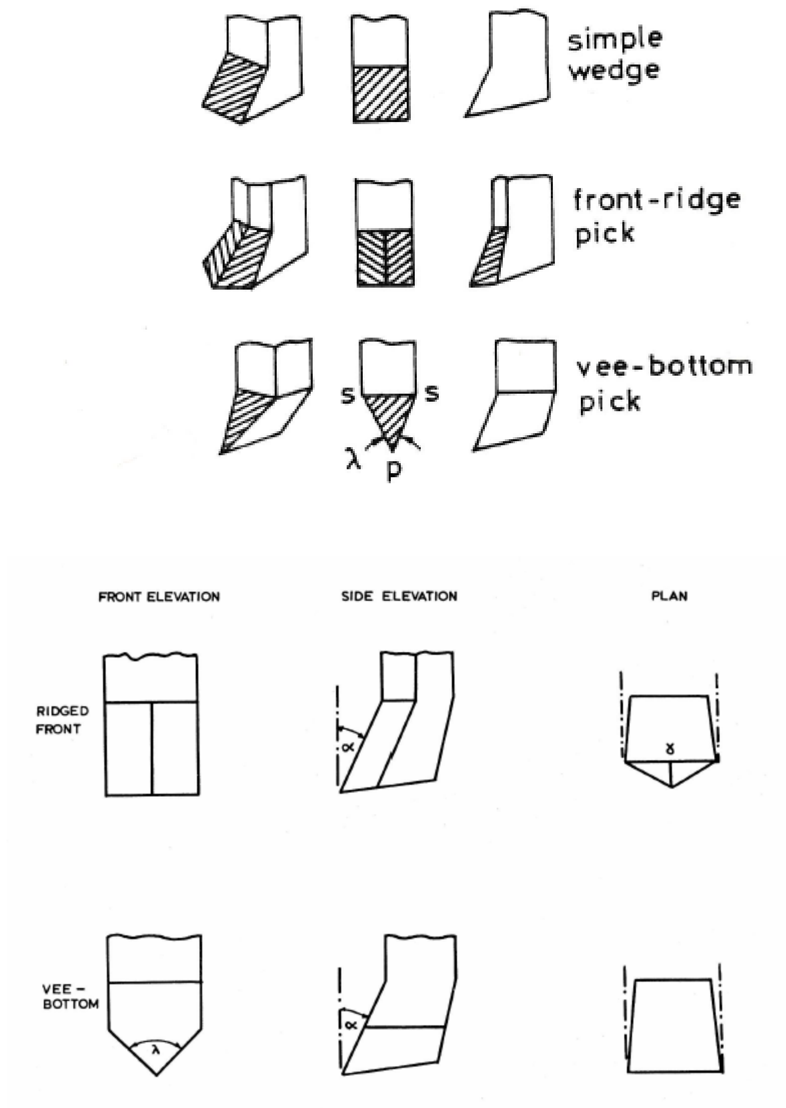


Figure 2.19 Chisel and complex shaped picks
(Source; Phillips & Roxborough, 1979a)

While the force vs front ridge angle graphs appear linear, this is not the case for vee-bottom tools. Figure 2.21 shows that the cutting force is little reduced when the angle is below about 120º, with the normal force following the same trend.

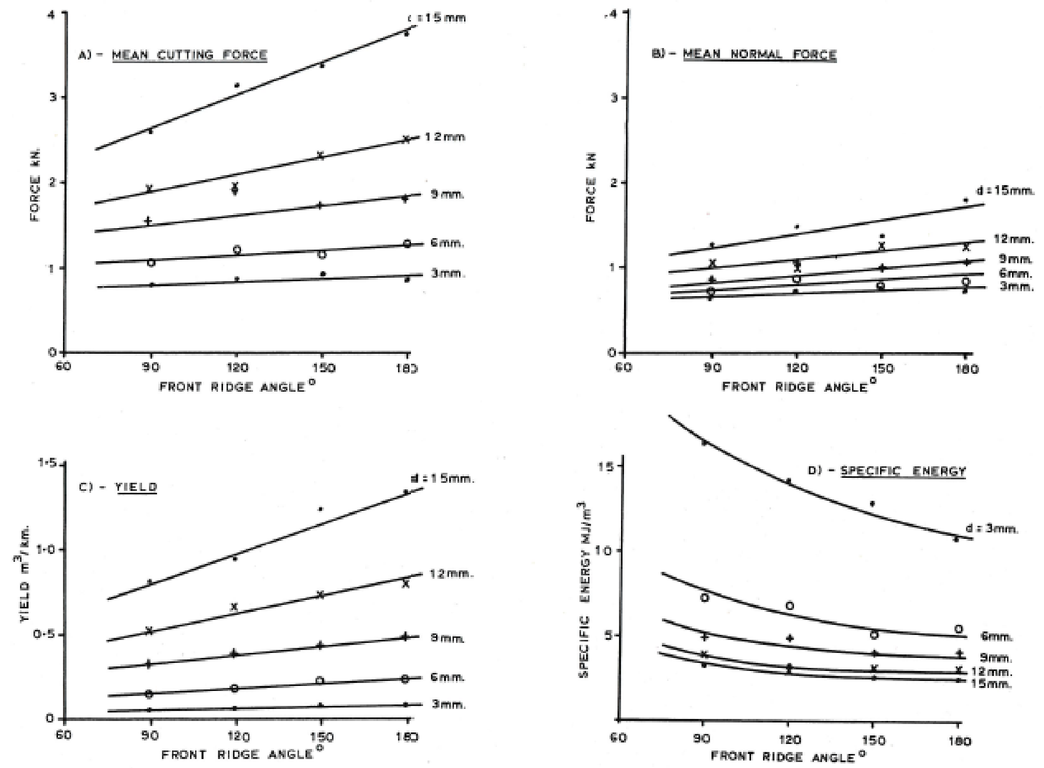


Figure 2.20 Effect of front ridge angle (Source: Roxborough and Phillips, 1975)

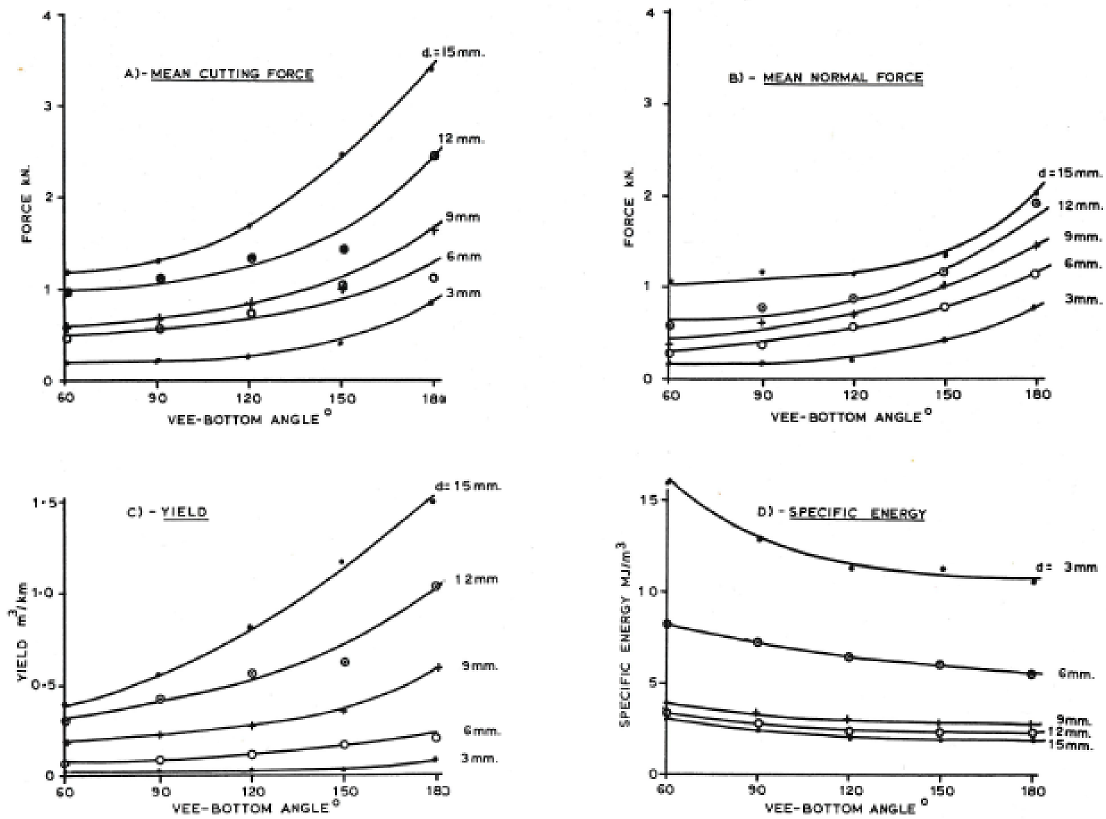


Figure 2.21 Effect of vee-bottom angle (Source: Phillips and Roxborough, 1979a)

The results for rock yield, as shown in Figure 2.21, were found to decrease as the vee-bottom angle became more acute, resulting in specific energy being the lowest for the simple wedge. Similar results had been taken by the National Coal Board's cutting mechanics group to support their view that simple wedge shapes were always the most efficient. In order to explain why machines and particularly continuous miners performed better with point attack picks it was necessary to consider the action of the machine as a whole. In order to sump into the coalface the machines available in the 1970s used friction between their caterpillar tracks and the coal or rock floor. The co-efficient of friction between steel and dry solid coal, μ, had been shown to be about 0.3 and that the presence of water would reduce this by at least 30% (Evans and Pomeroy, 1966). Given that the floor in room and pillar coal mining is almost inevitably covered with water and mud, the co-efficient of friction which, together with the weight of the machine, determines the thrust the machine can apply is likely to be well below 0.2. In effect the thrust, which in turn determines the maximum normal force each pick can apply, is limited. Using this argument Figure 2.22 was constructed from the data shown in Figures 2.20 and 2.21 and this can be used to illustrate the following line of reasoning.

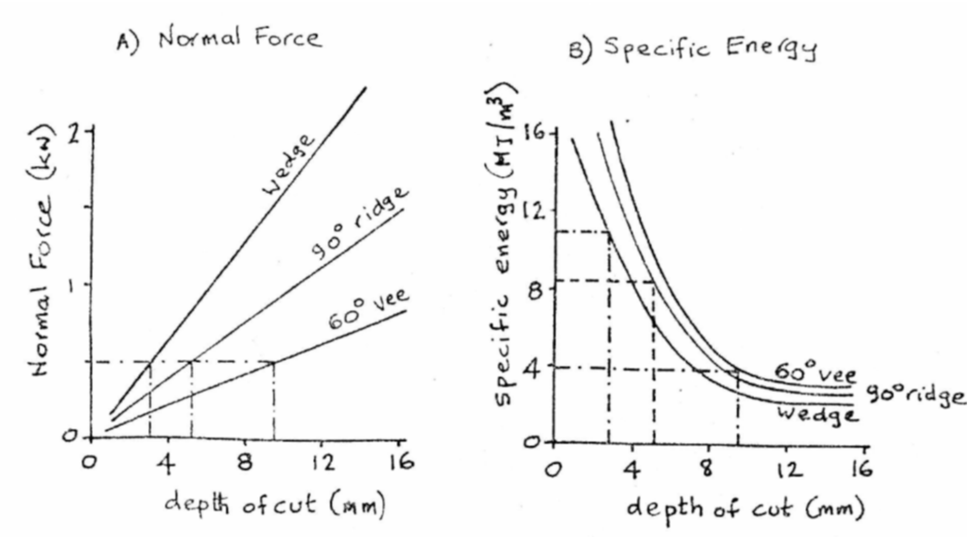


Figure 2.22 Penetrating capability of different pick shapes
(Source: Roxborough and Phillips, 1981b)

Suppose that a continuous miner has only 0.5 kN of available thrust per pick in order for its cutting head to penetrate the face. (0.5 kN is a very low value in practical terms but this is of no consequence for the purpose of this example). From Figure 2.22, at 0.5 kN of normal force the wedge (chisel pick) can achieve a depth of 3mm, the ridged pick about 5mm and the vee-bottom pick 9.5mm.
At these respective depths of cut the following operational specific energies will apply

Wedge	11.0 MJ/m^3
90º ridge	8.3 MJ/m^3
60º vee	3.5 MJ/m^3

A combination of ridge and vee-bottom, as occurs in the pencil point pick, is therefore likely to lead to an even better operational efficiency. When considered in this way there is no conflict between the established fact that the simple wedge is the most efficient shape when all tools are cutting at the same depth and the better efficiency of pointed tools which can operate at greater cutting depths.

My final direct involvement with pick cutting research was the opportunity presented by a six month period of study leave away from the University of New South Wales. It was spent, between July and December 1981, at the Mining Operations Laboratory of the Chamber of Mines Research Organisation (COMRO) in Johannesburg.

Continuous miners were first introduced into South African coal mines in the mid-1970s and by December 1978 there were 34 machines in operation. However, due to the high strength and abrasiveness of South African coal seams their performance left much to be desired, with frequent machine breakdowns. In fact, at that time, a full 40% of the total downtime for these continuous miners was due to mechanical failures associated with cutting. This was double the figure of similar machines in the United States.

To address this situation COMRO undertook a major research programme consisting of three separate but interrelated projects.

a) a laboratory investigation into the cutting characteristics of a range of South African coal seams, together with investigations into particular cutting situations, e.g. corner cutting. This work made use of a fully instrumented planning machine capable of accepting coal blocks up to one metre cube.
b) an assessment of coal seam cuttability used a portable in-situ penetrometer. This device, which was inserted into boreholes, indented picks and cut grooves. Study of the force records resulted in a data base of comparisons between test results, other coal properties and machine performance, thereby allowing pre-assessment of new sites (Whyte and Grant, 1979)
c) a major study of the performance of a continuous miner when head lacing variables and operational parameters were changed. This programme was designed to determine the relevance and value of the results of all previous basic coal cutting research to the practical production situation.

Although both conical and chisel picks had been used during the continuous miner studies at Kriel Colliery (Roxborough, King and Pedroncelli, 1981), it had been felt that such factors as optimisation of tool geometry would best be left to laboratory experiments where precise measurements could be made and the influence of the dominant variable, depth of cut, could be strictly controlled (see section “a” above). This became my project for 6 months and two research reports were produced. The first dealt with cutting the No 4 seam from Kriel Colliery using chisel picks (Grant, Pieterse and Phillips, 1981), while the second dealt with cutting the corner of the excavation using conical picks as gauge cutters (Phillips et al, 1983).

Because of the difficulties of obtaining blocks of coal and the inherent variability within the samples, cutting variables were studied at only 3 levels. A knowledge of the trends obtained when cutting fairly uniform rocks such as Bunter Sandstone and Magnesian Limestone helped enormously in interpreting the data. While nothing fundamentally new emerged from these experiments, the main aims of the project had always been to quantify the forces involved in cutting the hard South African coal seams and also to show changes in tool performance when mounted in pick boxes of varying angle of attack. This second point had been the subject of considerable debate within the industry, with conflicting claims being made as to the optimum angle of attack.

The main conclusions of this work (Grant, Pieterse and Phillips, 1981) were

1. Forces and specific energy are considerably reduced by the provision of a back clearance angle of up to 6 or 7º. Any further increase in this angle appeared to have no influence on tool performance. In the abrasive South African coal, chisel picks rapidly wear until the back clearance angle is zero. This provides more than adequate justification for the choice of conical picks.
2. The optimised spacing to depth ratio was found to be about 3 and this resulted in a 30% reduction in cutting force and specific energy over the unrelieved situation
3. The angle of attack has been defined earlier in Figure 2.18. Increasing it has a positive influence on the cutting performance of new picks provided the back clearance angle for the pick is still between 6º and 7º. Situations where the back clearance became small or negative resulted in forces climbing rapidly and should be avoided, i.e. where the angle of attack is too large in relation to the pick tip geometry. Under these circumstances very high forces, particularly normal forces are experienced and a continuous miner would experience difficulties in sumping.

The second COMRO report dealt with cutting into a corner of the excavation (Phillips et al, 1983). The important role of corner cutting in the mechanised excavation of coal had long been recognised. However, the advent of machines employing fewer picks, each taking deep cuts at slower speeds, had highlighted the need to quantify and understand the peculiar requirements of gauge cutters. This is particularly true when considering the design of continuous miners, since these machines possess a corner cutting sequence at either end of the drum. Indeed pick arrays on the Lee Norse continuous miner that had been used for experimental purposes at Kriel Colliery required between 30 and 45 per cent of the total number of tools to be disposed in corner cutting situations. This disparity between the duty of line-cutters and gauge cutters was common knowledge in the early 1980s and machine manufacturers compensated for this by using two or sometimes three cutters per line for excavating the corner. In this way forces were thought to be more equally distributed amongst the cutters.

The main conclusions of this report (Phillips et al, 1983) were

1. The restricted space at the drum ends, together with the common practice in the 1980s of using end-rings to mount pick boxes disposed at two or more skew angles, inevitably leads to a much closer pick spacing than that adopted across the remainder of the drum. The results of this research indicated that the number of picks per line for corner cutting could be reduced from three to two without the cutting and normal forces exceeding those experienced by the line cutters, i.e. a gauge cutter operating in an array and to a depth of say 22.5mm is stressed to approximately the same level as a line cutter excavating 45mm in a single pass. This of course presumes relief is provided to the corner cutting tool.

2. If the cutting sequence starts in the corner and works towards the centre-line of the machine, then not only are the individual tool forces found to be significantly higher, with a consequent greater risk of pick damage, but the overall performance of the array is impaired. This effect is most pronounced when considering two identical arrays designed to cut out of and into a corner. The experimental studies showed that, for a corner cutting array, torque increased by 20%, thrust requirement by 30% and specific energy by 22% when cutting out of rather than into a corner. When it is considered that the number of picks on a cutter head has steadily been reduced over the years since this report was produced, it can be appreciated that savings in torque, thrust and energy consumption of the order of 20 to 30% are highly significant and that this research was the first time these advantages were quantified.

2.4 Personal research into cutting rock with disc cutters

Tunnel boring machines had been around since the early 1950s and by the late 1960s raise boring technology was beginning to prove itself a viable alternative to the more hazardous, traditional methods. The majority of these machines, when deployed in soft and medium strength rock formations, made use of free rolling disc cutters, as shown in Figure 2.5. A valuable insight into the performance of disc cutters had been obtained by research, principally being conducted in the United States (Morrell et al, 1970 and Rad & Olson, 1974) and also in Japan (Takaoka et al, 1973). However, this work generally involved single, or at most, two variable studies in a range of rocks.

Since the pick studies in both the Bunter Sandstone and the Magnesian Limestone had produced such unambiguous results, a similar programme was undertaken using disc cutters. The variables most likely to influence disc performance and the five levels at which each would be studied are given in Table 2.3.

Table 2.3 Levels of variables in main disc cutting experiment

Variable	Symbol	Levels
Diameter	D	100, 125, 150, 175, 200 mm
Penetration	p	2, 4, 6, 8, 10 mm
Edge angle	ϕ	60, 70, 80, 90, 100º
Cutting speed	v	76, 102, 127, 152, 178 mm/sec
Spacing	s	12, 24, 36, 48, 60 mm

These variables, together with the three forces which were measured are depicted in Figure 2.23.

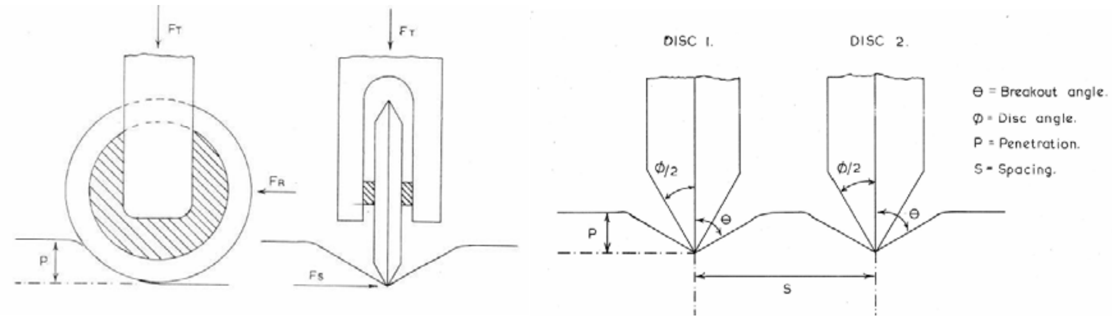


Figure 2.23 Disc cutter variables and forces
(Source: Phillips and Fowell, 1976)

Since it was considered likely that there would be interaction between some of the above variables, it would normally have been necessary to undertake a full factorial experiment involving all the interacting variables. Full factorial coverage of 5 variables each at 5 levels would have involved 12 500 cutting tests, allowing a minimum of 4 replications. Hence the same design of partial factorial experiment, as previously described in Chapter 2.3.1 above, was used.

The results for the unrelieved (i.e. discs operating in isolation) cutting programme in Bunter Sandstone resulted in the establishment of unambiguous relationships. As an example Figure 2.24 shows the effects of penetration at the mean diameter (150mm) and mean edge angle (80º) of the experimental matrix. Changing the speed had no discernible influence on the measured parameters. While thrust force increased with disc diameter, rolling force and yield remained constant. Since the specific energy is calculated by dividing rolling force by yield it also is unaffected by changes in disc diameter (Roxborough and Phillips, 1975a). A study of the effects of edge angle revealed that both thrust and rolling force increase with increasing edge angle, while yield remains constant. This results in the specific energy, which is the inverse of efficiency, increasing as more blunt discs are used. All the above results, which were published in a journal paper (Roxborough and Phillips, 1975b), merely confirmed previously published information and did not significantly increase our knowledge concerning the behaviour of discs.

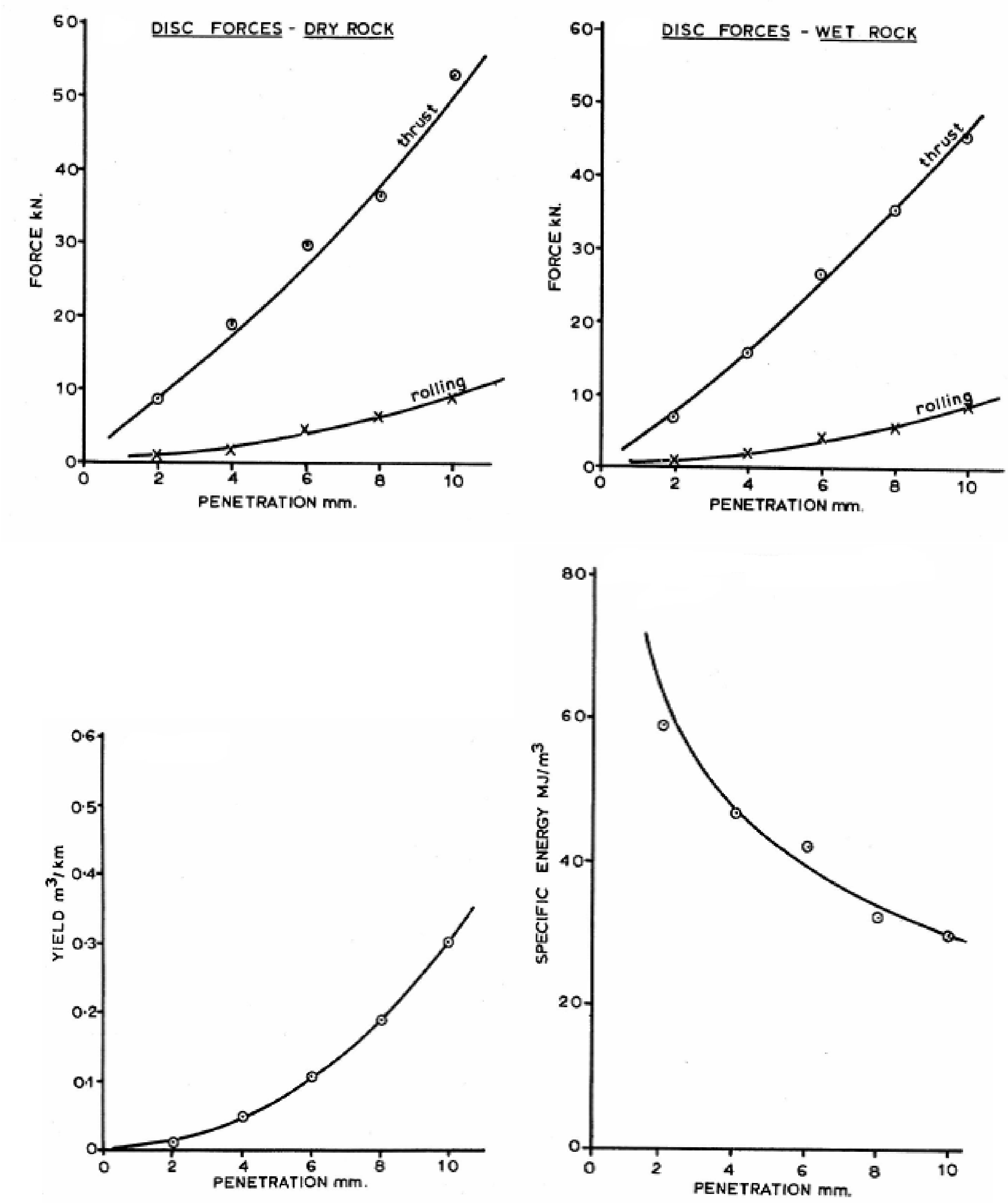


Figure 2.24 The effects of a disc cutting at different, unrelieved penetrations (Source: Phillips, 1975a)

However, undertaking this laboratory work stimulated my interest in seeking a more theoretical understanding of the action of disc cutters.

The starting point was the appreciation that the forces involved in cutting with a disc will be influenced by the strength of the rock it is required to penetrate and the geometry of the disc. As the development of the theoretical analysis progressed it became necessary to make several assumptions and approximations, none of which departed too far from reality, but the introduction of which significantly simplified the mathematics.

The situation depicted in Figure 2.25 represents a static disc of edge angle ϕ and diameter D, penetrating a rock surface to a depth p. The vertical thrust force necessary to cause this penetration is designated as F_T.

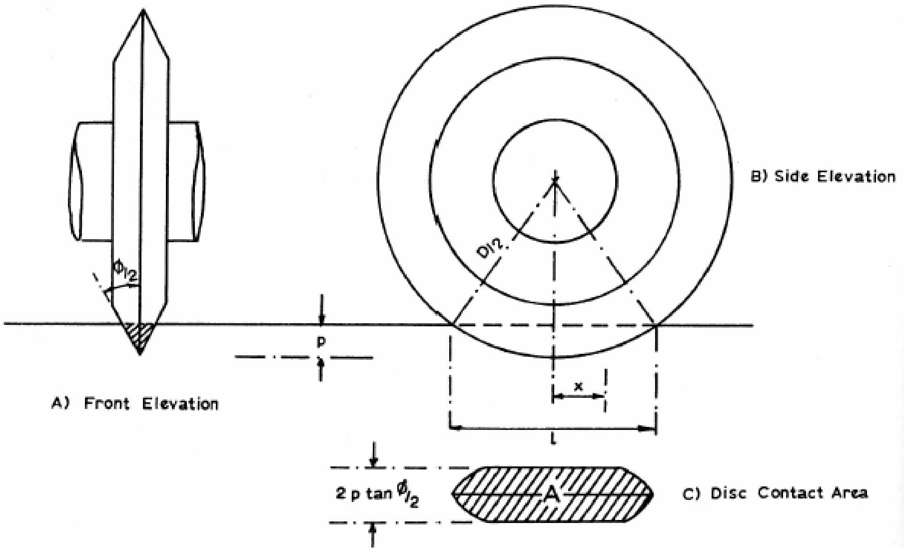


Figure 2.25 Geometry of disc penetration
(Source: Phillips, 1975a)

As penetration increases the chord length of contact, l, also increases since

$$l = (Dp - p^2)^{½} \text{.............................(Eq 2.1)}$$

The resistance to penetration can be assumed to be compressive and that the thrust force causing the penetration is equivalent to a compressive stress acting over the projected area of disc contact 'A'. Although a precise determination of area A was initially undertaken, it was established that the complex expression could be simplified with negligible loss of accuracy to

$$A = 2pl \tan\frac{\phi}{2} \text{...(Eq 2.2)}$$

This analysis led to the mathematical model for calculating the thrust force to cause a penetration of p being:

$$F_T = 4\sigma \tan\frac{\phi}{2} (Dp^3 - p^4)^{½} \text{..........................(Eq 2.3)}$$

where σ is the unconfined compressive strength of the rock.

While establishing equation 2.3, which is for the force necessary to push a static disc into a flat surface, was relatively straight forward, modelling the rolling force under dynamic conditions took several more months and relied on making several assumptions and relied on some close observations of the disc cutting experiments.

The first observation was that the thrust force for static penetration of the rock surface remains substantially constant when the disc is made to roll. This is illustrated in Figure 2.26 and this observation was shown to be repeatable. The reasons are not fully understood but it was observed in practice that the broken rock remains in place behind the disc and so it is reasonable to suppose the buried section of the disc remains in bearing contact with the rock.

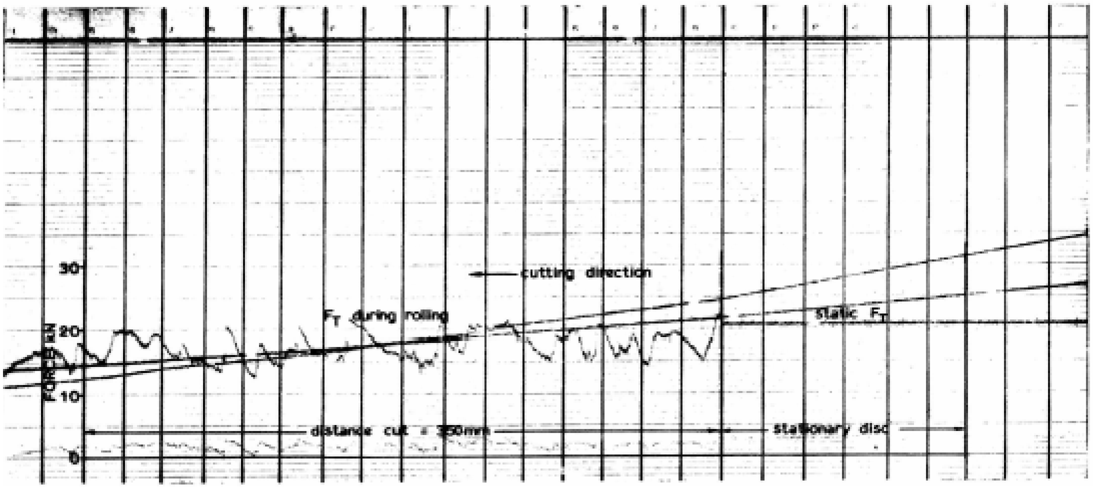


Figure 2.26 Ultra-violet trace of thrust force for stationary and rolling conditions (Source: Phillips, 1975a)

Mathematical modelling is of limited academic value unless the results can be verified by at least some physical measurements. This was done initially using the results obtained by Morrell and others working at the US Bureau of Mines (Morrell et al, 1970). They had performed disc cutting tests on two rocks, Tennessee Marble quoted as having a uniaxial compressive strength of 116 MPa, and Indiana Limestone with a compressive strength of 69 MPa. Using equation (2.3) theoretical estimates of thrust force were compared with the measured thrust force for two 178mm discs (60º and 90º edge angles). The good agreement, which is illustrated in Figure 2.27, provided encouragement to try to extend this analysis.

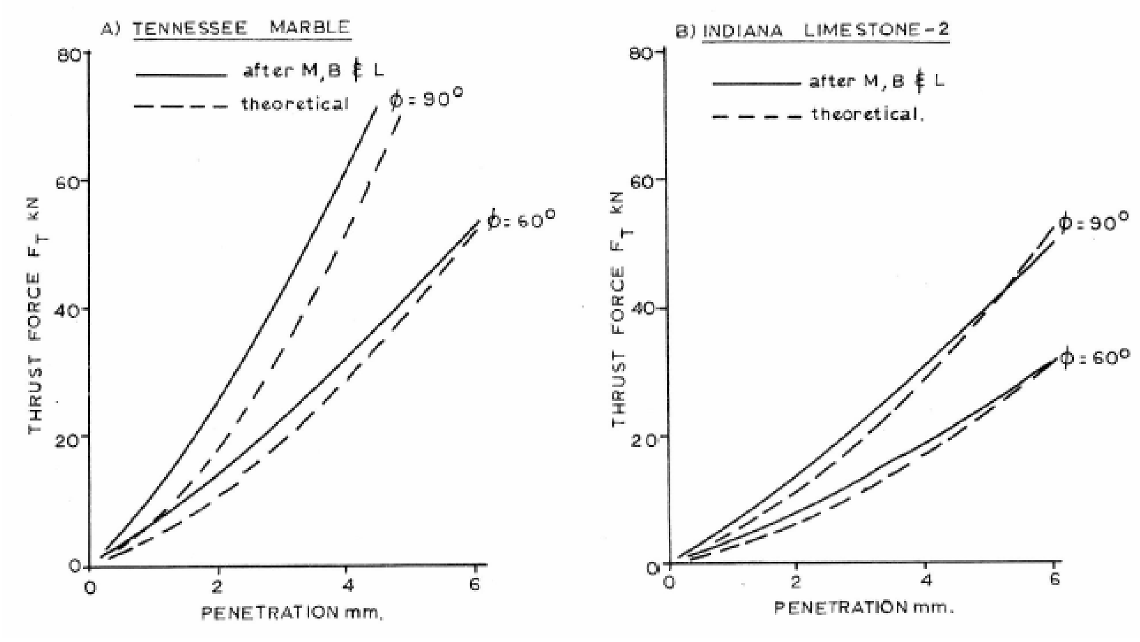


Figure 2.27 Comparison of theoretical thrust forces with measured results (Phillips, 1975a)

The next observation was that a disc is free rolling rather like the front wheel of a bicycle. Since there is no torque driving it, the resultant of any forces causing it to rotate must pass through the centre of rotation.

In order to establish an equation linking the rolling force F_R with rock strength the forces shown in Figure 2.28 must be considered.

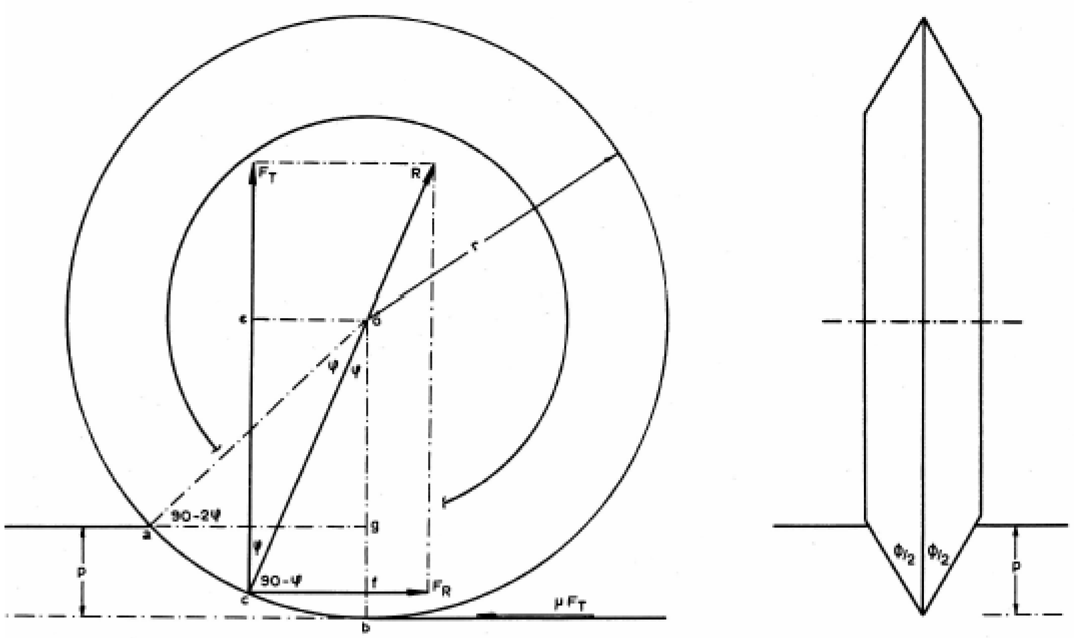


Figure 2.28 Orthogonal forces acting on a disc
(Source: Phillips, 1975a)

If the resultant force not only passes through the centre of rotation but also acts at the centre of the arc of contact, point c, then for zero torque

$$F_R.of = F_T.oe$$

so that

$$\frac{F_T}{F_R} = \cot\psi \text{..................................(Eq 2.4)}$$

and

$$\frac{og}{oa} = \frac{\frac{D}{2} - p}{\frac{D}{2}}$$

$$= \cos 2\psi$$

$$= \frac{1 - \tan^2\psi}{1 + \tan^2\psi}$$

hence

$$\tan^2\psi = \frac{p}{D-p}$$

or

$$\frac{F_T}{F_R} = \left(\frac{D-P}{p}\right)^{1/2} \text{...........................(Eq 2.5)}$$

Combining equations (2.3) and (2.5) gives a direct expression for the rolling force as

$$F_R = 4.\sigma.p^2 \tan\frac{\phi}{2} \text{.....................(Eq 2.6)}$$

This analysis has ignored the friction term μF_T. Rolling friction is likely to be the most appropriate coefficient and with a value of less than 0.01 (Cook, 1968) the effect of friction is likely to be negligible.

It was possible to verify equations (3) and (6) using the results of cutting tests in seven rocks that had been used for disc cutting tests at the University of Newcastle upon Tyne (Phillips and Bilgin, 1977). Thrust and rolling forces were predicted using the uniaxial compressive strength

Table 2.4 Uniaxial compressive and shear strengths of test rocks (MPa) (after Phillips and Bilgin, 1977)

Test	Anhydrite	Bunter Sandstone	Dunhouse Sandstone	4 Fathom Limestone	Gypsum	Magnesian Limestone	Mansfield Sandstone
UCS	112.9	49.2	55.8	127.3	45.0	84.9	71.3
Shear	12.5	7.3	11	20	4.5	12	11

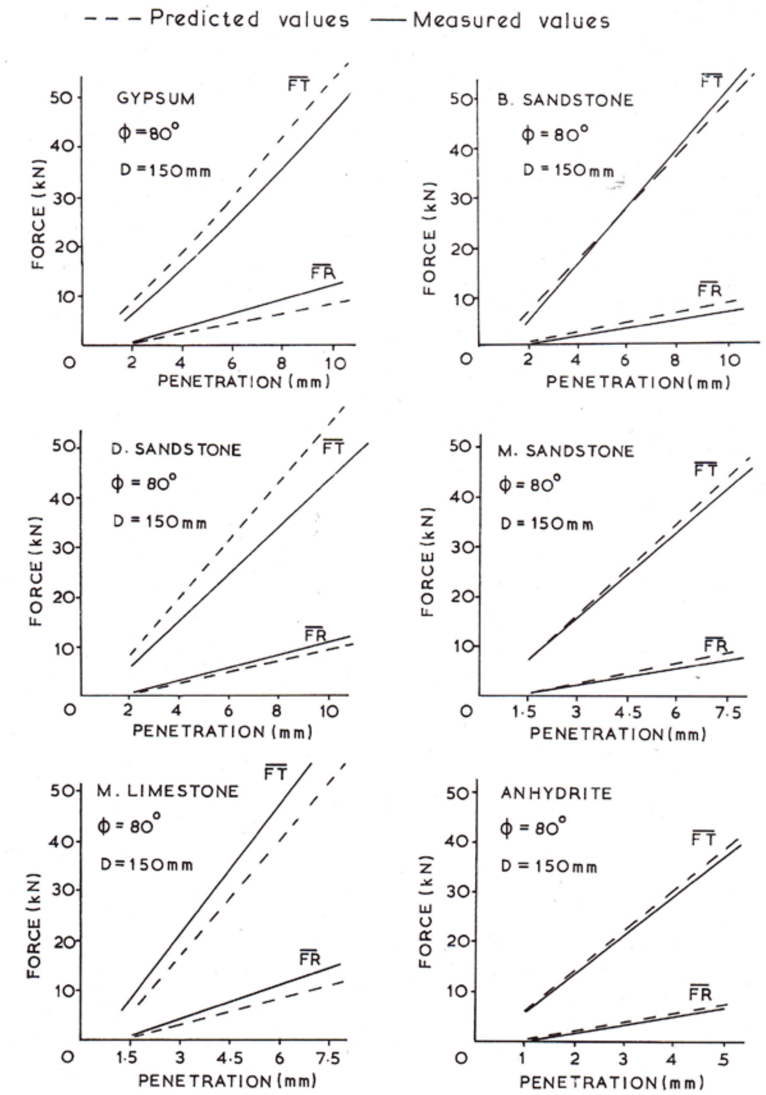


Figure 2.29 Variation between measured and predicted forces against penetration (Phillips and Bilgin, 1977)

Having developed equations which allow the prediction of thrust force and rolling force, the last remaining element in the theoretical description of disc behaviour was the question, which rock properties determine the optimum spacing of discs and how they are interrelated?

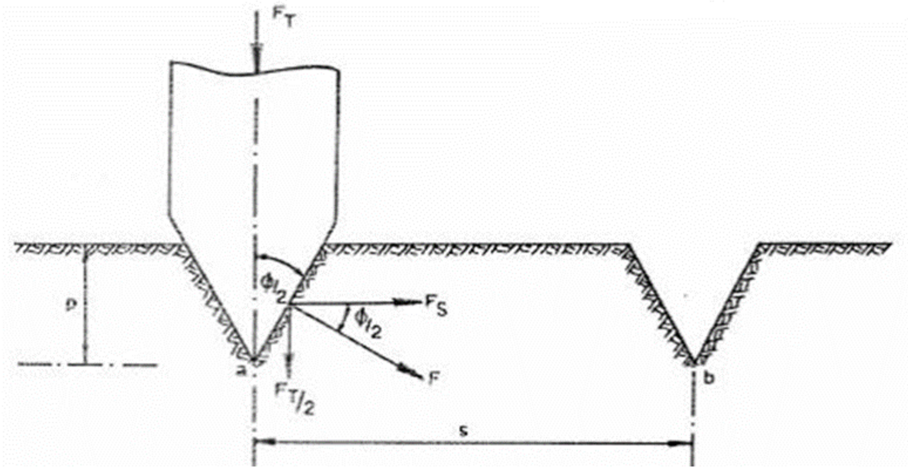


Figure 2.30 Shearing force between disc grooves
(Source: Phillips, 1975a)

Spacing, as depicted in Figure 2.29, was one of the variables studied at five levels in the main experimental matrix. Zero spacing is when one disc travels directly behind or in the shadow of a previous disc. Other levels of spacing are shown in Table 2.3. Initially the rate of increase in the forces is high, but as the spacing becomes larger, both forces tend to a constant value, which in the case illustrated, is reached at between 40 and 45mm, beyond which interaction ceases and the disc is effectively operating in isolation. This limit of interaction in Figure 2.31 is peculiar to the mean penetration level of 6mm. Obviously a shallower penetration would reduce the maximum spacing for interaction and conversely a higher penetration would increase it. For this reason spacing is normally combined with penetration to provide the ratio s/p, which in the case of Figure 2.31 defines the limits of interaction as a ratio of s/p = 7.

The trends which are apparent for disc forces are very similar to those found when cutting with picks.

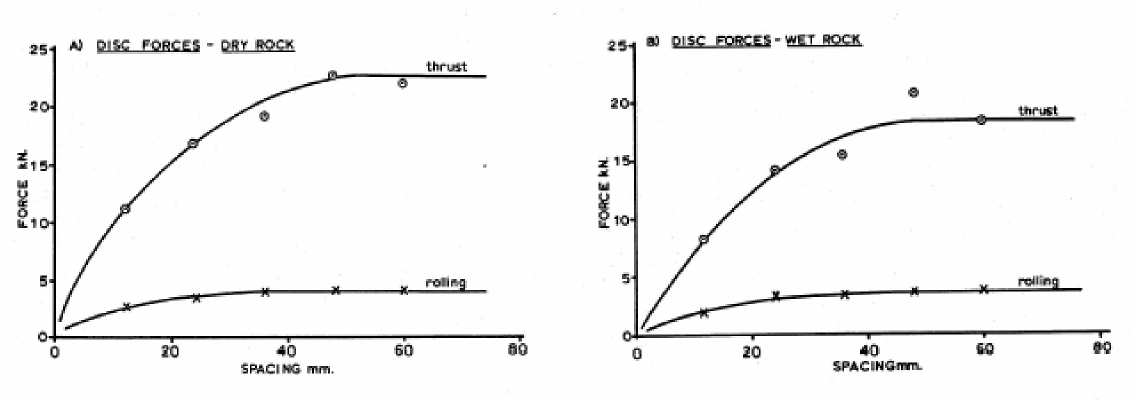


Figure 2.31 Disc forces against spacing
(Roxborough and Phillips, 1975a)

Unlike the disc forces, rock yield from disc cutting is very different to that when cutting with picks. With picks yield increases and approaches the unrelieved yield asymptotically, whereas with discs there is a small range of s/p ratios where yield can be as much as twice the unrelieved value. This is best illustrated by expressing yield as a multiple of the yield obtained from the equivalent disc operating unrelieved. This introduces the parameter Q_R/Q_U, where Q_R and Q_U are the relieved and unrelieved yields respectively. A typical trend for Q_R/Q_U is shown in Figure 2.32. In addition, this results in a very obvious s/p ratio where the efficiency of disc cutting is at a maximum (efficiency is the inverse of S.E.)

The results presented in Figure 2.32 are for discs cutting at a penetration of 6mm in Bunter Sandstone.

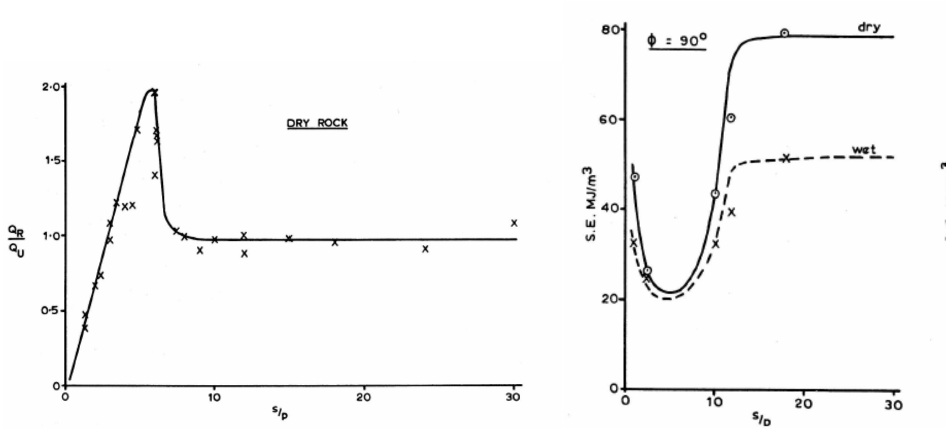


Figure 2.32 Yield and specific energy against s/p ratio (Roxborough and Phillips, 1975b)

The yield was observed to fall abruptly to its unrelieved value when the s/p ratio was approximately 7 and the physical explanation of the peak yield was that slabbing was occuring between adjacent grooves. Similar slabbing has been observed during tunnel boring operations, when the thrust applied to the cutter head creates the appropriate level of disc penetration to change the debris from "flour" to slabs of rock. An example of such a slab is shown in Figure 2.33. Such breakage implies a shearing of the rock across a surface extending between the centre of each groove and at the full depth of penetration.

These observations allowed the establishment of a theoretical model for the optimum spacing of discs on a cutter head. If, as indicated in Figure 2.30, the rock shears across the horizontal plane bounded by a,b and if F_S is the shearing force:

$$F_S = F.\cos\frac{\phi}{2} \quad \text{……………..………………(Eq 2.7)}$$

and

$$F_T = 2F.\sin\frac{\phi}{2}$$

so that

$$F_S = \frac{F_T}{2.\tan\frac{\phi}{2}} \quad \text{……………………………......(Eq 2.8)}$$

Figure 2.33 Slabbing between adjacent discs cutting South African Norite

The same assumption was made as that invoked when equation 2.3 was being developed, i.e. the force F_T is equivalent to a compressive stress acting over the projected surface area and for a disc contact length l:

$$F_T = 2p.\tan\frac{\phi}{2} . l.\sigma \text{ ………………………(Eq 2.9)}$$

where σ is the compressive strength of the rock

combining equations (Eq 2.8) and (Eq 2.9) gives

$$F_S = p\ l\ \sigma \text{..(Eq 2.10)}$$

also, if the rock fails in simple shear due to F_S, then

$$F_S = s.\ l.\ \mathcal{T} \text{ ...……………………….....(Eq 2.11)}$$

where $\mathcal{T}$ is the shear strength of the rock.

Equating Eq 2.10 and Eq 2.11 now gives

$$s/p = \sigma/\mathcal{T} \text{ ....(Eq 2.12)}$$

This rather elegant solution using the ratio of compressive to shear strength for predicting the optimum s/p ratio seems too simplistic. However, when first developed it was compared with Bunter Sandstone results. The unconfined compressive strength and shear strength, as quoted in Table 2.4, were 49.2 and 7.3 Mpa respectively. Hence the ratio of σ/ $\mathcal{T}$ is 6.7, which corresponds well with the observed limit of interaction when yield returns to its unrelieved level, as shown in Figure 2.32.

Eventually the Bunter Sandstone research (Roxborough and Phillips, 1975a) was extended to a further 6 rock types (Phillips, 1975b) and (Phillips and Bilgin, 1977). In all cases the relationship given in equation 2.12 above was considered to be valid.

Two further aspects of disc cutting were studied and the results published (Phillips, Bilgin and Price, 1978) and (Roxborough and Phillips, 1981a).

Firstly it was recognised that virtually all research into the performance of disc cutters had been undertaken using sharp edged discs, even though the effects of different edge angles had been studied. Only Rad, working at the U.S. Bureau of Mines, had investigated some of the effects of a radiused cutting edge (Rad, 1975)

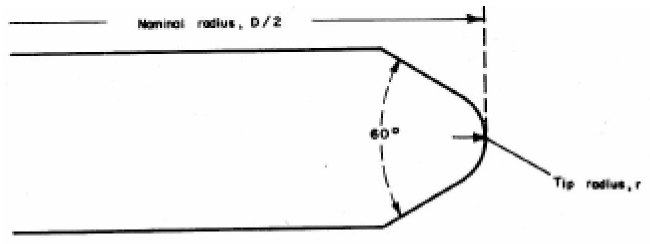


Figure 2.34 Schematic of disc edge radius
(Source: Phillips et al, 1978)

By the mid-1970s tunnel boring machine designs had developed to the point where suffient force and energy was almost always available to fragment the rock. However, their successful application was often determined by their ability to control cutter costs. It was known that the provision of a tip radius, as shown in Figure 2.34, had a dramatic effect on both disc wear rate and the damage inflicted on the disc tyre during cutting. Because of this, research was undertaken using Bunter Sandstone to determine the cost, in terms of disc performance, at which this increase in disc life could be achieved (Phillips, Bilgin and Price, 1978).

As could be expected the results for unrelieved cutting showed forces increased for increasing tip radius, i.e. the more blunt the tool the higher the forces needed to excavate rock. Yield also increased resulting in specific energy (efficiency) remaining virtally constant, except at shallow penetrations, when tip radius and penetration were about the same, see Figure 2.35.

However, it was the relieved cutting tests which proved most revealing. Figure 2.36 shows that the maximum yield varies with tip radius. For sharp discs this maximum occurs at a s/p ratio of 5, while a disc having a 3mm tip radius produces its maximum yield at a s/p ratio of 7. The relationship found between specific energy and spacing were typical of those for disc cutters, with clearly defined minimum values occuring. However, the spacing to penetration ratio for this minimum was found to vary with tip radius, as shown in Figure 2.36. In practice, discs with a 3mm tip radius protecting the cutting edge have an optimum separation between cutters that is 1.4 times the equivalent for sharp discs. This means that 25% fewer blunt discs are required to perform the same duty. This, together with the improved wear characteristics of such tyre configurations, must lead to a substantial reduction in cutter costs.

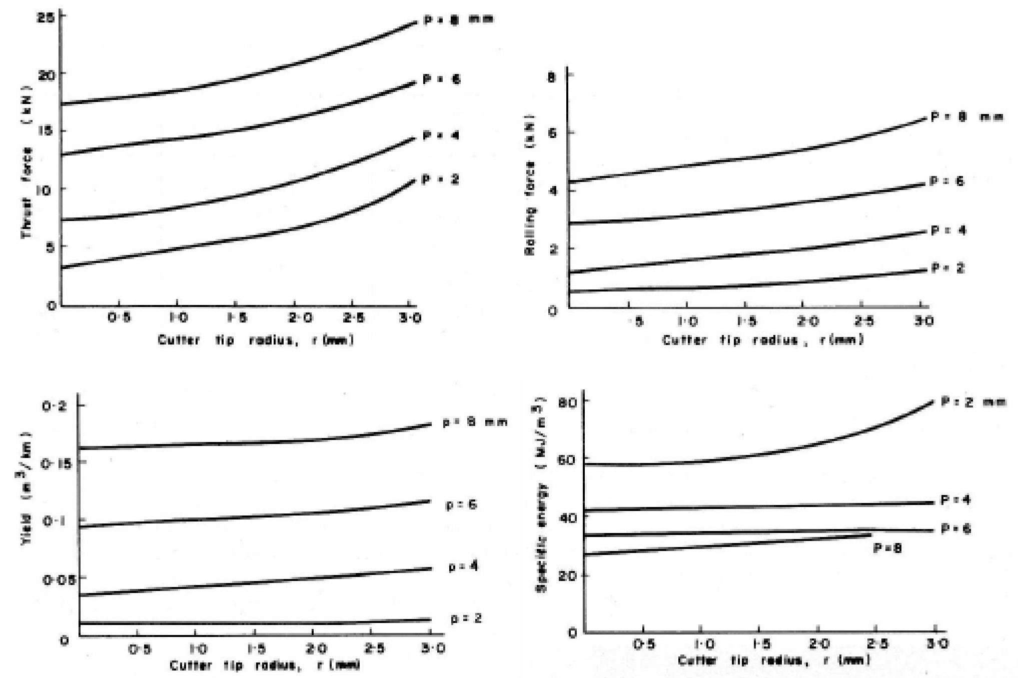


Figure 2.35 The effects of tip radius cutting unrelieved
(Phillips, Bilgin and Price, 1978)

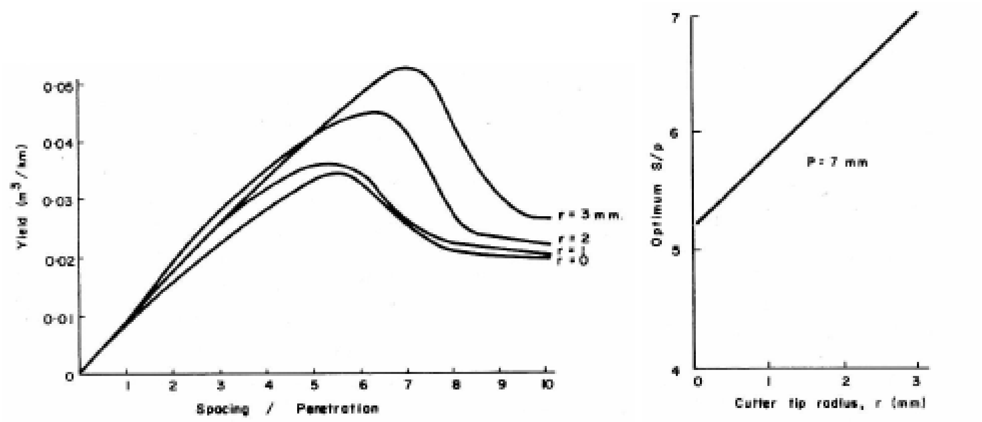


Figure 2.36 Effect of tip radius on optimum s/p ratio
(Source: Phillips, Bilgin and Price, 1978)

It was realised that there is a downside to introducing a tip radius and using fewer discs. While the overall thrust on the cutter head will not change significantly, the thrust per disc will increase. Analysis of the data showed that at a penetration of 7mm each tool will require about 22% additional thrust. Since bearing failure was, and still is, problematic for the smooth operation of a continuous tunnel boring process, it was considered appropriate to investigate the performance of stalled discs (Roxborough and Phillips, 1981a).

The experiment, again in Bunter Sandstone, involved using a 200mm, 60º edge angle disc penetrating to depths of 2, 4, 6, 8 and 10mm and operating in both the unrelieved mode and at a s/p ratio of 5. A sample of the results are shown in Figure 2.37.

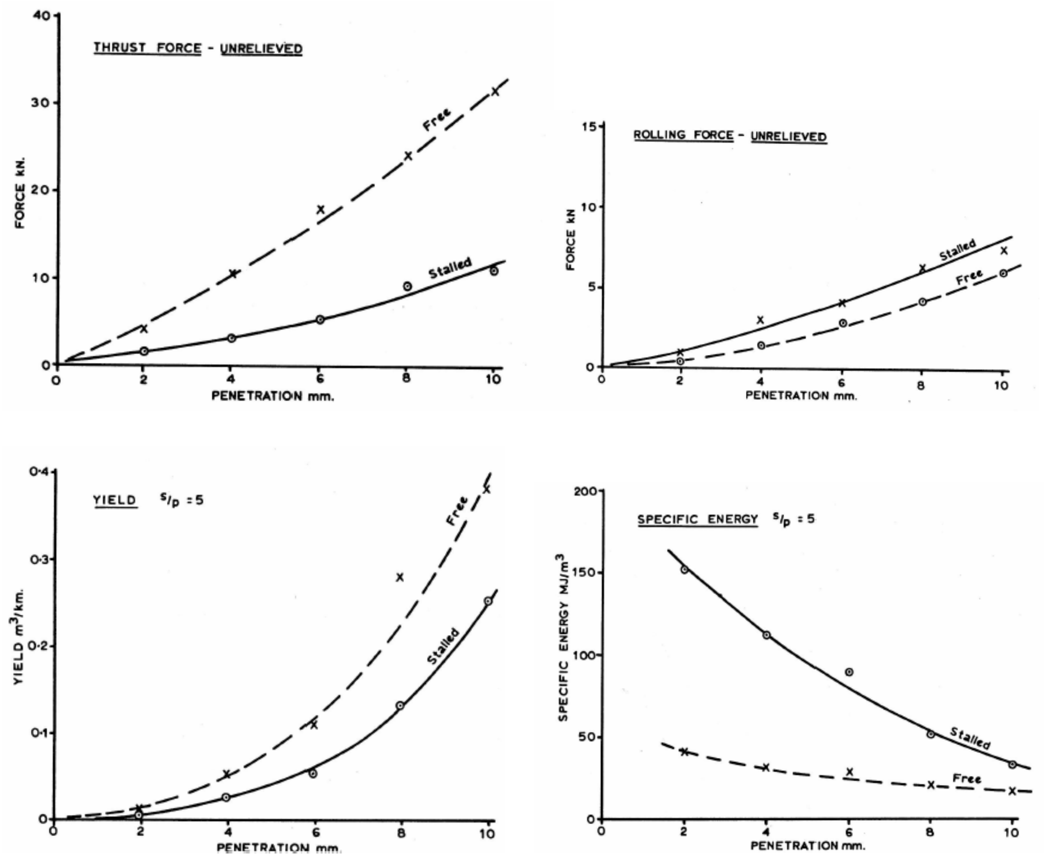


Figure 2.37 The effect of penetration on free rolling and stalled discs (Roxborough and Phillips, 1981a)

Although all forces increase with penetration, the thrust requirement for a stalled disc is considerably lower than that for the free rolling condition. Conversely the rolling force is significantly higher with the disc stalled. This leads to the conclusion that the behaviour of a stalled disc resembles that of a drag pick. Indeed, when stalled, a disc can be likened to a pick having a vee-front face of negative, varying rake angle. By any standard this is a highly complex shape for a pick.

Extending this drag pick analogy, thrust force is equivalent to pick normal force and rolling force becomes cutting force. Thrust force, when translated into a normal force, can be expected to fall because the edge presented to the rock surface is not now required to be forced down into the surface. As a normal force, it is merely required to hold the "pick" at its prescribed depth of cut. This force will of course be higher than for a conventional pick due to the negative curved clearance, which will create a greater tendency for the tool to lift out of the rock.

Slabbing between adjacent grooves is a feature of free rolling discs, being attributable to the high component of thrust mobilised in the lateral direction. When stalled, however, and thereby behaving more like a pick, the disc is unable to generate such a potent, sideways influence and lateral breakthrough was seen to be inhibited. In effect the most productive of the force components has been suppressed, if not lost. Figure 2.37 also shows results obtained when the penetration to spacing ratio was 5. This s/p ratio was chosen as a value at which breakthrough of the intervening rock between adjacent discs could be guaranteed for free rolling discs. The fall in yield when the disc stalls, together with a higher rolling force, resulted in significantly higher energy requirements once a disc has stalled. The paper (Roxborough and Phillips, 1981a) concluded with calculations based on three realistic conditions. The machine torque and thrust requirements were calculated and compared for a 4m diameter cutter head with a) all cutters free rolling, b) the outermost triple disc assembly having failed and c) a further triple disc assembly at mid-radius also failed. In effect this took the research out of the laboratory and into the field. The calculations showed that the collape of only one triple disc assembly in a typical array would produce a significant increase in torque requirement. An experienced machine operator should, therefore, be able to detect a disc failure using conventional machine instrumentation.

My final contribution to disc cutting research was to participate in the design and to oversee the construction of a large, multi-tool rig capable of examining the performance of an array of discs. This test rig, nicknamed “Big Jim” was comissioned just before my departure for Australia in 1977 and was used for on-going research for many years (Figure 2.38).

2.5 Collaborative research with the Turkish coal industry

In 1989/90 Professor Nuh Bilgin, Head of the Department of Mining Engineering at Istanbul Technical University spent a period of sabbatical leave at the Department of Mining Engineering at Wits. Because of his interest in mine mechanisation he arranged for coal samples to be sent from Turkey for testing. This was the start of a collaboration that lasted for 5 years and led to a paper (Bigin, Phillips and Yavuz, 1992) and a chapter in a book (Bigin and Phillips, 1995). By using the results of coal ploughing simulations undertaken during my M.Sc. research (Phillips, 1971) and scaling these results to the mechanical properties and cutting characteristics of coal from the Eli Darkale coal mine in Turkey, Professor Bilgin and I were able to advise the mine on the most appropriate settings for their newly installed coal plough. Bilgin and I prepared the manuscript of a paper and it was translated into Turkish by Yavuz, who was also involved with the installation of this plough face at Eli Darkale colliery.

Following completion of this research and its publication Professor Bigin and I were invited to contribute a chapter on mechanical properties of coal in a book published in 1995. This looked at the petrography of coal (Falcon and Falcon, 1987) before examining a wide range of commonly used tests and indices of strength and abrasivity. The chapter concluded with an analysis of the relationship between the results of these tests and the cuttability of coal. An example of tests conducted for the Aegean lignite mine illustrated how the common indices could be used to predict the cutting characteristics of a coal seam and hence the likely success in applying a coal mining machine (Bilgin and Phillips, 1995).

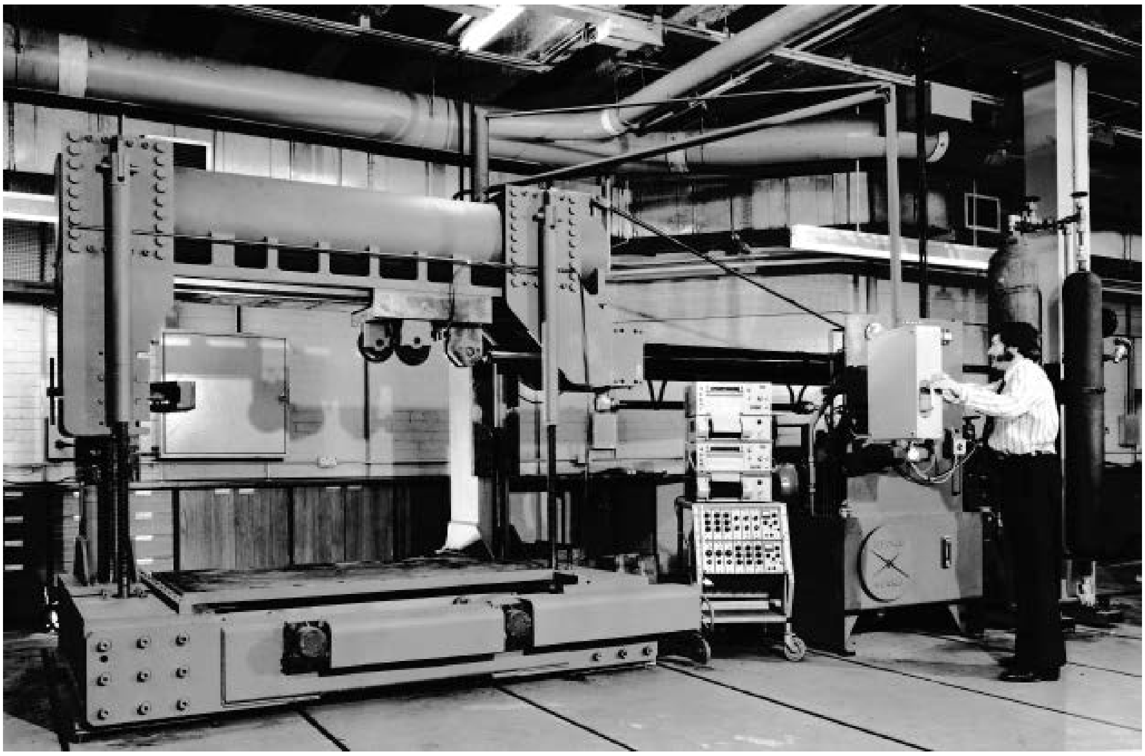


Figure 2.38 “Big Jim” and myself during commissioning trials

CHAPTER 3 SPONTANEOUS COMBUSTION RESEARCH

Spontaneous heating of coal occurs when its temperature rises due to exothermic internal reaction, mostly as a result of oxidation. If the heat produced by this low temperature reaction cannot be dissipated by either conduction or convection the temperature will continue to rise. As the temperature increases so does the rate of oxidation, resulting in ever increasing temperature until auto-ignition occurs. This tendency for self-heating of coal decreases with increasing rank of the coal, i.e. lignite coals are more prone to self-heating than bituminous coal, while the phenomenon rarely occurs in anthracite. Carbonaceous shales found in proximity to coal seams are often extremely reactive.

Spontaneous combustion has been a major hazard in underground coal mining for at least four centuries, with one of the first recorded incidents being as early as 1604 (Morris, 1986a). From then onwards there has been ongoing research interest in the spontaneous combustion of coal and the huge amount of literature has been reviewed on many occasions, e.g. Wade (1988), Chakravorty and Kolada (1988) and Eroglu (1992). From the extensive observations reported in the literature the critical factors contributing to the risk of spontaneous combustion have been grouped into intrinsic, i.e. those that cannot be controlled (coal properties and geological features) and extrinsic, i.e. those that can be controlled (mining practices). These factors are listed in Table 3.1.

Table 3.1 Critical factors contributing to spontaneous combustion
(after Chakravorty and Kolada (1988)

Coal Properties	Geological Features	Mining Practices
High volatile matter	Thick seams	Leaving roof and floor coal
High moisture	Presence of inferior pyrite bands and carbonaceous shale	Poor maintenance of roadways and old districts
High pyrites	Presence of faults	Inadequate measures to prevent air leakage through air crossings, doors,etc.
High exinite	Weak and disturbed strata conditions	Caving to surface under shallow overburden
High friability	High strata temperature	Close proximity of multi-seam workings
		Poor ventilation management

Despite the voluminous research, including work undertaken or being undertaken in the late 1980s in South Africa (Falcon, 1986), (Glasser, 1990) and (Farquharson, 1988) and a better understanding of its causes, incidents of spontaneous combustion continue to occur in underground coal mines in many parts of the world. Where seams are gassy there is always the potential for spontaneous combustion to lead to disastrous explosions, such as that experienced at Moura #2 coal mine in 1994 (Windridge et al, 1995). However, methods of detecting heatings have been developed and improved over the years and usually provide an early warning before auto-ignition occurs.

In South Africa Morris and Badenhorst reported that, in the period 1970 to 1985, 169 underground coal mine fires occurred, including 70 which were initiated by spontaneous combustion, i.e. 41% of the total (see Table 3.2 taken from Morris and Badenhorst, 1987). All underground fires are hazardous because of the toxicity of the gases released, the consumption of oxygen leads to an asphyxiating atmosphere and there is always the danger of a gas explosion leading to a coal dust explosion. In addition, an underground fire almost always results in a loss of reserves.

Table 3.2 Causes of coal mine fires 1970 – 1985
(after Morris and Badenhorst, 1987)

	Total	Spontaneous Combustion	Electricity	Methane	Welding	Others	Unknown
1970	10	7	3	-	-	-	-
1971	3	2	-	1	-	-	-
1972	8	3	1	2	-	2	-
1973	11	6	1	2	-	1	1
1974	8	6	1	-	1	-	-
1975	8	6	1	-	-	1	-
1976	5	3	2	-	-	-	-
1977	13	7	2	1	1	1	1
1978	8	3	4	1	-	-	-
1979	14	9	5	-	-	-	-
1980	6	2	2	2	-	-	-
1981	13	4	3	5	-	1	-
1982	13	2	10	-	1	-	-
1983	8	1	4	2	-	1	-
1984	17	5	6	5	-	1	-
1985	24	4	10	5	-	5	-
	169	70	55	26	3	13	2

During the period 1975 to 1985 the South African coal industry invested well over R1 500 million in expanding production capacity, much of it on underground equipment. While the risk of explosions caused by spontaneous combustion in the low methane regime of the Witbank/Highveld coalfields was seen as small, in each fire incident there was the distinct chance to lose expensive equipment. As a consequence there was a demand from the industry to quantify the risk at any particular mine. This was driven in part by the fact that the frequency of occurrence of spontaneous combustion instances varied very significantly across the country. Morris and Badenhorst (1987) indicated which coalfields contributed to the total number of heatings and Wade (1988) related these figures to production in those regions of the country. Table 3.3 gives a clear indication of the disparities according to region.

Table 3.3 Distribution of self-heating incidents on a regional basis

Region	Heatings	Production 10^6 tons	10^6 tons mined/ heating
Vaal Triangle	38	210.3	5.53
KwaZulu-Natal	25	206.2	8.25
Witbank/Ermelo	7	1387.3	198.19
Total	70	1803.8	14.34

The data in Table 3.3 indicates that KwaNatal and the Vaal Triangle had a heating incidence broadly 24 to 36 times greater than the Witbank/Ermelo region. In order to quantify this risk more accurately and on a mine by mine basis, the Explosion Hazard Advisory Committee, a sub-committee of the Coal Mines Research Controlling Council requested that the Department of Mining Engineering at the University of the Witwatersrand develop suitable test apparatus and a test protocol for determining the propensity of coal samples for spontaneous combustion and to make this work available to the coal industry. The laboratory work commenced in 1985 and, under my supervision, research resulted in the award of three Ph.D. degrees (Wade, 1988), (Eroglu, 1992) and (Gouws, 1993), together with the successful completion of two M.Sc. dissertations (Gouws, 1987) and (Uludag, 2001) also under my supervision.

3.1 The choice of a spontaneous combustion testing procedure

A thorough review of the literature revealed that research workers had used six principle techniques in order to establish the propensity of different coals to spontaneously combust. These were :-

1. direct observations
2. ignition temperature determinations
3. adiabatic calorimetry
4. isothermal calorimetry
5. chemical techniques
6. oxygen sorption measurements

The direct observation of spontaneous combustion events relies on them occurring but, more importantly, there are so many variables to be considered that this technique, although useful, is not suitable for producing an index.

The literature concerning the use of chemical techniques shows that a few researchers used the adsorption of halogens (chlorine, bromine, iodine and astatine) by coal samples as a simple and rapid test for reactivity. However, later work failed to show agreement between the results of iodine adsorption tests and observations of rates of oxidation.

Once these two tests had been eliminated the advantages and disadvantages of the remaining four techniques were listed, as shown in Table 3.4 (Wade, Phillips and Gouws, 1987a). At this time research into spontaneous combustion was also being undertaken in the Department of Chemical Engineering at Wits. This work, which was subsequently published (Glasser, 1990) used the oxygen sorption technique. The apparatus he used automatically measured the rates of oxygen sorption of twelve coal samples at a time under temperature controlled conditions. Small samples of coal were located in a sealed flask of oxygen. As the oxygen was absorbed a vacuum was created, drawing oil into the flask. Regular weighing of the flask, by a computerised automatic system, allowed the volume of oxygen adsorbed per unit time to be calculated.

While it appeared attractive to use the same technique and share data, oxygen sorption at constant temperature cannot give any indication of how the reaction speeds up as temperature increases and this method was eliminated. For the same reason isothermal calorimetry was rejected. Of the two remaining methods the ignition temperature technique was seen as the easiest way to obtain results (Wade, Gouws and Phillips, 1987b) but there was also a decision that an adiabatic calorimeter should be built at a later stage (Gouws, Gibbon, Wade and Phillips, 1991).

Table 3.4 Comparison of spontaneous combustion tests
(after Wade, Phillips and Gouws, 1987a)

Method	Advantages	Disadvantages
Ignition temperatures	• rapid tests • simple, cheap apparatus	• crossing point artificial • difficult to recognise “ignition” • insulation characteristics not similar to practical situation
Adiabatic calorimetry	• duplicates practical insulation characteristics	• lengthy tests • apparatus costly, difficult to make and calibrate
Isothermal calorimetry	• negates variability of rate of reaction with temperature	• doesn’t consider self-magnifying effect of increased temperature • difficult to maintain true isothermal conditions
Oxygen sorption	• simple, cheap apparatus • multiple simultaneous tests are possible	• rate of oxygen consumption <u>assumed</u> indicative of heat generation

3.2 Development of apparatus for determining ignition temperatures

Within the overall ignition temperature method of determining the propensity of coal to spontaneously combust, there are two tests that can be performed using the same apparatus. Firstly, a sample of crushed coal is placed in a container while an identical mass of inert material is placed in a second, identical container. Both are gradually heated in an oil bath and initially the coal temperature lags behind the temperature of the inert material due to the cooling effect of the evaporation of moisture inherent in the coal. Once all the moisture has evaporated the coal temperature increases at a rate faster than that of the inert. This is due to the exothermic oxidation of the coal. The crossing-point is the temperature where the coal and the inert material are at the same temperature. The crossing point temperature is assumed to indicate how prone the coal is to spontaneous combustion, i.e. the lower the crossing point, the more reactive the coal. A second test using the same apparatus involves Differential Thermal Analysis (DTA), which is a technique for studying the heat generated due to coal oxidation (or heat absorbed due to evaporation of moisture) at different temperatures. DTA provides a dynamic study of the coal's thermal behaviour, as opposed to the single empirical value obtained from cross-point temperature testing. In doing so a graph is produced of the differences between the coal and the inert material temperature versus the temperature of the inert material. This is called a coal thermogram and typically has three stages. The initial stage depicts endothermicity due to the evaporation of inherent moisture. Stage II commences from the lowest negative differential and represents predominantly exothermic reactions taking place as the coal oxidises. A point is reached where the thermogram slope increases rapidly into Stage III, at which point the coal is on the verge of ignition. The steeper the gradient of stage II and the lower the transition from Stage II to Stage III the more liable the coal to spontaneous combustion

Both crossing-point temperature and DTA concepts are illustrated in Figure 3.1.

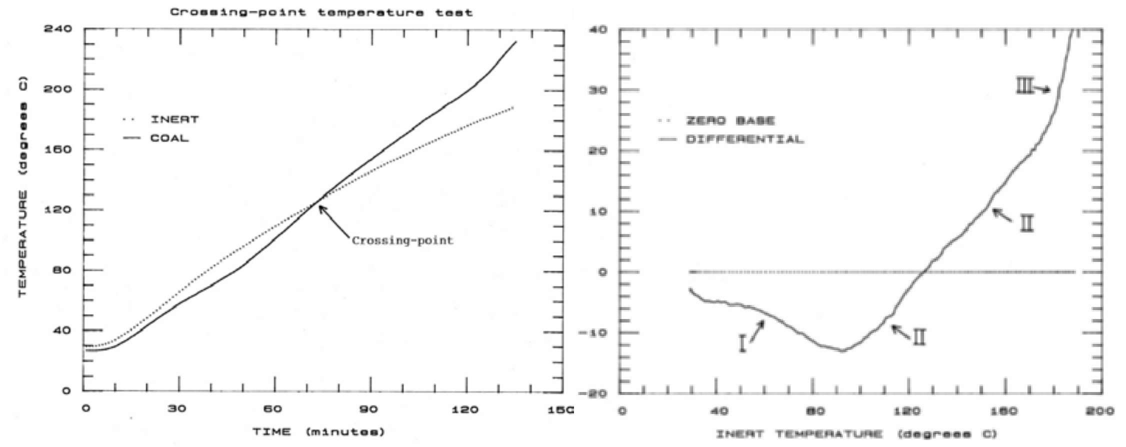


Figure 3.1 Crossing-point test and coal thermogram
(Source: Wade, Phillips and Gouws, 1987b)

After reviewing the literature, cells were designed and manufactured in the workshop of the Department of Mining Engineering. The six cells were to be immersed in an oil bath equipped with a thermostatically controlled heater/ recirculating pump. In order to improve contact between the cells and the hot oil they were ribbed, as shown in Figure 3.2, and to improve accuracy the inlet air was heated and platinum resistance thermometers used for temperature measurements.

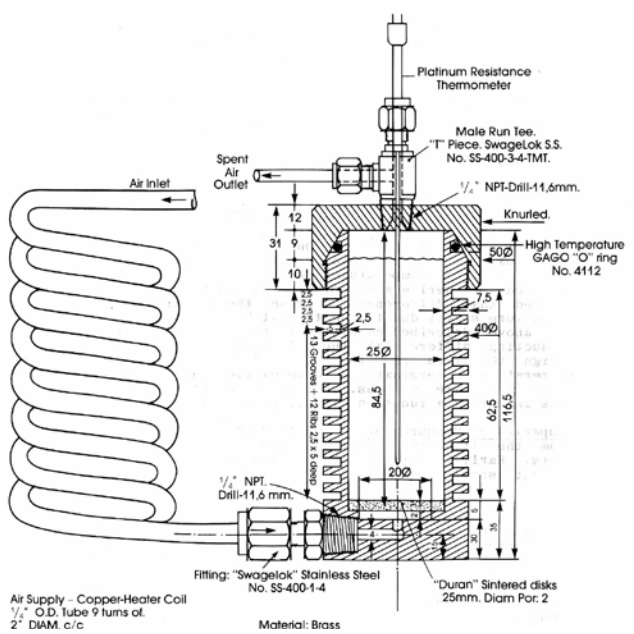


Figure 3.2 Design of sample cells
(Source: Wade, Phillips and Gouws, 1987)

Five coal samples plus a sample of calcined alumina (inert material) could be tested simultaneously with the six cells attached to a manifold before immersion in the oil bath, see Figure 3.3. Full details of the test apparatus are given in Wade, Phillips and Gouws, 1987b.

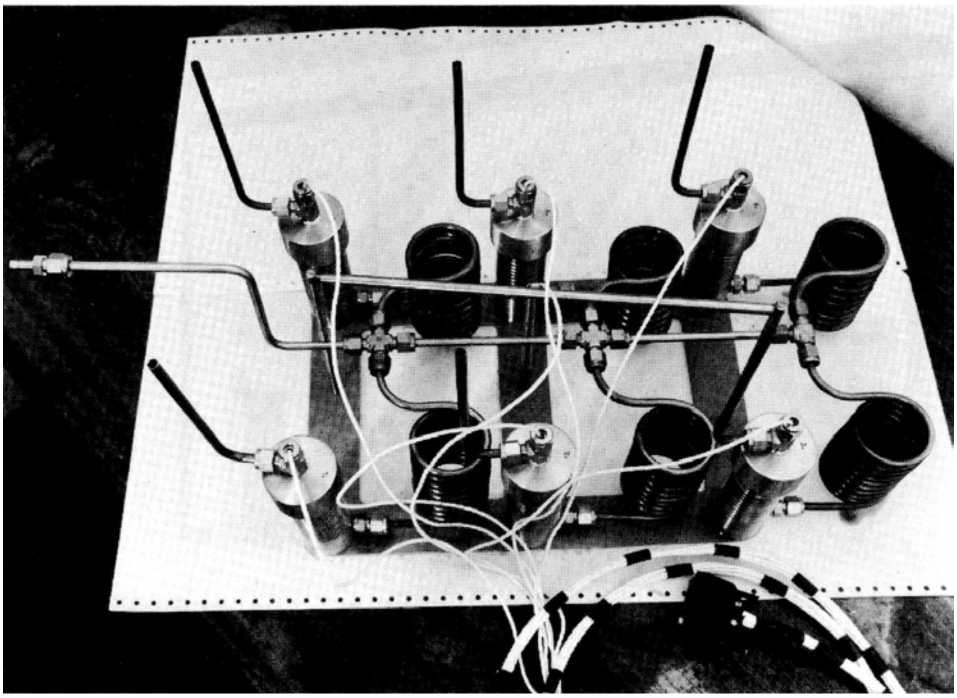


Figure 3.3 Six sample cells mounted in manifold
(Source: Wade, Phillips and Gouws, 1987)

3.3 The Wits-EHAC index

While the two tests described above yielded valuable information, the mandate from the Explosion Hazards Advisory Committee was to produce a single index to allow mines to assess their risk of spontaneous combustion.

Two indices proposed by previous researchers were considered, with data from our early experiments used to evaluate them. These were the FCC index (Feng, Chakravorty and Cochrane, 1973) and the MR index (Mahadevan and Ramlu, 1985).

The FCC Index was defined as:

$$\text{Index} = \frac{\text{average heating rate between 110 and } 220^0\text{C}}{\text{relative ignition temperature}} \text{ x } 1000$$

The choice of 110ºC as a bottom limit was to ensure all moisture had evaporated, while the 220ºC was to ensure the coal contained all its volatile matter and none had been driven off. When examined against the results for South African coals two problems emerged. Firstly, several samples had crossing-points below 110ºC and secondly, all transitions from stage 2 to stage 3 occurred below 220ºC. This excluded the FCC Index from consideration.

In order to eliminate the arbitrary selection of temperature ranges the MR Index was based on a segmented approach, as shown in Figure 3.4.

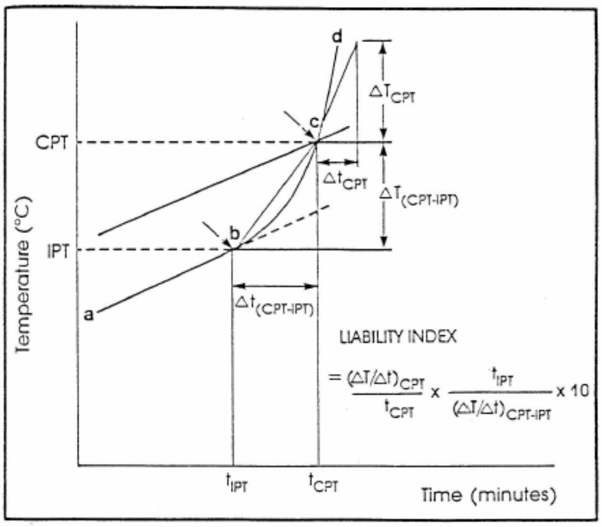


Figure 3.4 Heating curve analysis
(Source: Mahadevan and Ramlu, 1985).

The segments used by Mahadevan and Ramlu were :-

a-b, from the commencement of the experiment to the point where internal heating is first noticeable
b-c, from the inflection point to the crossing-point. The comparable section of the thermogram shown in Figure 3.1 is the slope of the initial part of Stage II.

c-d, from the crossing-point temperature to the point of active combustion. This corresponds to the upper portion of Stage II in a thermogram.

Relating the MR Index to the thermogram in Figure 3.1 gives

$$\text{MR INDEX} = \frac{\text{Upper Stage II slope}}{\text{X. P. T.}} \text{ x } \frac{\text{Inert temp. @ max. −ve differential}}{\text{Lower Stage II slope}} \text{ x } 10$$

When analysed using experimental data, this index proved to be nonsensical. In particular, Mahadevan and Ramlu state “the index decreases with increasing liability” which was directly contradicted by our data and is also counter-intuitive.

The exclusion of these two indices caused the research group to look for our own index and this work was led under my supervision by M.J. Gouws, a postgraduate student (Gouws, 1987). Working on the premise that crossing-point temperatures and Stage II slopes are independent predictors of spontaneous heatings, Gouws used these two characteristics as the axis on a graph, as shown in Figure 3.5. By plotting the inverse of the crossing-point temperature (X.P.T.) on the y-axis, the Stage II slope on the x-axis and joining the two points a triangular area is enclosed. This area now takes into account the two parameters measured in the test protocols used at Wits

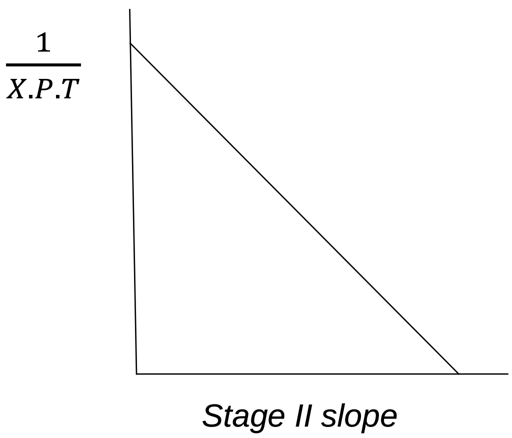


Figure 3.5 Triangle formed by joining the two characteristic values of a coal (after Gouws, 1987)

Using simple geometry this area is calculated as

$$\text{Area} = 0.5 * \text{Stage II slope} * \frac{1}{\text{X. P. T.}} * 1000$$

i.e. the Wits – EHAC index is $\frac{\text{Stage II Slope}}{\text{X.P.T.}} * \text{constant}$

This index was used by the research group for classifying the risk for coals from mines and coalfields across South Africa. The index correlated well with high values being obtained for coals known to be prone to spontaneous combustion. The test became available to industry as a service and thirty years after the apparatus was built, it is still in use with approximately 25 tests being undertaken each year on a commercial basis (Genc and Cook, 2013) and (Genc and Cook, 2015).

3.4 Adiabatic calorimetry

In 1990 the research group developed an adiabatic calorimeter to compare adiabatic incubation of spontaneous combustion with the results obtained using the cross-point temperature apparatus. The adiabatic calorimeter, together with the controlling and recording apparatus, is shown in Figure 3.6, with the arrow indicating the calorimeter. The development of this laboratory has been fully described (Gouws, Gibbon, Wade and Phillips, 1991). The calorimeter had three test options:-

1) an incubation test
2) a minimum self-heating test and
3) the crossing-point temperature test

Over a period of time the calorimeter provided useful results, which were compared with the results of proximate, ultimate and petrographic tests. The best correlations were adiabatic index, oxygen to carbon ratio and dry ash-free (daf) oxygen content. Despite the fact that the adiabatic calorimeter was fully automated, virtually maintenance free and required no consumable items, the mining industry declared its preference for the Wits-EHAC index and the apparatus was decommissioned at the end of the research programme.

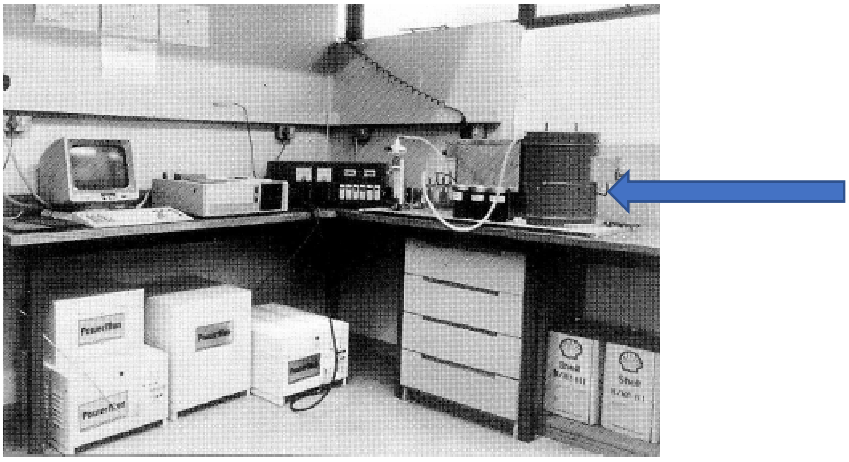


Figure 3.6 The adiabatic calorimeter

3.5 Mapping of spontaneous combustion risk across mines and coalfields

Once the Wits-EHAC index was accepted by the South African coal industry as a standard test, a regular flow of samples was received from the coal mines. One mine in particular, the Durban Navigation Colliery, had experienced several spontaneous heatings leading to the loss of reserves and equipment. Their request for multiple samples to be tested was an opportunity not just for commercial testing but also to research the possibility of contouring the spontaneous combustion risk across the lease area of a single mine (Eroglu, Gouws and Phillips, 1991). A Ph.D. student worked on this project and was awarded his degree in 1992 (Eroglu, 1992)

The Durban Navigation Colliery, situated in the Klip River coalfield, was first developed in 1902 to provide bunker coal for steamships visiting the port of Durban. In 1954 it was purchased from the original owners by ISCOR (the state owned Iron and Steel Corporation) to provide coal for the Newcastle steelworks some 30 km away. New shafts continued to be sunk until 1982. Two seams, the Top and Bottom seams were being mined in the late 1980s with a paring between them consisting of sandstones and carbonaceous mudstones varying from 1 to 10m in thickness across the mine. The Top seam averaged about 1.1m in thickness while the bottom seam was actually a composite consisting of two or three individual coal seams depending on its location and averaging about 1.5m in thickness.

As part of this research project 58 coal samples were collected from as wide a coverage of the accessible underground areas as possible. The location of each sample was recorded on the mine plans and, in addition to obtaining the Wits-EHAC index for each sample, they were all subjected to proximate, ultimate and petrographic analyses as detailed in Table 3.5. Thirty one of the coal samples were also subjected to tests in an adiabatic calorimeter.

Table 3.5 Analyses undertaken on samples from Durban Navigation Collieries

Proximate Analyses	Ultimate Analyses			Petrographic Analysis
Moisture content Ash content Volatile matter content Fixed carbon content	Phosphorus content Carbon content Hydrogen content Nitrogen content	Oxygen content Total sulphur Pyritic sulphur Sulphate sulphur	Organic sulphur Gross calorific value Roga index Swelling Index	Maceral analysis Microlithotype analysis Reflectance of vitrinite

The thrust of this research was to contour the Wits-EHAC index for each seam on the mine plans. This was done using an early version of SURFER software. Thereafter a transparency overlay was prepared showing the location of previous incidents of self-heating and/or spontaneous combustion. (It must be remembered that in 1992 digital mapping was very new and not in common use on coal mines).

The main conclusions of this research were:-

a) samples taken from the Top and Bottom seams had significantly different Wits-EHAC indices, with the Bottom seam results indicating a higher propensity for spontaneous combustion. This was in-line with the records of the number of incidents in each seam.
b) the contours of Wits-EHAC indices indicated high risk areas of the mine and there was good agreement between previous experience of spontaneous combustion events and these areas
c) the adiabatic calorimeter, when used in the incubation mode, confirmed the results of the Wits-EHAC index tests

d) attempts to correlate coal properties to the Wits-EHAC index were disappointing. This was attributed to the fact that all the coal samples collected had similar rank values. This resulted in a very limited range of results and subsequently poor regression co-efficients.

By the late 1990s a very large number of crossing-point tests had been conducted and compared with the coal properties of the samples. Most samples were from the Witbank coalfield, since it was known to have a rare but recurring spontaneous combustion problem. This is a large coalfield extending approximately 180 km east-west and 40km north-south and although a database existed it was extremely difficult to locate problem areas within the coalfield and to visualise the relationships existing between coal properties and spontaneous combustion risk. The use of overlying, manually produced transparencies for such large data sets was not an option. To address this problem a GIS tool (ILWIS) was purchased by the School of Mining Engineering and used to map the Witbank coalfield (Uludag, Phillips and Eroglu, 2001).

Management of spontaneous combustion depends largely on how information is handled in order to define and monitor problem areas. It was well known that there are two components to any geography dependent data; its geographic position and its attributes or properties, i.e. spatial data (where is it?) and attribute data (what is it?). The attribute data in this case was those factors that could affect spontaneous combustion.

The process this research followed was firstly to produce a digital locality map of the Witbank coalfield collieries using 15 sheets of 1:50 000 scale maps. This map is shown in Figure 3.7

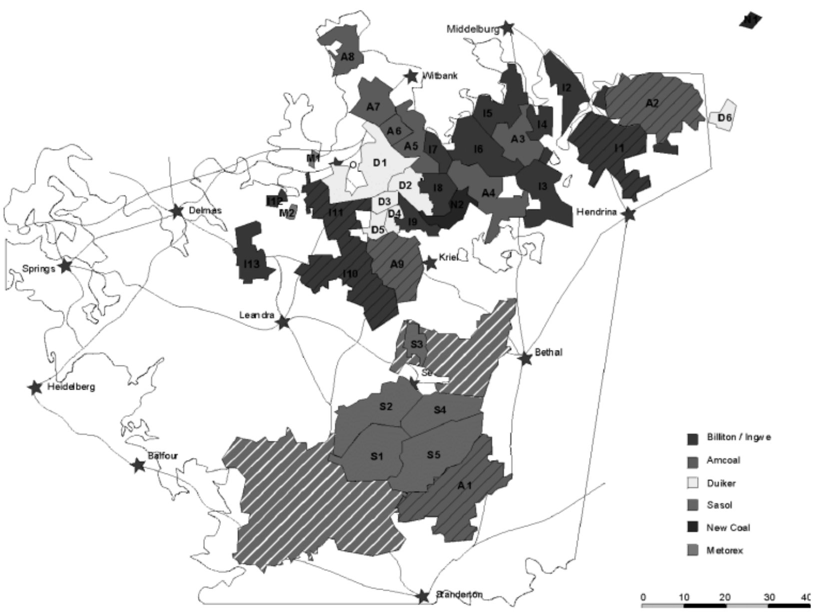


Figure 3.7 Locality map of Witbank coalfield collieries
(Source: Uludag, Phillips and Eroglu, 2001)

Digitised maps of three types were used: point maps (showing collieries or sample points), polygon maps for the boundaries of collieries and segment maps for topography. These maps can be overlaid on each other, or mathematical and statistical calculations can be performed.

Although Table 3.1 listed 16 critical factors, it was considered that the most significant were the coal properties and depth of the coal seams below surface. Depth was significant because in places it was so shallow that air was able to enter the underground workings through cracks running to surface. The point map of collieries was linked to an attribute table in which the relevant coal properties were listed in separate columns. For each attribute column a point map was created with a value domain. These point maps were rasterized in order to be able to make mathematical calculations. In a raster model spatial data is organised in grid cells or pixels, which are the basic unit for which information is explicitly recorded. A resolution of 200m by 200m per pixel was chosen, since this allowed the smallest item within the coalfield, such as a panel length to be seen.

Before the individual maps for each of the coal factors and depth can be combined, the risk posed by each factor had to be assessed. In doing so the literature was examined and practical experience of spontaneous combustion in South Africa evaluated. In particular, the work of Singh et al (1984) was used in determining the rating for each factor and these were used to create weighted maps. These were simply added to obtain an overall risk map of the Witbank coalfield as shown in Figure 3.8. In this map the light colours represent the higher risk areas.

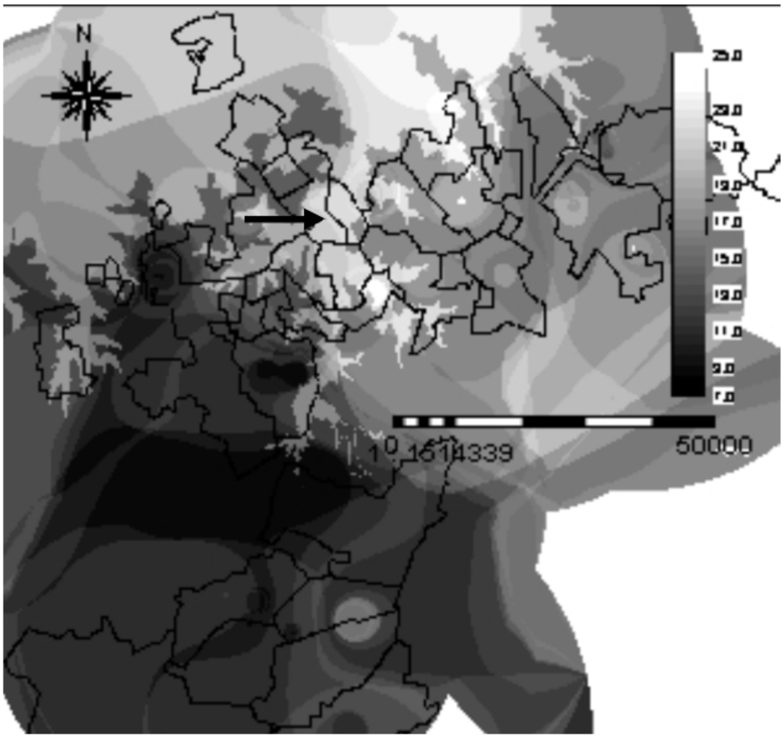


Figure 3.8 Combined risk map of the Witbank coalfield
(Source: Uludag, Phillips and Eroglu, 2001)

As an example of comparing research results with practice, the arrow in Figure 3.8 points to a colliery in the high risk area and it is known that this colliery has almost continuously experienced serious spontaneous combustion problems. In fact this colliery practiced opencast mining of old pillars left from previous underground operations and this topic is discussed in Section 3.6.3. It was the subject of research funded by Coaltech (Phillips, Uludag and Chabedi, 2010).

The application of GIS for environmental risk assessment allows all kinds of information to be brought together and visualised. The paper (Uludag, Phillips and Eroglu, 2001) demonstrated that individual factors known to affect the propensity for spontaneous combustion can be combined graphically to provide a risk map for either a single mine or an entire coalfield.

3.6 Three practical spontaneous combustion investigations

While the research described so far in this chapter could be described as "fundamental research investigating some of the influences leading to spontaneous combustion", during the same time other, personal effort was directed at solving urgent, practical problems. Three of these are discussed below

3.6.1 Fires in shallow abandoned mines

Coal has been mined extensively in the Witbank area since 1890 and it should not be surprising that Witbank was re-named eMalahleni (the place of coal) in 2006. Historically the near surface coal seams, sometimes as shallow as 10m, were mined by room and pillar methods, which left a minimum of 20% of the reserve in-situ. Spontaneous combustion of these pillars, together with any overlying top coal or carbonaceous shale has created a severe problem in the suburbs of Witbank, with underground fires burning for decades.

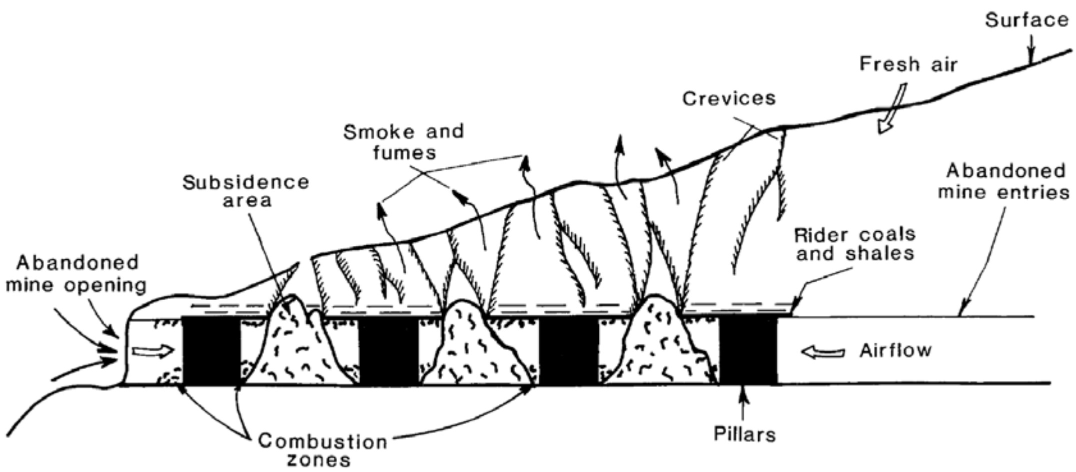


Figure 3.9 Fire(s) in shallow mines
(Source: Kim and Chaiken, 1993)

As can be seen from Figure 3.9 the traditional fire-fighting method of excluding oxygen from the affected area is generally ineffective since air can easily penetrate shallow workings. In addition, the effects of the fire can cause severe environmental damage. Surface subsidence due to collapse of burnt out areas of the workings can be extensive, while heat and toxic gases reaching the surface are an obvious threat to surface vegetation, crops, buildings and people within the vicinity. The growing urbanisation of the South African population in the 1980s and the creation of informal settlements in close proximity to the fire areas resulted not only in a concern for the health of those affected but also in a significant number of disappearances of persons assumed to have fallen into burning sink holes. Figure 3.10 illustrates the surface instability caused by an underground fire.

Figure 3.10 Partial collapse of a burning bord and pillar intersection (Photograph taken by Phillips)

In 1986 one of the underground fires close to a suburb of Witbank threatened the coal pillar left under the main railway line from Pretoria to the port of Maputo. This resulted in the whole area being declared a disaster under the Civil Defence Act and I was invited to join the four person Advisory Group set up by the responsible Minister. An urgent report was prepared in four weeks and this prompted the Coal Mining Research Controlling Council to approve funding for research into fires in shallow workings. This work was initially undertaken by the Department of Mining Engineering at the University of the Witwatersrand and then jointly with the Coal Mining Laboratory of the Chamber of Mines Research Laboratory. This research was intended to identify or develop methods of detecting subsurface heatings, to monitor underground conditions and to evaluate alternative methods for extinguishing or controlling these fires. Several reports were produced detailing the results of this research and a summary paper published (Phillips and Sullivan, 1991). Some of the fires investigated were thought to have been ignited by an extension of surface fires into the workings but there was plenty of evidence that others were the result of spontaneous combustion.

Spontaneous combustion or self-heating occurs when organic material oxidises under conditions which inhibit the removal of heat from the process. Shallow abandoned mines with fissures extending to surface and mines with multiple openings at the outcrop are obviously prone to this problem since sluggish airflow will take place. Furthermore, the heat generated is likely to accumulate more rapidly than it dissipates because the coal and surrounding areas absorb radiant heat, while at the same time are poor conductors. In fact, the only significant heat loss following the initiation of combustion is by convection currents through surface openings and crevices. Although they remove heat these convection currents occur late in the heating process and are likely to increase its severity by drawing fresh air into the fire area.

Fires in the shallow workings surrounding Witbank were most often detected because of physical indications on surface – loss of vegetation, smoke or hot gases rising from surface fissures or, in extreme cases, surface collapse with the formation of sink-holes in which open flames may be observed. Once observed it is important that the centre of the fire is detected and its progress monitored. Two techniques tried with limited success were infra-red aerial photography and the collection of gas samples. However, the most effective method of monitoring the heating threatening the railway line was the monitoring of temperatures in boreholes, which were sealed between the times when monitoring was in progress to ensure no additional air became available in the fire zone (Phillips and Sullivan, 1991).

The urgent need to protect the railway line resulted in both a literature search and contact with known authorities in several countries. Five possible solutions were considered

1) Surface sealing
2) Inertisation using selected gases
3) Flushing of fine inert material and water into the workings
4) Direct attack using water
5) Containment by constructing a barrier

If surface sealing is attempted all openings that are accessible from surface are closed and sealed and cracks or fissures covered with sand or soil. If the area is extensive, a seal can be established by ploughing the surface to a depth of 1.5 to 2m. This would entail deploying heavy equipment over fire areas, such as those illustrated in Figure 3.10. The risk to life was considered too high to contemplate using this method.

Inertisation of the mine atmosphere using gases such as nitrogen, carbon dioxide or flue gas is a well proven technique for dealing with mine fires. However, in shallow abandoned workings there are so many leakage paths to surface that this technique could not be expected to succeed.

Flushing is a method designed to fill any voids in the underground area and has the dual effect of minimizing air flow and providing additional support to prevent any further caving to surface. However, this inert fill will only reach the roof of the workings leaving top coal and carbonaceous shales to burn above the mined seam. Considered opinion is that flushing will slow down the progress of the fire but will not extinguish it.

The traditional method of dealing with surface fires is to spray them with large volumes of water, thereby excluding oxygen and cooling the fire simultaneously. In the case of underground fires it is also possible to use water to inundate or completely saturate the fire area. However, water merely sprayed onto large fires, as shown in Figure 3.11 is unlikely to be effective

Figure 3.11 Full collapse of an intersection and an attempt to control an underground fire using water (Photograph taken by Phillips)

One limitation in completely flooding a mined out area is the consideration of whether the seam outcrops above the water table. Leakage from the outcrop, ineffective seals and the high permeability of fractured pillars make it extremely difficult to raise the water level in the workings high enough to extinguish the fire. Furthermore it has been shown in Table 3.1 that one of the critical factors in the development of spontaneous combustion is when moisture content changes. Even if the fire is extinguished there is a strong possibility it will re-ignite once pumping ceases and the water level starts to drop.

In the Witbank area the flooding of abandoned mines has been attempted on a number of occasions with limited success. When it is considered that the water leaking from a fire area typically has a pH value of approximately 2, the benefits of flooding a mine for fire-fighting purposes must be carefully evaluated against the environmental damage elsewhere as this water leaks out from the affected area.

The final method considered was containment, which involves the construction of a barrier to isolate the burning area from the surrounding fuel source. The barrier must break the continuity of both the coal and carbonaceous shale and be wide enough to prevent the transfer of heat from the fire area to susceptible materials on the inactive side of itself.

In seeking a solution that would extinguish the fire threatening the railway line all 5 methods were considered but not one was considered to provide the guaranteed protection required at an acceptable cost. A combination of methods was finally chosen and a containment trench dug for nearly 2 km using a small dragline and the area on the fire side of the trench was flooded. The design of the containment trench is shown in Figure 3.12

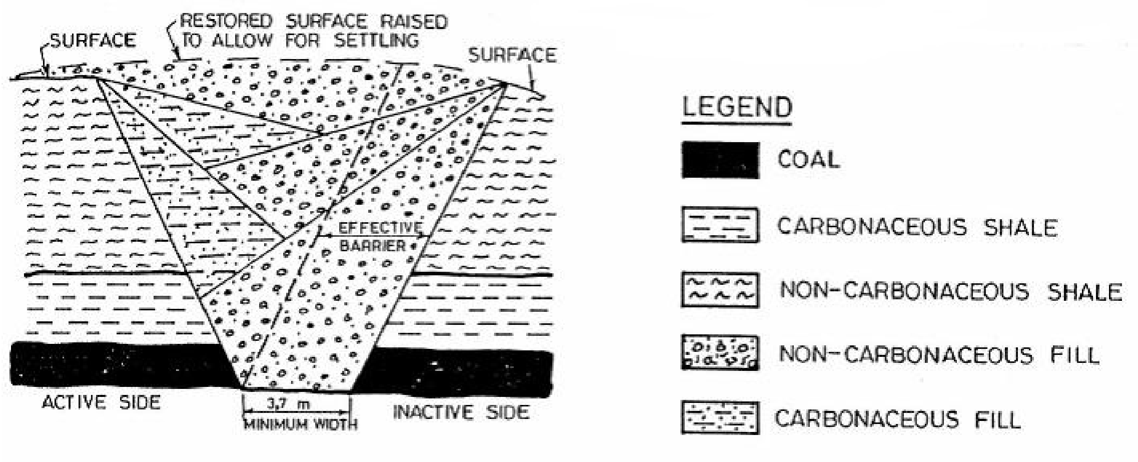


Figure 3.12 Selective placement of excavated materials in an open trench (Source: Phillips and Sullivan, 1991)

The two methods, in combination, proved successful in this instance and continued monitoring showed a steady decline in borehole temperatures over a number of years, with no indication of the “hot” area moving towards the railway. However, two extremely important questions remain unanswered to this day. Firstly, as communities expand due to urbanisation, there is increasing need for industrial and housing development to take place on undermined land. After 120 years of intensive mining of the Witbank coalfield vast tracts of land have been undermined and abandoned. The question is then asked how can we assess the potential for heatings in any particular area and, if they occur, how can they be detected at an early stage before strata instability and poisonous gases affect the safety and health of the population. The second question relates to the cost of dealing with a fire and who should be responsible. In many cases the surface rights are no longer held by the company which mined the property. Since the costs of firefighting are very significant (up to R100 million is not uncommon), with claims for compensation adding to the complexity of the situation, the situation in the eMalahleni (Witbank) area will remain problematic for decades to come.

3.6.2 Disposal of waste material at Grootegeluk coal mine.

In 1980 ISCOR, the state owned Iron and Steel Corporation, opened the Grootegeluk open-pit coal mine in the Waterberg coalfield to supply metallurgical coal for its own use and coal to the nearby Matimba power station. The Waterberg coalfield is situated in the western part of the Limpopo province of South Africa and has an approximately 88km east-west strike length, complemented by a north-south width of about 40km. Although this presents a small foot-print compared to other South African coalfields, the resource is many times that of any of the other coalfields due to its large equivalent seam thickness. The coalfield still has not been fully explored and this accounts for an estimated resource of 55 billion tonnes and the 2011 reserves of only 6.7 billion tonnes (Chabedi and Phillips, 2012). The Waterberg coalfield is significantly different to other coalfields in South Africa, with a package of 11 coal seams over a stratigraphic thickness of at least 110m.

By the year 2000 the Grootegeluk mine was mining a total of 54 Mt per year of coal and waste rock. From this total 34 Mt of raw coal was sent to the five coal preparation plants and, with an yield of only 50%, this resulted in just over 16 Mt of saleable product. In other words, over 37 Mt per year of plant discards and inter-burden waste was being stacked on discard dumps. These were prone to spontaneous combustion and were clad with a layer of clay. Unfortunately the harsh weather conditions at the mine, including a wide temperature range, rain and wind all caused erosion of the clay and despite constant maintenance these dumps constituted an environmental hazard.

After visiting the mine and reviewing their previous research, I started to work with the pit superintendent, who registered to undertake a PhD under my supervision (Adamski, 2003). My role in this research was to ensure academic rigour, while Adamski conducted the research and senior mine management assessed the risks involved in implementing the recommendations.

Adamski had observed the huge variations in reactivity of different benches being mined in the pit and was aware of the different times to ignition of the different surface dumps. He was also concerned with the costs involved in hauling inter-burden waste from the lower levels of the pit to the surface dumps. However, it was thought too risky to consider backfilling any of this material in the pit bottom. In a discussion with Professor Glasser of the Department of Chemical Engineering at the University of the Witwatersrand, the concept of using a column of reactive material to obtain a more realistic assessment of oxygen depletion rates, particularly as a function of depth and reactivity of the material, was considered. It was felt that with a highly reactive material oxygen depletion could occur at such shallow depth in a dump that heat dissipation could be complete and temperature increase could be avoided. In a very low reactive material the slow rate of heat generation, even deep in the pile of material, would also prevent spontaneous combustion (Adamski, 2003). With this as a background, a study to determine whether backfilling could be an option commenced in 2000.

The basic conditions to be met before backfilling could be considered were:-

" 1) *Minimal Risk* of spontaneous combustion, local ignitions….. with minimum influence of pollution on the pit's workers and environment.

2) *Minimal cost* including an implied cost reduction for trucking waste

3) *Maximum safety* for workers involved in backfill operations and all other workers at the mine

4) *Uncomplicated technology and a simple code of practice.* This should take cognisance of the skills of the workforce" (Adamski, 2003)

In order to meet condition 1 above, a research programme was initiated. As well as a thorough literature search and review of 20 years of research by Glasser, Ren and others (Glasser, 1990) and (Ren, Denby and Herbert, 1996) there was the need for additional laboratory work. Samples of each horizon within the pit, together with discard from the five preparation plants was sent for both the Glasser oxygen absorption test and the Wits-EHAC test. However, there was the need for additional work which was undertaken by Adamski. This included the column test programme.

Figure 3.13 shows one 0.5m section of the column and 6 sections combined to make a 3m high column, with gas sampling ports. The column test was developed to determine how far oxygen drawn from the top surface would penetrate material of different reactivity profiles.

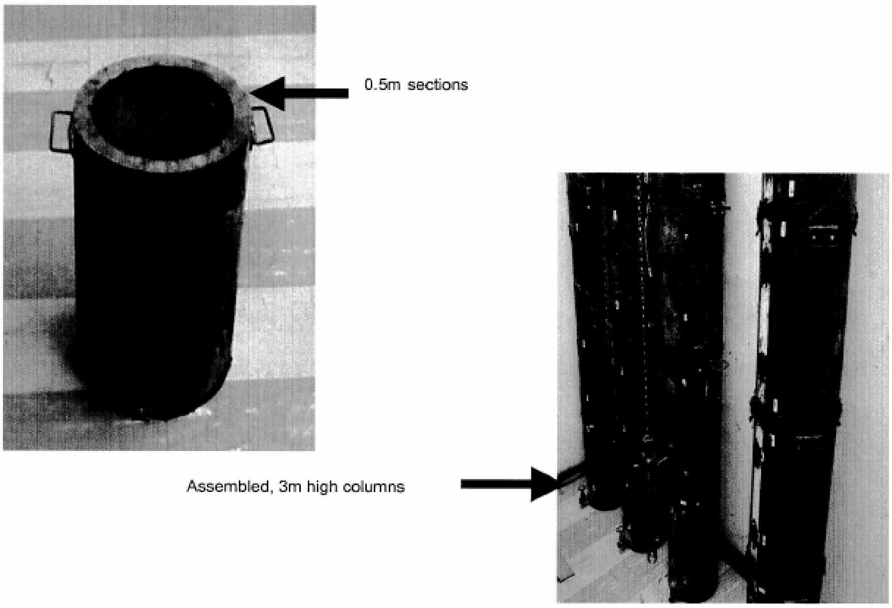


Figure 3.13 Structure of column test apparatus
(Source: Adamski, 2003)

The columns were used for three series of tests

"a) to determine the best sealing material. In these tests the reactive material was kept constant (10% ash coking coal product), the thickness of the sealing material was kept constant at 2 metres and all the potential sealing materials available at the mine were tested in sequence

b) to determine the effect of sealing material thickness. In these tests the reactive material was kept constant, as above, and the sealing material was kept constant (the most promising of the materials used in point a above) and the thickness of the sealing material was varied (0.5m, 1.0m, 1.5m and 2.0m)

c) to determine the influence of the same sealing layer on materials of different reactivity. In these tests a 2m thick layer of the best sealing material (weathered overburden) was used above a variety of reactive materials (10% ash coking coal, plant discards, Bench 7A inter-burden and Bench 8 inter-burden)

Gas samples were drawn from the sampling ports and tested after 1, 3 and 7 days. Thereafter, sampling was conducted at intervals of 7 or 14 days until completion of a test, usually a period of a few months. The longest running test was 8 months in duration" (Adamski, 2003).

The overall objective of this research was to design a waste storage facility within the open-pit and to ensure that spontaneous combustion would not create a problem. This could be achieved either by using the theory of sealing or by developing a risk model in which the contributory factors could be combined mathematically. While the sealing model was regarded as a safe approach (after all, additional sealing material could be added at any time should temperatures rise in a large scale test), the need was to produce a credible model taking all the test results into account. To do this a Database Mathematical Risk Model was constructed using all known contributory factors. Weightings were attributed to each factor based on experience at the mine, the results of the literature search and the empirical data obtained from laboratory tests. The final conclusion was that it was safe to proceed to a large scale test involving backfilling a small portion of the pit provided temperatures were monitored and additional sealing material was kept in reserve.

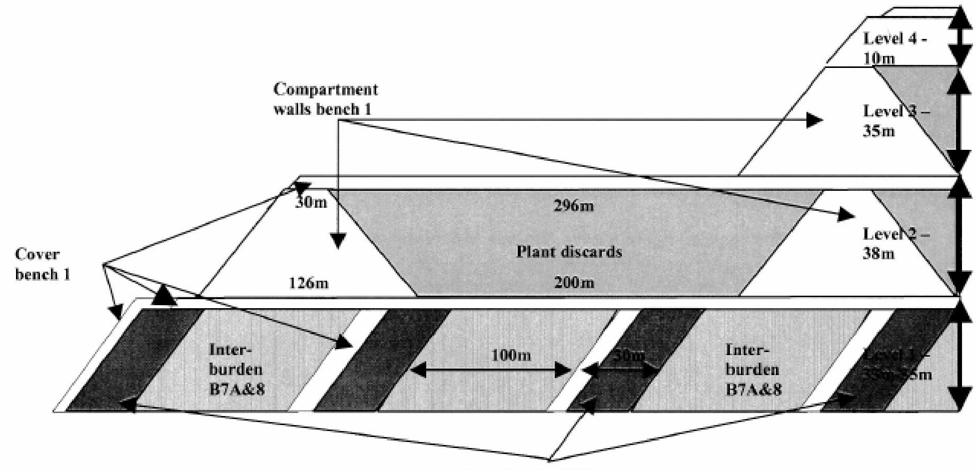


Figure 3.14 Back-filling method (east-west section)
(Source: Adamski, 2003)

It was extremely important to build the backfill in compartments and to tip the waste material in the correct location and in the required time frame. Inert weathered overburden was used as cladding and by using a 30m wall of low reactive material oxygen content was depleted in an area where heat could be dissipated. This design is illustrated in Figure 3.14.

The most important standard procedures developed for the backfilling operation were:-

“ a) filling material into pre-built compartments to prevent the occurrence of spontaneous combustion of the reactive material within slopes, where the risk is the highest and to contain eventual combustion within one compartment in the case of combustion

b) building compartments with a fixed width to maintain a constant stacking rate, determined by the risk model as the safe rate and to maintain a constant ratio between different types of the waste material

c) sealing of the compartments to isolate reactive material between levels and compartments

d) isolate the reactive waste from the pit high-walls by a layer of inert material to prevent transfer of combustion into the high-walls, in case of combustion” (Adamski, 2003)

The backfill experiment ran for 3 years from May 2000. The tonnages disposed of in this way are given in Table 3.6

Table 3.6 Backfilling test programme
(Source: Adamski, 2003)

Bench	2000/2001 Tonnage	2001/2002 Tonnage	2002/2003 Tonnage
7A&8	5123544	4938334	5427925
10	1944520	2172774	1671039
Total	7068064	7111108	7098964

In total some 21.2 Mt of reactive material was disposed of in three years. During this time the temperature in each layer in each compartment was monitored. The most reactive material, the plant discards, showed the greatest temperature increase but remained within acceptable limits and after reaching a maximum deceased quite considerably, as shown in Figure 3.15.

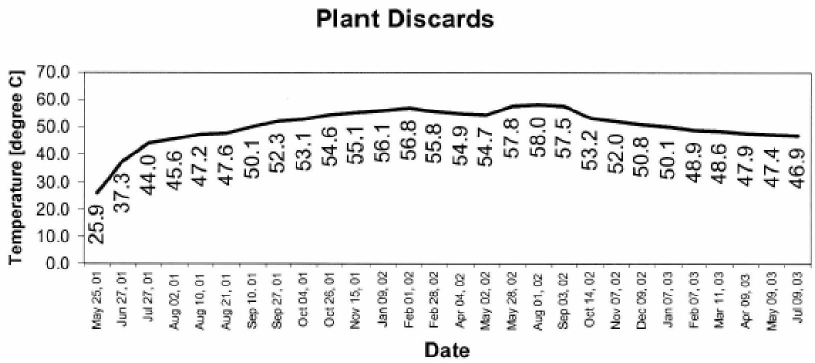


Figure 3.15 Temperature fluctuations in the plant discard compartment (Source: Adamski, 2003)

In addition to solving an environmental problem, there was a significant cost saving. Before the implementation of the backfilling programme the inter-burden material was hauled to surface, an average vertical distance of 85m and dumped outside the pit. Because this material was now hauled a much shorter distance, and downhill, the cost of inter-burden handling was substantially reduced. This resulted in an estimated cost saving of R10 million per year (in Y2000 money).

3.6.3 Prevention and control of spontaneous combustion – best practice guidelines for surface coal mines in South Africa

Several surface coal mines in South Africa experience problems of spontaneous combustion, primarily during the mining of pillars left by previous underground workings. Hazards associated with this phenomenon include hot blast holes, potential collapse of areas of the highwall and high concentrations of carbon monoxide. In addition, there can be a considerable loss of coal inventory due to in-situ burning of the old pillars and also significant atmospheric pollution. Figure 3.16 shows a highwall in a previously mined area where spontaneous combustion has resulted in a fierce fire. If left unchecked this can destroy the pillars, weaken the overburden and potentially cause a collapse of the highwall, endangering personnel and equipment. Figure 3.17 shows a dragline attempting to dig out burning material. While this may be successful it comes at a cost since high temperatures melt the grease from the ropes and repetitive temperature cycling shortens their safe working life.

In 2007 Coaltech awarded a research contract to the School of Mining Engineering at the University of the Witwatersrand (Wits) to investigate this problem. A great deal of research had already been completed locally. This had been funded by the Coal Mining Controlling Council at Wits and by Coaltech at the CSIR (Eroglu and Moolman, 2003). The 2007 project was never intended to repeat previous work but was initiated to review the current situation on those surface mines affected, to determine if any new strategies for dealing with this problem have been developed at individual mines and to produce guidelines on best practice. Recent years have seen very significant staff movement between mines and often managers and supervisors who were new to an affected mine had difficulty in finding relevant information to brief themselves on the causes of this problem and the most effective remedial procedures.

Figure 3.16 Surface mining of old bord and pillar workings

Figure 3.17 Dragline operating in spontaneous combustion area

The mines visited as part of this study were New Vaal, Landau, Kleinkopje, ATCOM and Middelburg Mine Services, while Kumba Resources provided information regarding spontaneous combustion during the disposal of waste. Information was freely shared by the mines, with examples of successes and failures in dealing with this problem being discussed openly.

The research resulted in a report (Phillips, Chabedi and Uludag, 2010) and a fold-out pocket book “Spontaneous Combustion – The Do and Don’ts” which was printed in 2011 and distributed widely at surface coal mines. It was perhaps ironic that the 62 page report hardly caused a ripple of interest while the 10 page, small format pocket book was received enthusiastically by industry.

The major findings from this study were consistent with previous work and can be summarised as :-

1. Since the spontaneous combustion problem, when surface mines rework previously mined areas, is caused by the interaction of coal and oxygen under very specific circumstances, its prevention relies on two mechanisms, the prevention of a weak airflow and the mining of exposed coal before heating occurs
2. Current problems are nearly always associated with airflow into open bords in the highwall of surface mines and the chimney effect caused by cracks to the surface behind the highwall.
3. The prevention of airflow can be achieved by cladding the surface of the highwall with sealant, blocking the ingress of air to the old workings by sealing the open bords with soft material or by collapsing the overburden into the bords by means of buffer blasting. Figure 3.18 illustrates an area where cladding has been undertaken

Figure 3.18 A highwall clad with soft, sandy material
(Source: Phillips, Chabedi and Uludag, 2010)

4. The consensus is that buffer blasting of the overburden is the most effective technique to control spontaneous combustion in previously mined areas and that the buffer must be maintained throughout the mining cycle to prevent ingress of air from the highwall. The concept is illustrated in Figure 3.19.

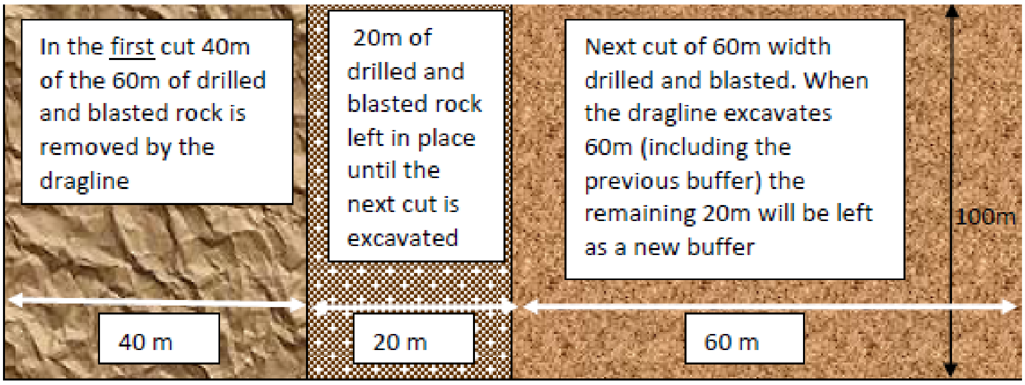


Figure 3.19 Section through the overburden showing a minimum 20m buffer (Source: Phillips, Chabedi and Uludag, 2010)

In general the mines that have used buffer blasting have been successful in controlling their spontaneous combustion problems, while those that have not used this technique continuously have experienced ongoing problems.

5. Traditionally the width of the buffer has been related to the width of the cut. However, the chimney effect, i.e. the differential pressure caused by hot air rising from the heating, is proportional to the height of the bench. One mine has designed minimum buffer widths as a function of bench height with good results.

The combination of control measures found to be most effective were:-

- buffer blasting. The dragline mines 60m wide cuts with a minimum 20m buffer maintained at all times. This requires discipline but the success of this measure is well proven
- cladding of the surface with soft material or by ploughing and compaction reduces the chances of a chimney developing
- the use of a pre-split blasting technique should be avoided as it creates cracks allowing oxygen into the old workings
- constant monitoring of the highwall should be undertaken and any change indicating a heating should be acted upon
- it is known that changes in moisture content can trigger spontaneous combustion so great care must be taken in the planning of dewatering operations. If possible they should not be undertaken ahead of time
- venting blast holes and hot holes should be closed or sealed immediately "Flower pots" and/or gas bags should be used to seal hot holes

- using hot blast holes is hazardous and a “hot hole” procedure should exist. This should detail how hot holes should be treated and how drilling should be done near hot holes
- each mine should determine the maximum time coal can safely be left exposed in the pit before it is mined and loaded out.

This was a very practical project but provided useful guidance. All coal mines should be vigilant regarding spontaneous combustion, but surface mines extracting pillars left from previous mining activity must be pro-active in their approach. Safety, health, environmental and economic considerations all dictate this.

CHAPTER 4 HEAT LOADS AND COOLING OF DEEP GOLD MINES

4.1 Heat loads and cooling practice in deep South African mines

While there are many sources of heat in all underground mines, e.g. machinery, lighting and oxidation of timber, minerals and some rocks, deep mines have two major additional factors, auto-compression of the ventilation air and heat ingress from the surrounding strata. Both of these factors are a function of depth below surface; auto-compression results in a dry bulb temperature rise of approximately 1 ºC for every 100m the air descends, while the heat flow into the mine is affected by the geothermal gradient in the area where the mine is situated. Figure 4.1 compares the virgin rock temperature (VRT) against depth for different geographical areas around the world where mining has taken place. It is fortunate that the gold mining areas in the Witwatersrand basin have a relatively low geothermal gradient (an average increase of 1.1 ºC for each 100m of depth, starting at an average stable temperature of 21.3 ºC at 30m below ground level). In contrast the underground platinum mines in the Bushveld complex have a geothermal gradient of 1.69 ºC per 100m of cover and a higher starting temperature of 24 ºC at 30m below surface.

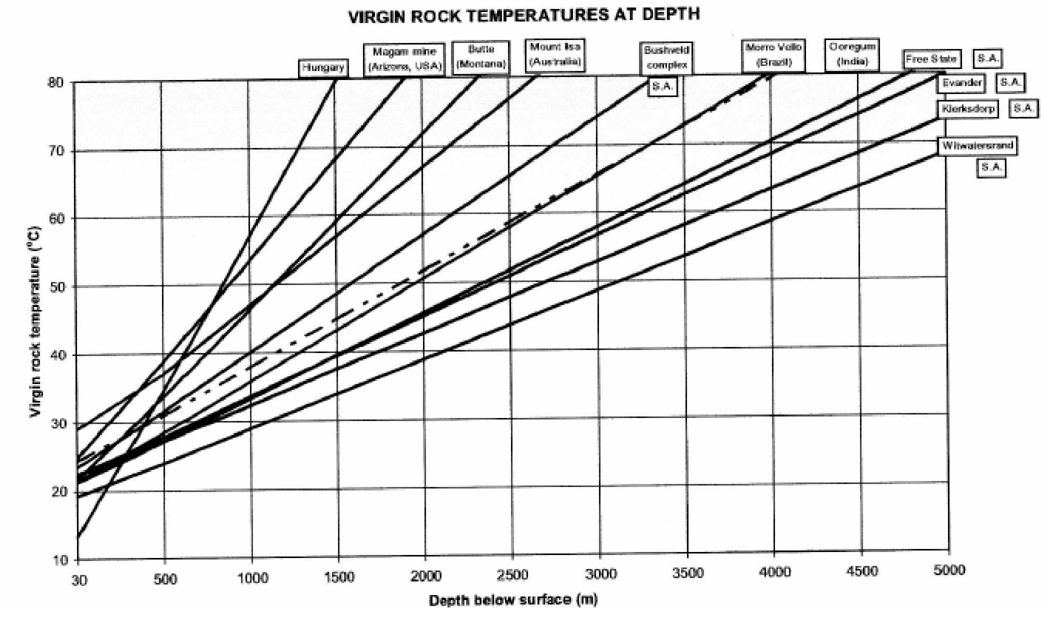


Figure 4.1 Rock temperature versus mining depth for different areas (Source: Rawlins and Phillips, 2005 compiled from MVSSA, 1987; Jeppe, 1947 & Whillier and Ramsden, 1975)

While the geothermal gradient appears modest it must be remembered that the gold mines in the Witwatersrand basin are amongst the deepest in the world, as indicated in Table 4.1, with virgin rock temperatures in the deepest workings close to 70 ºC. In most of these mines the gold bearing reefs dip sharply and it was noted from the earliest days of mining that worker performance and the number of heat stroke cases increased exponentially once a wet bulb temperature in the working places exceeded 27 ºC. By the time a wet bulb temperature of 33 ºC is reached it is an absolute necessity to install an effective refrigeration system.

Table 4.1 The top ten deepest mines in the world
(Source: Mining-technology.com, 2013)

Mine	Commodity	Country	Max. Depth
Mponeng	Gold	South Africa	3900m
TauTona	Gold	South Africa	3900m
Savuka	Gold	South Africa	3700m
Driefontein	Gold	South Africa	3400m
Kusasalethu	Gold	South Africa	3276m
Moab Khotsong	Gold	South Africa	3054m
South Deep	Gold	South Africa	2995m
Kidd Creek	Copper/Zinc	Canada	2927m
Great Noligwa	Gold	South Africa	2600m
Creighton	Nickel	Canada	2500m

Because of the historic scale of the South African gold mining industry, a truly massive research effort by the Chamber of Mines Research Organisation (COMRO), starting in the 1960s and continuing for decades, investigated refrigeration techniques, heat loads, thermal conductivity of different rocks and the physiological effects of working in hot, and humid conditions. This made South Africa a world leader in controlling environmental conditions in deep mines by air conditioning. It is not intended to review this body of work, much of which has been published widely and some of which has been summarised and incorporated into a Mine Ventilation Society of South Africa publication (MVSSA, 1987). In particular, Austin Whillier examined the whole question of heat transfer (MVSSA, 1987 Chapter 19, p. 465-494) and Roger Hemp reviewed the large body of work concerned with the sources of heat in mines (MVSSA, 1987 Chapter 22 p. 569-612). These two chapters provided a solid platform on which to base the study reported here.

The cooling strategies applied by deep level mines in South Africa, in order of their application, were

1) Bulk cooling of intake air on surface. While this is the least expensive option, both in terms of capital and maintenance costs, it comes with significant inefficiencies. These include the rise in temperature (marginally less than 1ºC per 100m) caused by auto-compression (the Joule-Thomson effect), the need to vary the refrigeration load according to ambient temperature and the heating of air passing higher temperature strata. This last source of heat can be restricted by insulating the intake airways and, in particular, if they can be kept dry then increases in wet bulb temperature can be limited.

2) Bulk air cooling underground. This can occur at convenient locations in the underground workings of the mine and has a much better positional efficiency. In addition, because the temperature deep underground is

sensibly constant all year, the cooling plant can be run at a constant capacity. If this cooling strategy is adopted water can be chilled on surface and used in large spray chambers or refrigeration plants can be installed underground. Although significantly less than for air, auto-compression (the Joule-Thomson effect) raises the temperature of chilled water by 0.234 ºC for each 100m of vertical travel and also requires the delivery pipes to be insulated to minimise heat transfer from the surrounding environment. On the other hand the efficiency of heat rejection by an underground refrigeration plant into return airways is usually low due to the hot and humid conditions associated with return air.

3) Workplace cooling. This is the usual strategy in very hot mines and involves a central underground refrigeration plant and a network of pipes through which chilled water is reticulated to the working places. In an open circuit air passes through a spray chamber and is cooled by the chilled water. In a closed circuit system air is blown through a heat exchanger and cooled, while the warmed water is returned to the plant also in insulated pipes. Workplace cooling can also be achieved by using chilled service water for dust suppression.

4) In the deepest mines the cooling requirement becomes so large that the cost of pumping the used water back to surface becomes very significant. Even if closed circuit systems are used the quantity of chilled service water can be as high as 2 tons of water per ton of rock mined. To reduce pumping costs a few mines manufacture ice on surface and pipe it underground. Because of its latent heat one ton of ice can replace up to 5 tons of chilled water, depending on the temperature at which the water is delivered.

The brief description above of the four most common cooling strategies all have one thing in common – the need for insulation, whether it is insulated and waterproof airways or insulated water pipes. Various materials had been used for this purpose but these choices were called into question by a tragic incident in 1987. On the 16th September, 1987 a welder repairing track-work in an intake airway at the Kinross gold mine damaged an acetylene bottle and set fire to cables and insulation foam installed to prevent the ingress of heat and water. The insulation used was polyurethane foam and contained a water repellent sealant, Rigiseal. This combination of materials, when burning, gave off toxic fumes resulting in the death of 177 underground workers. Almost immediately those mines using either roadway insulation or chilled water reticulation systems reviewed the materials used and, in many cases, stripped out old insulation and applied approved materials. However, this often led to less efficient insulation and the need for greater volumes of chilled water and, where possible, water chilled to lower temperatures. This coupled to increased electricity costs often resulted in cooling costs rising to between 10 and 15% of the working costs for deep mines.

In 1998 several gold mining companies, together with the Chamber of Mines of South Africa and the Miningtek division of the CSIR embarked on the 5 year DeepMine Collaborative Research Programme, which sought to create the technological and human resources to be able to mine safely and profitably down to a depth of 5000m. Cooling strategy was an obvious element of this and several research contracts were awarded to the CSIR, consulting companies and universities. The School of Mining Engineering obtained sufficient funding to appoint a Research Officer and Mr Alex Rawlins, an experienced Mining Engineer, with a particular interest in mine ventilation, joined the staff. During his 10 years in the School Dr Rawlins obtained his MSc by research (Rawlins, 1998), his PhD (Rawlins, 2002) and was appointed Research Officer, Lecturer and Senior Lecturer in succession. Between 2001 and 2005 he and I published 4 papers on the research he had undertaken and which I had supervised.

Since the Deepmine Programme was all about extending gold mining to a depth of 5000m, there was a need to consider the increased cooling requirements as mining operations extend deeper below surface and the impact this would have on working costs. Figure 4.2 indicates the need for ever larger refrigeration plants (more cooling) that is required to maintain a mean design reject temperature of say 28ºC. At the average rock breaking depth in year 2000 of approximately 2700m the contribution of a mine environmental control system (ventilation a refrigeration) was of the order of between 10 to 15% of the cost of mining. Modelling undertaken as part of this project indicated that this percentage would exceed 20% if a mine deepened to 5000m. This clearly indicated that strenuous efforts would be required to reduce the heat load.

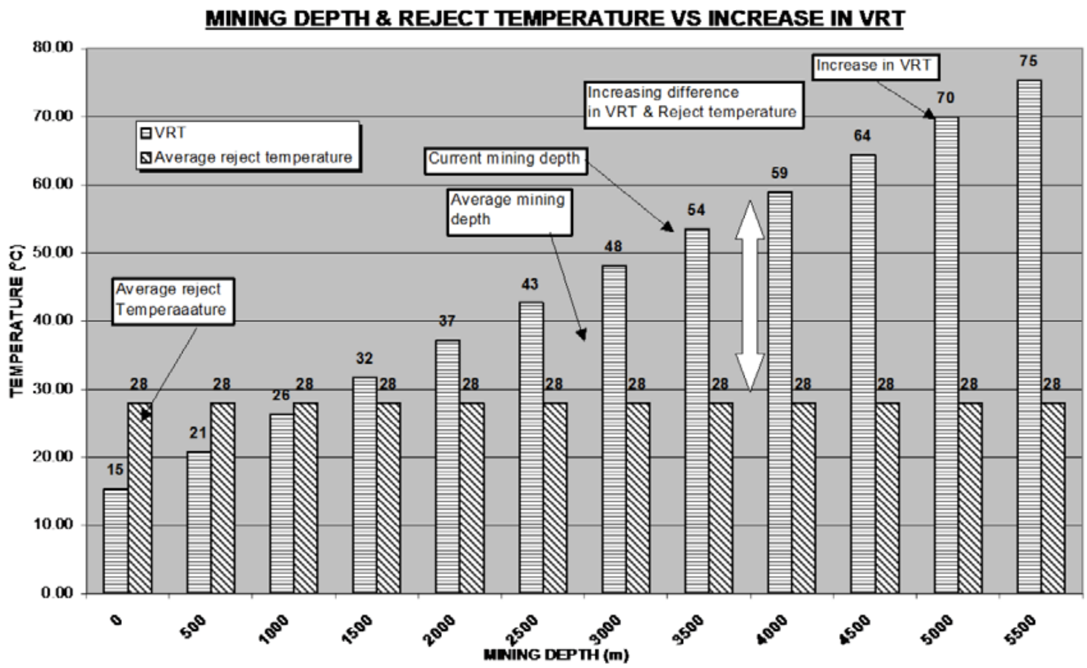


Figure 4.2 Increasing VRT and cooling requirements with depth
(Source: Rawlins and Phillips, 2001a)

A survey of the literature and analysis of data from the ventilation departments of several deep gold mines indicated that rock heat accounts for 70% of the total heat load at a deep mine, with the other 30% coming from machinery, fissure water, drilling and blasting, people, etc. Figure 4.3 also shows that of the 70% heat load attributed to rock exposure the majority (50%) comes from freshly exposed rock in the stopes and development ends, while the remaining 20% comes from haulages.

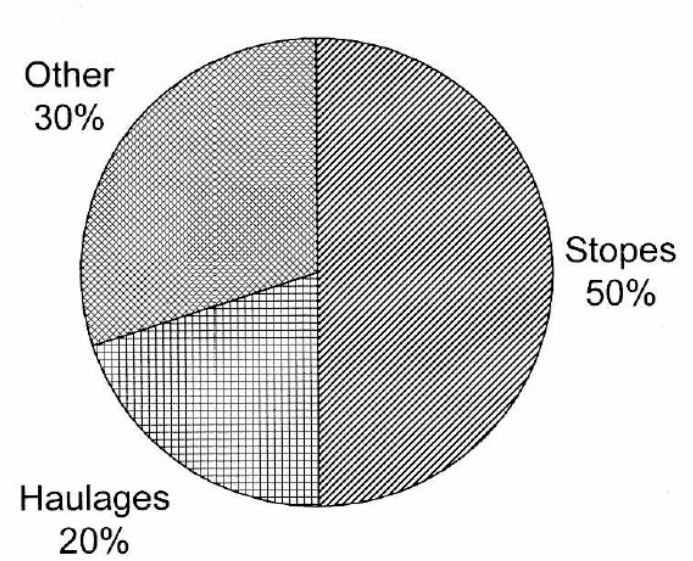


Figure 4.3 General in-mine heat load distribution
(Source: Rawlins, Phillips and Jones, 2002)

The collaborative nature of the Deepmine Programme meant that the research involving Rawlins and supervised by myself was undertaken at the CSIR Minintek, the South African Bureau of Standards and TauTona gold mine (at that time named Western Deep Levels (East Mine)), as well as at the School of Mining Engineering at the University of the Witwatersrand. This research was undertaken in four components, all aimed at controlling the costs of environmental control in ultra-deep mines. These were

I. minimising temperature rise during chilled water delivery
II. insulation of intake airways to control the ingress of heat
III. the use of backfill to control the ingress of heat in stoping areas
IV. a consideration of ventilation and cooling requirements in mechanised deep mines

Each of these components is discussed in the following sections.

4.2 The insulation of chilled water pipes

The quantities of chilled water used in cooling the underground environment at gold mines are high, with 1 200 kg/s being typical for a large shaft complex. The temperature range of this water leaving the refrigeration plant is typically between 3 and 5ºC and under ideal conditions water should arrive at its final destination at a temperature less than 10ºC. This is difficult to achieve in a deep and extensive mine due potentially to a number of factors (Rawlins and Phillips, 2001a). These can be listed as

- fluid compression with depth (Joule-Thomson effect, JT)
- friction losses
- absence of energy recovery systems
- heat flow from the surrounding rock

In addition, the design and maintenance of the chilled water reticulation system can also have an impact on chilled water temperature increase due to

- selection of fluid flow rate in the conveying pipes
- type and quantity of the chilled water pipe (CWP) insulation system
- damage to the vapour brake/barrier of insulation material
- size and insulation system of storage dams

It is well known by ventilation specialists that for deep mines the JT effect results in the largest increase in chilled water temperature but the second most significant factor is the poor quality of CWP insulation. Because of the fire-related problems associated with polyurethane and polystyrene based materials, as described previously, the gold mines embarked on a systematic removal of these materials from chilled water pipes. In some cases this proved impossible and various types of fire protection sleeves were installed in these areas. In addition, the replacement of the stripped insulation with alternatives such as glass fibre wool and half-round Phenolic foam insulation proved difficult for many practical and cost reasons. Consequently, by 1999, large portions of older CWPs in many mines remained uninsulated or poorly insulated. New sections of the mines developed since about 1990 have generally used pipes that are pre-insulated before being transported underground. These pre-insulated pipes have been described (Rawlins and Phillips, 2001a) and came mainly in two forms

- unplasticised polyvinyl chloride (UPVC) sleeves, with a maximum thickness of 3mm, around a metal pipe. The gap between the two elements is injected with phenolic foam. In this case the vapour barrier, which prevents the Phenolic foam becoming water-logged is the UPVC sleeve with the ends usually glued in place. The UPVC also acts as a mechanical protection to the foam material. This construction is illustrated in Figure 4.4
- pre-cut Phenolic foam half rounds that are encapsulated in a laminated aluminium polyethylene coated foil that acts as the vapour barrier and is kept in place around the foam by vacuum. The insulation and vapour barrier is then protected by a galvanised or stainless steel spiral wound sleeve. This form of insulation is shown in Figure 4.5

Research was conducted at three locations, with laboratory tests taking place in a specially constructed laboratory at CSIR Miningtek and at the South African Bureau of Standards (SABS), while underground measurements were taken at TauTona Gold Mine.

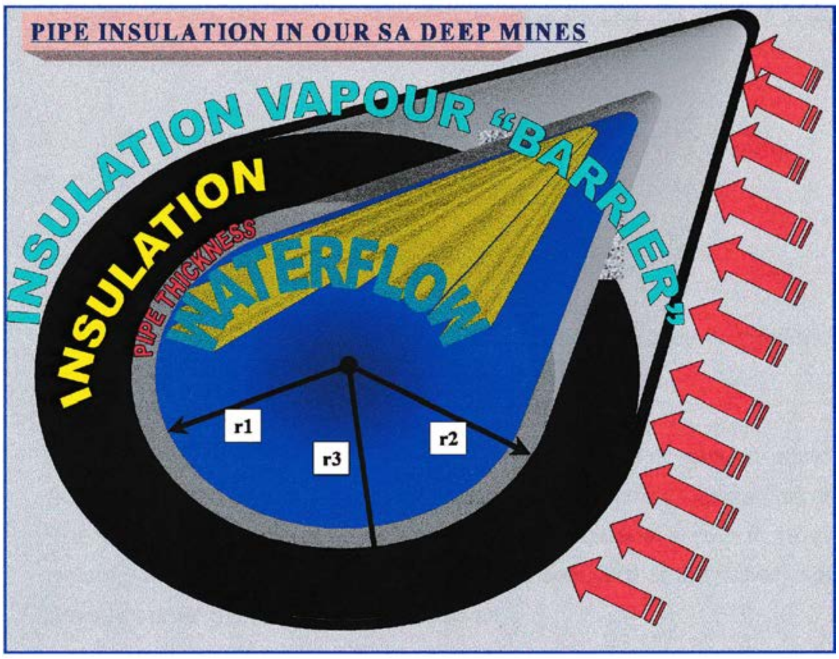


Figure 4.4 Chilled water pipe with insulation system
(Source: Rawlins, 1998)

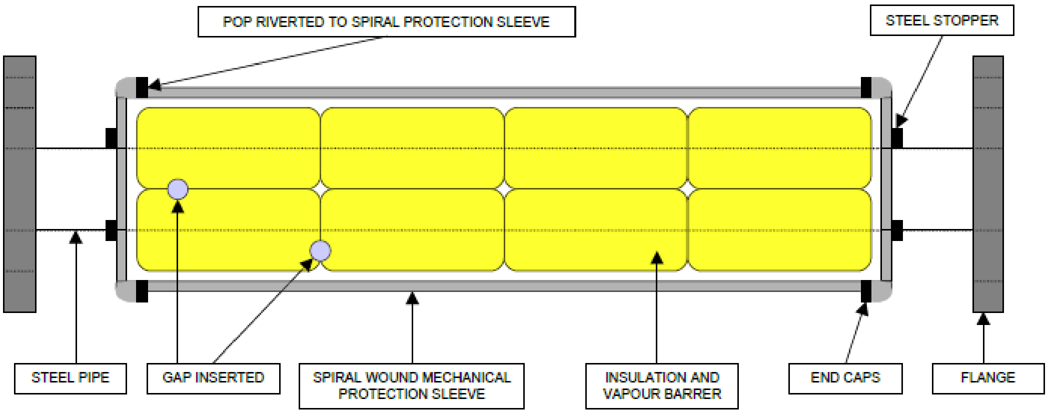


Figure 4.5 Typical half-round foam insulation of CWP
(Source: Rawlins and Phillips, 2001a)`

At the Miningtek facility (see Figure 4.6) the thermal conductivity of various insulation systems was measured since low thermal conductivity is essential if heat gain in the chilled water is to be avoided. Water vapour transmission tests were undertaken at the SABS laboratory, which at that time was the only laboratory in the Southern Hemisphere equipped for this purpose.

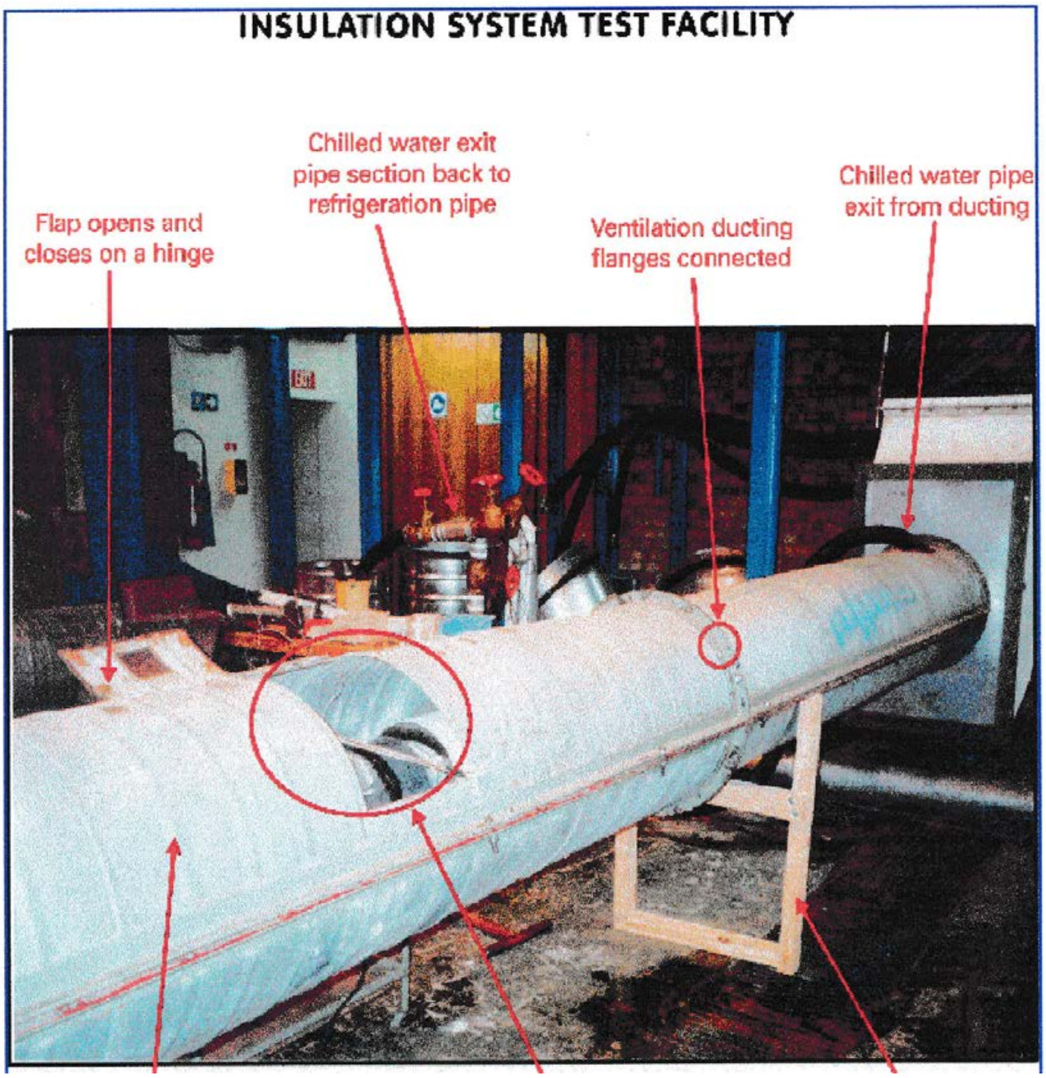


Figure 4.6 The CSIR insulation test rig
(Source: Rawlins, 1998)

Test measurements were performed underground at TauTona gold mine to determine the thermal insulation properties of CWP servicing development ends and stoping sections. While some tests were performed on relatively new installations, there was the opportunity to determine the deterioration in insulation properties for CWPs that had been in service for more than three years. Corrosion is a major problem for any exposed steel in the hot and humid conditions found at depth, as is the damage inflicted by carelessness (see Figure 4.7).

The results of this test work formed the basis of Rawlins' M.Sc. dissertation and a summary was published after some further work was undertaken (Rawlins and Phillips, 2001a)

Figure 4.7 Damaged chilled water pipe insulation system
(Source Rawlins, 1998)

The main findings of this research were

1) that over an extensive period of time the average temperature of chilled water arriving in the working sections had increased from an operational norm of 10ºC to 13ºC.
2) the impact of replacing polyurethane insulation on CWP with a less effective form of insulation had been minimal as there had been no significant change in face wet bulb temperatures. From this it could be concluded that the insulation properties of the polyurethane material had already significantly deteriorated.
3) it was found that the single most significant reason for loss of insulation efficiency was the ingress of water vapour and water into the insulation material.

Phenolic foams have very low thermal conductivity values (K-values) of approximately 0.03 W/mK. A range of different products were tested at the CSIR Miningtek facility in order to determine this value (Rawlins et al, 2001). However, these foams are known to be strongly influenced by the ingress of water vapour due to the vapour pressure difference that is generated between the actual in-mine atmospheric conditions and the atmospheric condition at the surface of the

CWP. The ingress of water vapour increases the K-value of the foam insulation and results eventually in vapour condensing on the outside of the insulation when dew point temperatures are reached. This condensate further penetrates the insulation through physical mechanisms such as hygrospicity, permeability and capillary action. When the K-value of the insulation reaches that of water (K of approximately 0.6 W/mK), water vapour condenses on the outside of the steel pipe and the entire insulation system becomes water logged. Normally the ingress of water vapour is controlled by a suitable vapour barrier, or more correctly, a water vapour brake, wrapped around the insulation material. The aim is to keep the water vapour flow rate to less than 0.2 g/m^2/24 hours, which would result in a maximum moisture/water absorption of 40% by volume after 20 years. This slow absorption of water also slowly alters the K-value of the insulation until it eventually reaches 0.25 W/mK after 20 years, which after all is the average lifespan of most projects.

The effects of vapour ingress were quantified in tests undertaken at the South African Bureau of Standards and reported in Rawlins and Phillips, 2001a. Figure 4.8 shows how thermal conductivity increases tenfold as the insulation becomes increasing wet.

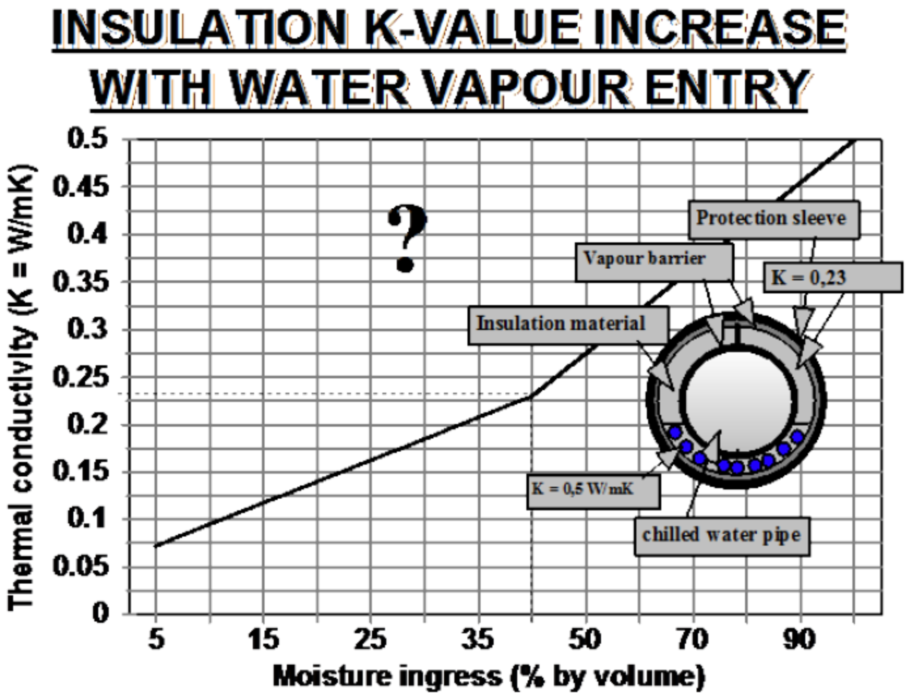


Figure 4.8 K-value deterioration with water vapour ingress
(Source: Rawlins and Phillips, 2001a)

This research (Rawlins and Phillips, 2001a) resulted in several significant findings which can be listed as follows:-

1) there was a general lack of understanding in the mining industry with regard to the technical importance of a WVB and this had allowed inferior products to be used in deep level mines

2) it is a fact that water vapour can easily penetrate most materials used to insulate CWPs due to the very high driving force generated by the partial vapour pressure differences across the insulation system. This partial pressure was calculated as part of this research and found to be approximately 3 500 Pa.

3) it is vital that the longitudinal, radial and end piece joints of water vapour barriers are sealed or bonded together. Despite this being particularly true when pipes are joined by insulated flanges, in practice this is rarely done

4) a financial model was created using the data obtained from Tau Tona (Western Deep Levels (East Mine)). The model clearly indicated the financial benefit of opting for insulation with a good K-value of optimal thickness. The numbers clearly indicated that in 1999 the annual cost per metre saving for good insulation (0.05 W/mK) was 2.5 times the cost of the original insulation. This meant that, at that time, the insulation was paid for in less than five months and even poor insulation (0.25 W/mK plus uninsulated flanges) had a payback period of eight months. Between 2000 and 2008 electricity costs rose at about the same rate as inflation but following the blackout crisis of 2008 electricity prices have risen at approximately double the inflation rate. Under these circumstances the benefits of quality insulation will be even more pronounced.

5) The financial model also indicated that even poor insulation is highly beneficial when compared to no insulation. It also highlighted the massive savings that can be achieved by delivering chilled water through good insulation systems rather than through poor ones. Over a 20 year period a saving of R16.6 million (in year 2000 money) could be achieved for one mining level alone. At a mine with four main intake levels the savings would be a very considerable R67 million, again in year 2000 money.

4.3 The reduction of heat ingress in intake airways

As part of the literature search at the start of this project, it was determined that about 20% of the total heat load entering a deep mine occurs in the intake airways (haulages) and results in a significant increase in the temperature of intake air. By 1999 many mines had already used sprayed-on material in their haulages but there was little information available regarding the effectiveness of this technique as a means of insulation, with some mines even reporting that the main purpose of doing this was stabilize the rock surface to prevent fall of ground accidents.

Since there was sufficient knowledge available regarding the time-dependent heat flow through rock and the research undertaken on insulation material (Rawlins and Phillips, 2001a) had yielded additional data, the effects of haulage insulation could be modelled. The two material properties which were technically evaluated were thermal conductivity (W/mK) and material strength (Pa) (compressive, tensile and energy absorption before failure).

These two properties, when combined, have a direct influence on each other in that they oppose one another. When the thermal conductivity value is good (say below 0.15 W/mK), then the strength properties are usually reduced (say to a tensile strength of 2 MPa) and visa versa. Therefore the choice of insulation material to be sprayed on the rock surface of a haulage is inevitably a compromise, where it has an acceptable thermal conductive value and an acceptable strength. There were two options considered in providing haulage insulation – a "one pass" system in which a material of adequate thermal conductivity and strength properties is applied and a "two pass" system where two independent materials are applied separately. One material would be the thermal insulation component and the second material would supply the necessary strength characteristic. Obviously the "one pass" system is preferable since a single application would provide both support and insulation in a logistical and cost effective manner.

In order to calculate the heat loads a model was specifically developed for this purpose and this has been fully described (Rawlins, 2002). The model was used on several occasions but, in Rawlins and Phillips, 2001b, the input parameters used were those given in Table 4.2, while outputs calculated by the model are given in Table 4.3. These outputs indicated that insulating a haulage at 5000m below surface would reduce the haulage refrigeration requirements to that of a mine at 3000m depth.

Table 4.2 Input data used in example of heat load calculation
(Source: Rawlins and Phillips, 2001b)

Haulage Properties		
Haulage length	1000	m
Haulage dimensons	4 x 4	m
Air temperature (Wet/Dry bulb)	20/20	°C
Air quantity	80	m^3/s
Age of airway	12	months
Rock Properties		
Rock thermal conductivity	6.0	W/mK
Rock density	2960	Kg/m^3
Rock thermal capacity	850	J/kgK
Geothermal gradient	0.99	°C/100m
Surface height above sea level	1600	m
Haulage depth below surface	5000	m
Mean surface temperature	20	°C
Insulation Properties		
Insulation thickness	50	mm
Insulation conductivity	0.15	W/mK

The sequence in which the research (Rawlins and Phillips, 2001b) was undertaken can be listed as follows

- haulage heat load calculation (uninsulated)
- specification of insulation/support material properties
- calculation of the new heat load
- determination of the reduction in heat load
- evaluation of the financial benefits

Table 4.3 Heat loads calculated by model
(Source: Rawlins and Phillips, 2001b)

Depth below surface	Uninsulated haulage (W/m)	Insulated Haulage (W/m)
1000 m	146.9	70.2
2000 m	357.1	179.9
3000 m	573.7	296.0
4000 m	795.8	419.1
5000 m	1022.7	549.9
6000 m	1253.7	689.3

When the analysis was extended to consider the extent of the workings of a typical gold mine operating at 5000m below surface, the reduction in haulage heat load was found to be some 47%. Since the Deepmine Programme was all about being able to mine safely and profitably down to a depth of 5000m below surface, the results of the haulage heat flow model were used as one of the inputs to a financial model (Rawlins and Phillips, 2001b) and Rawlins, 2002.

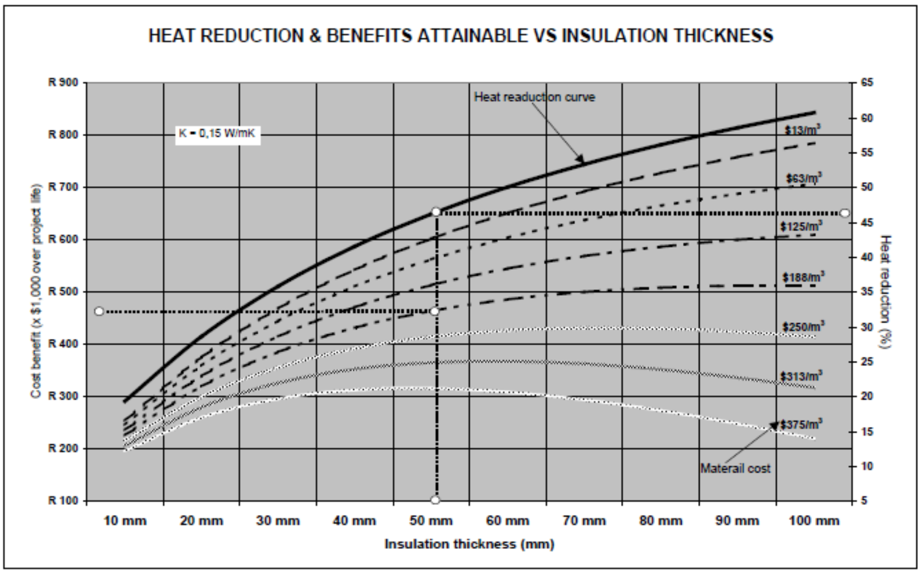


Figure 4.9 Cost benefits of insulating intake airways
(Source: Rawlins and Phillips, 2001b)

The capital and running costs used were in year 2000 money and in the intervening period until 2017 these have changed enormously. It would be extremely unwise to try to extrapolate this data and if the model were to be reused new costs would need to be obtained and then used as inputs. However, Figure 4.9 is presented here to show how, in 2001, the effects of spending more on insulation material could be related to the percentage reduction in heat load and the cost benefit of this over a 20 year lifetime.

4.4 The use of backfill to control heat ingress in deep level mining

As shown in Figure 4.3, 50% of all heat entering the ambient air in the underground workings of a deep mine comes from the fresh, hot rock surfaces (face, hanging wall and footwall) in stopes that are continually moving forward in the production process. Mine workings are, therefore, exposed to large surface areas of rock at the virgin rock temperature (VRT). This exposure is substantially reduced in many deep mines by the introduction of backfill in the worked out back areas of a stope. Although the primary reason for the use of backfill is that of rock support and the prevention of violent rock movement (rock bursts), backfill has an important secondary function in reducing heat load. However, the transportation and placement of large quantities of slurry type fluids in worked out areas constitutes an additional heat source (Rawlins and Phillips, 2001b). The third aspect of the research undertaken by Rawlins and myself as part of the Deepmine project involved quantifying this heat load and describing the properties of backfill as a precursor to developing a backfill cooling strategy. In order to do this it is necessary to know the thermal properties of the slurry both during transport and after placement. These depend in turn on the composition of the backfill (namely the amount of solid, water and air) and the thermal conductivity, density and heat capacity of the components. These parameters were measured in a separate programme running in parallel to the Deepmine project (Jones and Rawlins, 2002).

The results published by Jones and Rawlins showed that a typical slurry has a density during transport of 1700kg/m^3, with a conductivity of ~1.35 W/mK and a heat capacity of ~1980 J/kgK. These properties are such that the rate of increase in temperature of the backfill during its descent into the mine is more than twice that of water. Once the slurry is in place, its conductivity and density increase to ~2.65 W/mK and ~1980 kg/m^3 at the saturation point. As the backfill drains and becomes more porous the density falls to 1600 kg/m^3 and the conductivity decreases to ~0.5W/mK. The thermal conductivity of dry backfill is therefore one third to one tenth of that of the rock being mined. The heat capacity of backfill decreases continuously during dewatering and for a dry sample is about 820 J/kgK.

The main objective of this research (Rawlins and Phillips, 2002) was to determine whether or not the insulating effect of backfill was outweighed by the heat load introduced by transporting and placing of the backfill. In order to determine this a notional mine, 5000m deep and with the layout shown in Figure 4.10 below, was considered.

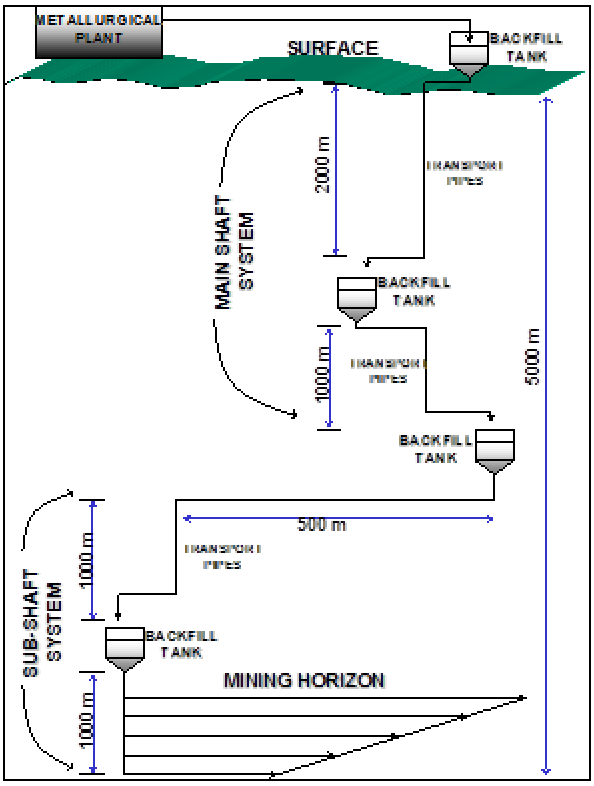


Figure 4.10 Backfill transport system for a notional mine
(Source: Rawlins and Phillips, 2001b)

A description of this notional mine, based on typical Witwatersrand gold mining practice of the late 20th century (except for the mean rock breaking depth), is summarised in Table 4.4, while the description of a single stope is summarised in Table 4.5.

Table 4.4 Design parameters of notional mine
(Source: Rawlins and Phillips, 2002)

Description	Value and units
Production rate	220000 tpm
Development (18.2% of production)	40000 tpm
Stope dimensions	1.0m x 30m
Rock density	2.78 t/m^3
Stope volume (Total mine)	64260m^3
Stope advance per month	15.36m
Number of stopes	140
Backfill density (feed – placed)	1.7 – 2.4 t/m^3
Backfill feed requirement	111000 tpm
Stope volume available for backfill	80.5%
Mean rock breaking depth	4700 metres below collar

The data given in Tables 4.4 and 4.5 were used to calculate the heat loads. These calculations were undertaken using the formulae developed by Von Glehn in

1987 as part of an M.Sc. study jointly supervised by Professor Tony Starfield and myself (Von Glehn, 1987).

Table 4.5 Stope description
(Source: Rawlins and Phillips, 2002)

Description	Value and units
Virgin rock temperature	67ºC
Dry-bulb temperature	28.3ºC
Wet-bulb temperature	26.3ºC
Heat transfer coefficient (unsealed)	3 W/m^{2}K
Heat transfer coefficient (other)	6 W/m^{2}K
Dip gully to face distance	50m
Face distance to ventilation control	3m
Strike gully width	2m
Number of strike gullies	6
Face length of stope line	181m
Airflow through stope	8 m^3/s

The heat load was calculated for three conditions:- a non-backfilled stope, a stope backfilled but with the areas between the packs not filled and finally for a fully backfilled stope. The results are shown in Table 4.6 below.

Table 4.6 Calculated heat loads
(Source: Rawlins, Phillips and Jones, 2002)

Stope Zone	No backfill (kW)	Backfill but not areas between packs (kW)	Full Backfill (kW)
Face and broken rock	67	67	67
Foot & hangingwall	295	295	295
Unsealed worked out area	362	110	0
Dip gully	7	7	7
Strike gully	37	37	37
Total stope heat load	768	516	407

It can be seen that the heat load reduction potential is substantial (768-407 = 361 kW/stope) although it diminishes from 47% for a fully filled stope to 33% if the stopes are not completely filled.

Having quantified the potential heat reduction benefits of using backfill, it was necessary to also consider the heat load introduced by transporting and placing the backfill. As described in Rawlins & Phillips, 2001b, the heat load for a single pipe system transporting backfill from surface to a single stoping section of an ultra-deep mine (5000m below collar) is in the order of 500kW, depending on the transport methodology followed. For the notional mine described in Table 4.4, the total backfill heat load could typically amount to 24MW during any particular shift with 50% occurring within the stoping sections of the mine and the remainder in the shafts and intake haulages. The heat load induced by the introduction of backfill results from the slurry properties and distances travelled.

Backfill properties, such as the following, are some of the important parameters.

I. particle material, i.e. quartzite
II. slurry make-up (water to solid mass ratio)
III. slurry density
IV. thermal capacity and
V. thermal conductivity

Information regarding slurry make-up and slurry density was obtained from the mines, thermal capacity was calculated and the thermal conductivity was measured in the laboratory.

a) Thermal capacity

The larger the thermal capacity, the more energy that is required to change the fluid temperature. For example, it is well known that water requires some 4187 Joules to raise the temperature of 1kg through 1ºC. In the case of backfill the thermal capacity range of the slurry is always lower than that of water, i.e. its capacity to absorb heat is that much greater. The reason for this property of the slurry is the combination of water and solids, with the solids having the lower thermal capacity. The solid particles in the slurry are waste from the metallurgical plant and in the case of gold mining are primarily quartzite particles. These have a thermal capacity in the range 750 to 1000 J/kgºC.

The ratio of quartzite particles to water determines the slurry's thermal capacity, which varies between 1500 J/kgºC and 3000 J/kgºC depending on the mix ratio. Typical slurry densities used in South African mines range between 1600 and 1750 kg/m^3. The thermal capacity calculated for a 1700 kg/m^3 density is 2041 J/kgºC and this provides the fluid with the capability of increasing its temperature by some 4.81ºC per vertical kilometre transported. Therefore, for a backfill temperature of 20ºC on surface, the arrival temperature resulting directly from a vertical descent of 5000m would be 44ºC. However, in practice it would be a few degrees cooler when it reaches the stopes due to heat exchange with the ambient air during the transport phase.

b) Thermal conductivity

Test work conducted at the Bernard Price Institute of Geophysics at the University of the Witwatersrand concluded that the thermal conductivity of backfill ranged between 1.3 to 1.37 W/mºC. The small change in thermal conductivity is related to temperature change, since the thermal conductivity of water increases slightly with temperature, although that of the solid particles remains constant.

c) Heat transfer coefficient

The heat transfer coefficient for slurry at a density of 1700 kg/m^3 was measured at between 500 to 3200 W/m^2 ºC within the flow range 5 to 50 m^3/hour. For water in the same flow range the values are 1100 to 7300 W/m^2 ºC. This change in heat transfer coefficient value is influenced by the combination of the Nunner value, Prandtl number, fluid velocity, thermal capacity and its Reynolds number. The boundary fluid velocity is also important in this determination. The measured heat transfer coefficient values for typical samples of backfill were measured to be between 2000 and 3200 W/m^2 ºC (Rawlins and Phillips, 2001b).

Having determined these values it was possible to calculate the heat load added to the mine by transporting the backfill. It must be remembered that the heat load ingress is of a cyclic nature, meaning that this occurs during the transport phase only and for a short period after placement in the stopes. Most mines use bare pipes and do not attempt any energy recovery technology for backfill. This results in about 510 kW of heat introduced into the mine atmosphere for each stope filled with backfill. As indicated in Table 4.4 a large ultra-deep mine could have 140 stopes being filled on a 3 day cycle. This means 33% or 46 stopes would be filled each day and the heat ingress due to backfilling would be 46*510 kW or 24 MW.

The 510 kW heat ingress per stope caused by the backfill, as opposed to the heat reduction of 361 kW for a fully backfilled stope, showed that backfilling through bare pipes is not advantageous. However, it is possible to transport the backfill through insulated pipes and refrigerate it at an appropriate point before placement. A practical and effective way of providing cooling is to use a heat exchange facility. The heat exchange unit investigated as part of this study (Rawlins, Phillips and Jones, 2002) was a “pipe within a pipe” system positioned close to the stope being filled.

An experimental laboratory unit, built to simulate the heat exchange process, is illustrated schematically in Figure 4.11. In this heat exchanger chilled water flows in a counter-flow direction relative to the backfill. The heat exchanger would, in practice, be fed with chilled water from the main chilled water pipe system used to supply refrigeration to deep underground mine workings. The laboratory experiments showed such a system can significantly reduce the heat ingress associated with backfilling. As described in Section 4.2 above, it is vital that the insulation material around the outer chilled water pipe is enclosed in a vapour brake material to avoid vapour absorption by the insulation material.

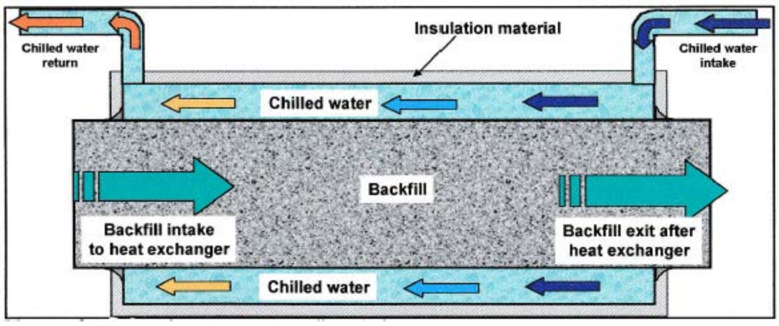


Figure 4.11 Schematic view of experimental backfill heat exchanger
(Source: Rawlins, Phillips and Jones, 2002)

Because backfill operations are of a cyclic nature, the “pipe within a pipe” heat exchanger can be set to operate only during backfilling, typically over a 4 hour period. In addition, the optimum use of the heat exchanger system should include insulating the backfill pipe from surface to just before the heat exchanger in order to minimise heat exchange to intake air in the shafts and haulages. In this way nearly all the heat is exchanged at the designated heat exchange facility close to the workings.

Extrapolating the laboratory results to the underground situation was relatively simple. For a backfill flow rate of 27 m^3/hour, a heat exchange unit of approximately 400 kW would be required to reduce the backfill temperature of about 40ºC to a design wet bulb temperature of say 28ºC. The tests indicated that this could be achieved by approximately 67m of heat exchange piping with a chilled water flow rate of 9 ℓ/s and assuming a heat transfer coefficient of 35 W/m^{2}ºC.

The paper (Rawlins, Phillips and Jones, 2002) contains a financial cost/benefit analysis of using backfill as a means of reducing the heat load in a deep mine, despite the fact that the backfill would be placed irrespective of this consideration. As stated previously its primary purpose is to prevent uncontrolled stope closure and to reduce the risk of rockfalls and rockbursts. The analysis did, however, indicate the positive financial benefit of insulating the haulages and using backfill. For the deep mine scenario outlined in Figure 4.10 and Table 4.4 (maximum depth of 5000m and mean rock breaking depth of 4700m), the heat load is 190 MW(H). By using haulage insulation and backfill this could be reduced to 136 MW(H). Despite the cost of introducing these two technologies, the total owning costs of refrigeration decreases, realising an 11% saving over the projected 12 year life of this notional shaft system.

4.5 The effects of mechanisation on gold mine ventilation

The Deepmine programme recognised that mining beyond a depth of 5000m would require a new approach, with the traditional labour intensive methods being replaced by mechanisation. Concepts such as ventilation on demand, autonomous transport systems and non-explosive mining systems were all examined. Even changes to the traditional stoping methods were examined and re-examined. Because of this the fourth and final aspect of our Deepmine project was an investigation into the effect on ventilation and cooling requirements when large, diesel powered equipment is introduced in deep level mining (Rawlins and Phillips, 2005).

While there are many benefits in using trackless equipment, the increase in air quantity requirements and the heat loads imposed need to be evaluated carefully and incorporated into the ventilation design. In particular, the amount of air required at the point of operation of diesel equipment must ensure that any gas/ fumes or particulate matter is adequately diluted to legally allowed Occupational Exposure Limits (OEL’s). In 2003 the generally accepted exhaust outlet pipe dilution factor was 0.06 m^3/s/kW of the diesel engine’s rated power and the loadhaul-dump (LHD) most commonly used at that time was equipped with a 186 kW engine. The required air quantit**y** to meet legal requirements could therefore be calculated as

$$\text{Fresh air quantity (Q)} = 186 \text{ kW} \times 0.06 \text{ m}^3\text{/s/kW} = 11.2 \text{ m}^3\text{/s}$$

In addition to pollutants, trackless diesel equipment generates considerable heat loads. The most accurate means of calculating this is from the fuel consumption and the properties of that fuel. This is best illustrated by an example using the 186 kW LHD and the diesel fuel it used.

At full power the LHD would use 0.178 ℓ/hour per rated kW of power,

i.e. 0.178 x 186 = 33.1 ℓ/hour of diesel with a calorific value of 35 400 kJ/litre

Thus:
Heat produced = Diesel usage x calorific value of the diesel/3600 seconds/hour

$$= \frac{33.1 \text{ x } 35400}{3600} = 325.48 \text{ kW}$$

The heat produced by diesel equipment has two components; sensible heat and latent heat. The latent heat component can be calculated from the water produced, which is normally taken to be that one litre of fuel results in 1 to 1.1 litre of water, i.e. 33.1 ℓ/hour (McPherson, 1993). Given that the latent heat of evaporation of water is about 2450 kJ/kg, then the latent heat produced will be

$$\left(\frac{33.1 \text{ x } 2450}{3600}\right) = 22.53 \text{ kW}$$

The sensible component is therefore 325.48 – 22.53 = 302.95 kW.

If it is assumed that the LHD will operate at full power rating and with a 100% utilization, then the heat load factor imposed on the surroundings will be

$$\left(\frac{325.48}{186}\right) = 1.75$$

Depending on such variables as diesel consumption rate, calorific value of the fuel, machine utilization, etc the heat load will also change. Figure 4.12 shows the heat load imposed on the surrounding air in relation to the 186 kW diesel engine operated in conjunction with its full power output at different utilization factors. Of course, when a diesel machine operates underground, the heat load influences the ambient air temperature. This means the air quantity determined from a fume dilution point of view may be insufficient to keep air temperatures within acceptable limits. In deep, hot mines trackless machine heat load could affect the ventilation cooling strategy quite significantly and could result in either the need to increase air quantity or that additional, localised cooling is required in order to ensure air temperatures do not exceed the mine reject temperature.

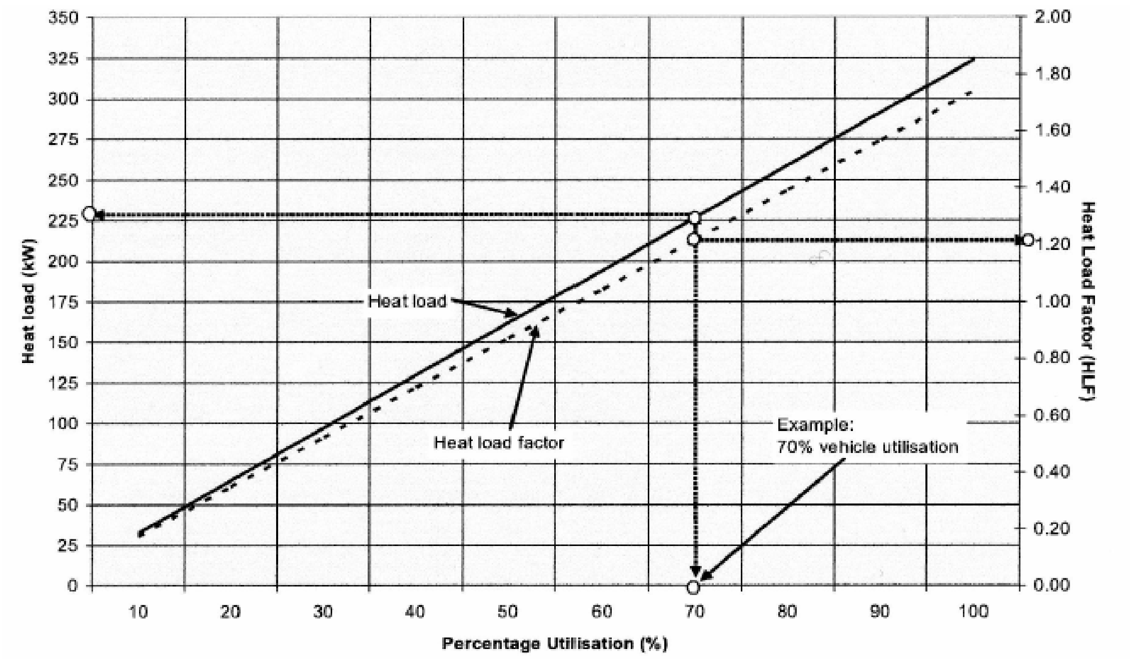


Figure 4.12 Heat load and heat load factor for a 186 kW diesel engine (Source: Rawlins and Phillips, 2005)

A simulation programme was developed to determine the maximum allowable wet-bulb temperature of the air arriving at a working machine in order to ensure the temperature at the machine did not exceed prescribed limits. The wet bulb reject temperature in most deep level mines in South Africa ranges between 27ºC and 28.5ºC and so, in the following example, a reject temperature of 27.5ºC was arbitrarily chosen. A simplistic psychrometric calculation was then conducted using the parameters in Table 4.7

Table 4.7 Input values for sample psychrometric calculation

Parameter	Value
Barometric pressure	96 kPa
Temperature after LHD	27.5ºC wet bulb 33.0ºC dry bulb
Air flow past LHD	20 kg/s 18.1 m^3/s

The calculations showed that when the air has passed over the LHD its sigma heat value is 87.84 kJ/kg and its energy content is 1754.03 kJ/s. Therefore the energy content of the air just before it reaches the LHD should not exceed (1754.03- 323.7) = 1433.23 kJ/s. This energy content corresponds to a sigma heat value of 71.66 kJ/kg. From this it can be deduced that the wet bulb temperature of air reaching the LHD should not exceed 23.7ºC, as anything in excess of this would increase the wet bulb temperature after the LHD to above the design temperature of 27.5ºC should the LHD be working at its maximum power rating.

One of the industrial partners in the Deepmine programme, Gold Fields Ltd, had an unusual resource at its South Deep mine, where a package of reefs could potentially be mined using massive mining techniques, A fully mechanised method, using multiple LHDs and dump trucks, at a depth of about 3000m would pose an unique ventilation and cooling problem. Using the mathematical models and software developed, a number of scenarios were considered and the maximum intake wet bulb temperatures for different suites of equipment calculated. The results were displayed in simple graphs, one of which is shown in Figure 4.13.

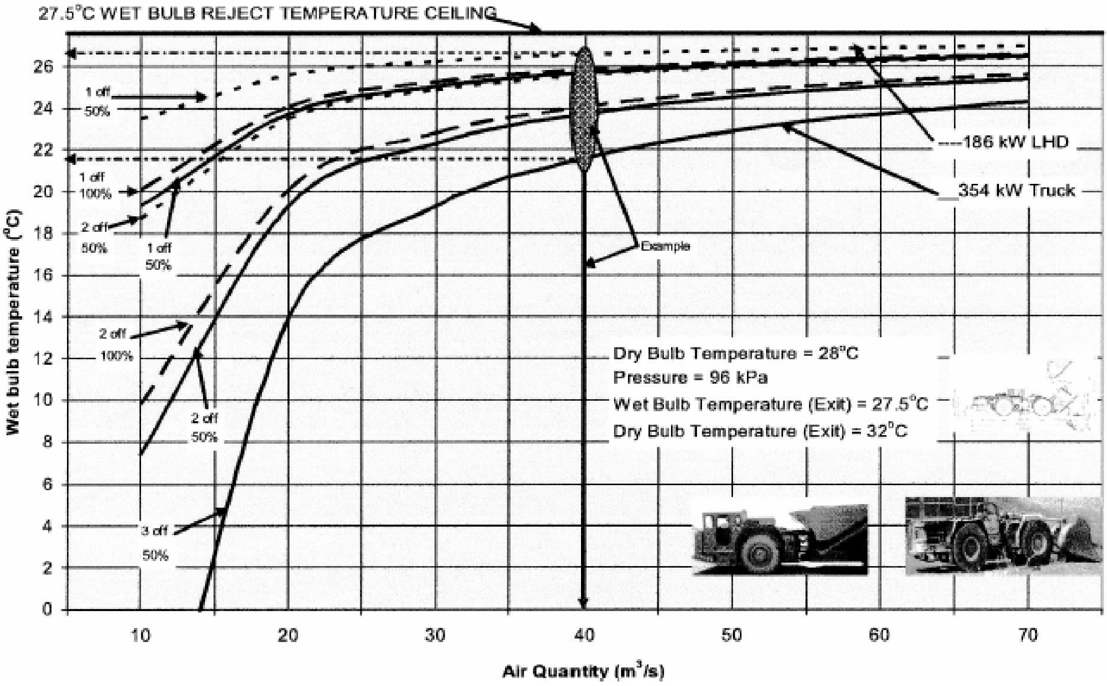


Figure 4.13 Maximum wet bulb temperatures before diesel vehicles operating in series (Source: Rawlins and Phillips, 2005)

The research described in this Chapter has indicated the extreme technical difficulties and the costs involved in maintaining an acceptable working environment in ultra-deep mines. Despite the fact that 15 years have passed since these results became available, it is only recently that the development of truly autonomous mining systems has started. In the meantime many mines have abandoned their ultra-deep reserves as being too difficult and expensive to mine.

CHAPTER 5 MEASUREMENT AND CONTROL OF RESPIRABLE DUST LEVELS IN COAL MINES

The mechanical excavation of coal or rock invariably leads to a product size distribution that includes fine particles. When the cutting is achieved by fast-moving drag bits mounted on rotating drums, this fine material may become airborne and thereby constitute a source of respirable, explosive or visible coal dust. The problems associated with airborne coal dust are well known and fall into three categories: health, safety and nuisance.

a) the health hazard associated with coal dust.

   Prolonged exposure to a dust-laden environment can lead to coal workers' pneumoconiosis in the case of respirable coal dust and silicosis when the rock being mined has any appreciable quartz content. Medical research has shown that it is the weight of dust, and not the number of particles, retained by the lungs that is the critical factor. It is the health hazard associated with respirable dust that is discussed in this Chapter.

b) the safety hazard associated with coal dust.

   Airborne coal dust can contain particles of sizes ranging from virtually 0 μm in diameter to approximately 400 μm in diameter. Particles less than 200 μm are usually well dispersed by the mine ventilation and settle over a wide area. It is this deposition of fine dust over the perimeter of a coal mine roadway and on any structures within the roadway that forms a potential risk of an explosion. This inherent danger caused by creating and distributing coal dust within the mining environment is obvious and the methods of preventing coal dust explosions will be dealt with in Chapter 6.

c) the nuisance value of dust.

   Particles of dust larger than those classified as respirable can cause considerable physical discomfort. Anyone who has worked in a dusty environment can testify to the effects of dust on their eyes, nose and throat. Furthermore, a reduction in visibility impedes the ability to perform many tasks accurately, e.g. operation of trackless vehicles, shuttle cars and continuous miners.

5.1 Coal Workers' Pneumoconiosis

Growing up in a coal mining village in South Wales in the 1950s and early 1960s I thought it was a perfectly normal phenomenon that old (i.e. > 50 years of age) men walked slowly, were short of breath and spat occasionally. It was only years later that I realised this was the result of an occupational disease. Even during my time working for the National Coal Board (1968 to 1973) the effects of dust diseases were hardly mentioned and although heavy, rubber dust filtration masks were available, it was not compulsory to carry them underground and, because they were so uncomfortable, they were rarely worn.

Lung diseases contracted by mine workers are categorised as pneumoconiosis, a term developed from the Ancient Greek words *pneûma*, "wind, breath, spirit" + *kónis*, "dust". Pneumoconiosis is the general term given to any lung disease caused by dusts that are breathed in and then deposited deep in the lungs causing damage. Pneumoconiosis includes asbestosis, silicosis and coal workers' pneumoconiosis (CWP) depending on the dust or fibres inhaled. Asbestosis is generally confined to workers in asbestos mines or trades which previously used asbestos for insulation and building purposes. These days most uses of asbestos are prohibited and asbestos mining has virtually disappeared in developed countries. In the early days of underground gold mining on the Witwatersrand, silicosis was a major cause of disability and death amongst mineworkers and, although compensation claims for silicosis currently receive much publicity, working practices in gold mines such as automatic watering down of muck piles, observing minimum re-entry times after blasting, etc. have done much to reduce the exposure of gold miners to respirable quartz dust.

A good layman's description of coal workers pneumoconiosis (CWP) appears in a document issued by the Department of Natural Resources and Mines of Queensland (Government of Queensland, 2016). "Coal workers pneumoconiosis, (sometimes called black lung), is an occupational lung disease caused by prolonged exposure to respirable coal dust. If a worker is exposed to high concentrations of coal dust over several years, the dust collects in the alveoli (air sacs) of the lungs causing a reaction that results in scarring of lung tissue, reducing the elasticity of the lung. If enough scar tissue forms, lung function can be seriously reduced. Coal workers' pneumoconiosis has a long time lag between a person's first exposure and the identification of the disease.

There are two types of coal workers' pneumoconiosis:

- simple (early stage) coal workers' pneumoconiosis; and
- complicated coal workers' pneumoconiosis, also known as progressive massive fibrosis (PMF).

"Simple CWP can manifest in different ways in individuals, depending on the composition of the dust, duration of exposure and individual factors. In the early stage of the disease, small scars called nodules between 1 – 2 millimetres in size, begin to form. A chest X-ray identifies the profusion of these round or irregular nodules to determine how advanced the disease is. In the early stages, it may be difficult to differentiate small nodules from other lung conditions, or even other normal structures in the lungs such as blood vessels" (Government of Queensland, 2016). An example of a lung affected by simple CWP is given in Figure 5.1

Figure 5.1 Lung specimen demonstrating simple CWP
(Source: Khan and Mosenifar, 2017)

"Progression to complicated coal workers' pneumoconiosis or progressive massive fibrosis (PMF) may occur in some workers if they remain exposed to high concentrations of respirable coal dust over a long period of time. Smoking and other lung conditions can contribute to the disease progressing. In complicated coal workers' pneumoconiosis there are large masses of dense fibrosis (scar tissue) in the lungs and workers with complicated coal workers' pneumoconiosis will have significantly decreased lung function, and the condition can be fatal" (Government of Queensland, 2016). Figure 5.2 shows a lung where CWP has progressed and massive fibrosis is visible.

Figure 5.2 Lung specimen demonstrating progressive massive fibrosis
(Source: Khan and Mosenifar, 2017)

Research into the causes of pneumoconiosis and into defining safe occupation exposure limits for various mineral dusts has been conducted for well over a century. Two of the earliest investigations were a report on Cornish tin mines (Haldane et al, 1904) and a report on "miners' phthisis on the Witwatersrand" (Miners' Phthisis Committee, 1916). On-going research into the epidemiology of pneumoconiosis, as well as methods of reducing respirable dust exposure continue to this day. However, despite this, the disease continues to cripple and kill miners world-wide.

In Australia "Safe Work Australia (SWA) collects national Worker Compensation (WC) data and they extracted data for pneumoconiosis claims from 2000-01 to 2013-14. They found 236 accepted claims for respiratory diseases such as silicosis and pneumoconiosis (due to coal dust or other causes). This included 162 claims for silicosis, 72 claims for pneumoconiosis (excluding asbestosis, CWP and silicosis), and 2 claims for Coal Workers' Pneumoconiosis. However, from May to December 2015 six confirmed cases of coal workers' pneumoconiosis (CWP) were identified by the Queensland Department of Natural Resources and Mines (DNRM) among coal miners in Queensland. Two further cases were later notified in the first half of 2016. Prior to this, the Queensland Coal Mine Workers' Health Scheme had not identified any new cases for many years and CWP was thought to have been eradicated in Queensland" (Monash University, 2016).

A more worrying picture emerges in the United States where an official report to Congress states "Coal mine dust is one of the most serious occupational hazards in the coal mining industry, and overexposure can cause coal workers' pneumoconiosis (CWP) and a number of other lung diseases, collectively referred to as black lung disease. CWP has been the underlying or contributing cause of death for more than 75,000 coal miners since 1968, according to the Department of Health and Human Services' National Institute for Occupational Safety and Health (NIOSH), the federal agency responsible for conducting research on work-related diseases and injuries and recommending occupational safety and health standards" (United States Government Accountability Office, 2012).

In South Africa "dust related lung diseases overshadow mine accidents in the number of workers affected. Among the 429 000 persons at work in all mines in 1998 there were 371 reported fatalities (0.85 per 1000) and 6 064 reported injuries (14.1 per 1000). By comparison there were 5 603 new or upgraded certifications for pneumoconiosis with or without tuberculosis and over 5000 new cases of tuberculosis (> 20 per 1000)" (White, 2001). In South African mining the situation is made more complex because of the known interactions of pneumoconiosis, TB and HIV. Studies conducted in the 1980s and 1990s showed the extent of this problem, which can be seen from Figures 5.3 and 5.4.

Prior to 1980 there was little reliable information available regarding coal workers' pneumoconiosis due to a number of reasons including disparity in the medical surveillance of white and black coal miners, the migrant labour system and the short (6 to 9 month) contracts undertaken by black mine workers. The data for 1980 suggested that at that time "certifications of pneumoconiosis tended to be higher in coal mines than in gold mines (approximately 6 per 1000 against 2 per 1000 in gold mines). CWP certification rates declined in the years 1980 to 1989 although in 1989 the rate was still over 4 per 1000, a rate similar to silicosis in the gold mines" (White, 2001). All available evidence is that the rates have continued to decline. While South Africa has not followed the example of installing highly productive longwalls as seen in the United States and Queensland, there has been a continuous improvement in continuous miner productivity and the CWP trends seen in Australia and the US could still occur in South Africa.

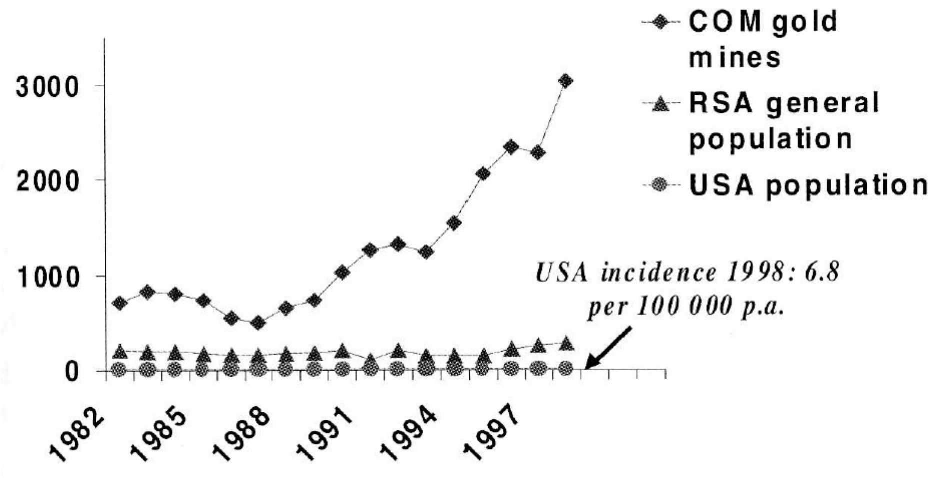


Figure 5.3 TB case rates in gold mining and RSA population
(Source: Churchyard and Corbett, 2001)

While the diagnosis of pneumoconiosis and the epidemiology of lung diseases is strictly the preserve of medical practitioners and medical researchers, the first line of defence is the monitoring of exposure to respirable dust and this is where occupational hygienists have a vital role to play. Since they work on mines and report to mine management, who also have the responsibility to take remedial action should the prescribed limit of exposure be breached, the measurement and control of respirable dust in mines becomes a legitimate concern for mining engineers.

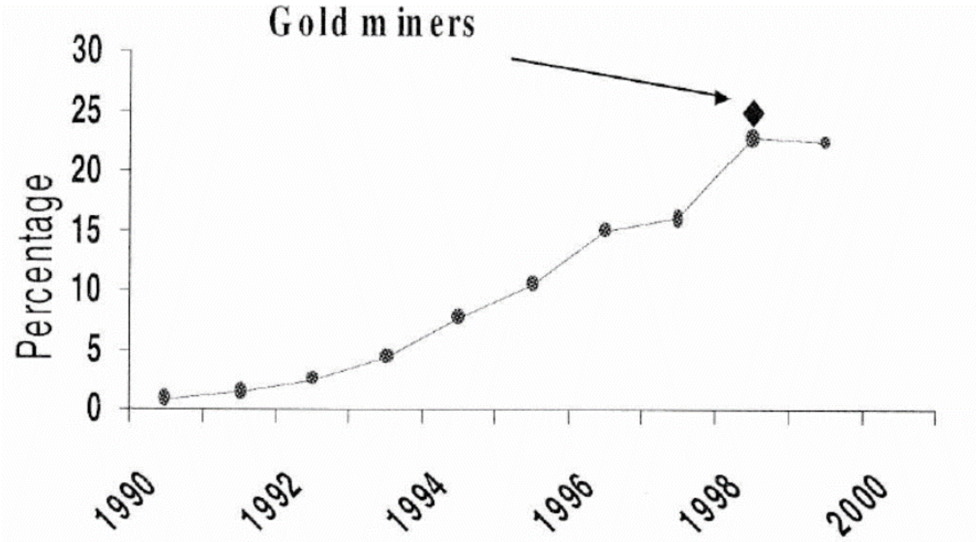


Figure 5.4 HIV prevalence among RSA antenatal attenders and miners
(Source: Churchyard and Corbett, 2001)

5.2 Setting gravimetric standards for the NSW coal industry

The Pneumoconiosis Conference held in Johannesburg in 1959 (Orenstein, 1959) transformed international thinking around the monitoring of respirable dust exposure in mines. Medical research had provided overwhelming evidence that it is the weight of dust, not the number of particles retained by the lungs, which is the critical factor in the occurrence of pneumoconiosis. This was confirmed, in part, by correlating the progression of simple pneumoconiosis with the measured mass concentration at 20 mines over a 10-year period (Jacobsen et al, 1971) and also by comparing lung analyses obtained by autopsy with the grade of fibrosis and the X-ray category.

While gravimetric monitoring was adopted by some coal mining countries in the 1960s and 1970s, in other jurisdictions there was a reluctance to do so, mainly because it was felt that existing practices were adequately controlling the problem and there was no substantial body of knowledge relating their standards to a gravimetric standard.

The first legislation quantifying the maximum permitted concentration of respirable dust in NSW coal mines was introduced in 1943. This stipulated that the measurement of dust concentration should be on a particle count basis, samples being obtained by means of the Owens Jet Dust Counter. The standard required that the average number of particles in the range 0-10 μm found in each cubic centimetre (cc) of air should not exceed 700 and that, out of a minimum of 12 samples taken in 1 hour, no more than 10% should exceed 1500 particles per cc (ppcc). Because virtually all coal mining activities in the 1940s were carried out in-seam, this standard failed to recognise any differences in the harmful effects of coal and quartz dusts. However, the advent of continuous miners, together with the introduction of roof bolting as a standard method of strata control, provided the potential for higher concentrations of respirable quartz dust. At about the same time medical evidence became available showing that dust in the size range 0-5 μm is the most readily retained by the lungs. This led in 1957 to a revision of the legislation, so the permitted dust concentration decreased linearly with increasing free-silica content of the material being mined, and the size range of the particles counted was set at 0-5 μm. The method of counting particles under a high-power, high-resolution microscope still proved troublesome, however, and was found to be the subject to operator error, particularly for particles in the range 0-1 μm. The respirable dust standards in NSW were further revised in 1967, with the number of particles permitted being reduced to take account of the smaller size range. However, the 1957 and 1967 standards both specified the Owens Jet Dust Counter as the instrument to be used and that a minimum of 12 samples would be taken in 1 hour. The Owens Jet Dust Counter was developed in 1926 to replace the Kotze sampler for use in coal mines. Its advantage was that it humidified the air sample before depositing it on an unvaselined slide, to which the dust adhered by the condensation of the moisture. The instrument was manufactured by C.F. Casella of London and is shown in Figure 5.5.

The 1957 and 1967 dust standards for coal mining in New South Wales are compared in Table 5.1. Approximately 5 500 sets of 12 samples were undertaken each year resulting in enormous effort, both in underground shifts and in laboratory microscopy by the specialist staff of the Joint Coal Board – a body established in 1946 through joint legislation of the Australian and New South Wales Governments. The Joint Coal Board was vested with statutory powers to control, regulate and assist the coal mining industry and to supervise and promote the health of coal miners throughout the state.

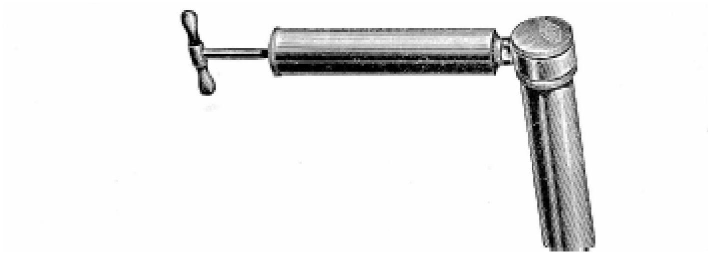


Figure 5.5 Owens Jet Dust Sampler
(Source: Moss, 1927)

Table 5.1 The 1957 and 1967 dust standards in NSW
(Source: NSW legislation and Phillips, 1984b)

Free silica content of parent rock, %	**1957** Average dust concentration (in particles less than 5μm) not to exceed	**1967** Average dust concentration (in particles less than 5μm) not to exceed
<0	700	175
10-20	600	150
20-30	500	125
30-40	400	100
40-50	300	75
>50	200	50

No reliable data exist on the incidence of pneumoconiosis prior to the creation of the Joint Coal Board's medical scheme in 1948. Following an initial survey in 1948, six further 3-year studies had been completed by 1980. The results were analysed by Leigh, who showed fewer than 1 in 1000 of working miners had any disability caused by pneumoconiosis (Leigh, 1981). The results presented by Leigh are summarised in Table 5.2. Apart from a rise in the 1970-73 figures (due to a change in classification) there was a continuous fall in the percentage prevalence of pneumoconiosis.

Table 5.2 Percentage prevalence of pneumoconiosis in the NSW coal industry (Source: Leigh, 1981)

	1948	57-60	60-63	63-65	70-73	73-76	76-79
All	16	3.6	3.5	3	4.5*	2.9	2.0
Category 2 or worse	4.5	1.0	0.6	0.4	0.13	0.08	0.06
% Workforce examined	10	85	85	60	96	100	100

* Rise due to change in classification technique in 1971; estimated comparative figure is approximately 3.0%

Wiles and Fairclough went further than Leigh and stated that, in the 30 years from 1948 to 1978, no man who entered the NSW coal industry without previous dust exposure and with a clear chest X-ray had developed a disabling degree of pneumoconiosis (Wiles and Fairclough, 1978).

Because of a literature study I had undertaken and published (Phillips, 1980) and the fact that I was using a SIMSLIN dust monitor to undertake work for individual mines, I was invited in February 1982 to become a member of the Joint Coal Board's Standing Committee on Dust Research and Control. It was against the above background of success in combatting pneumoconiosis using the almost obsolete Owens Dust Pump that I was asked to prepare a short note on gravimetric sampling for discussion at a subsequent meeting of the committee. This highlighted four compelling reasons for considering a change to gravimetric monitoring.

1. It was recognized that the random nature of spot samples obtained by using the Owens Jet Dust Sampler did not provide adequate or suitable monitoring of the dust created by the highly productive mining machines being introduced into the industry. Gravimetric sampling, on the other hand, could supply a full shift or crib-room to crib-room measurement of the dust concentration to which a man had been exposed.
2. The vast majority of medical research into dust-related diseases being undertaken worldwide used 'weight of respirable dust' as the basic measure of exposure. It became apparent, therefore, that if the industry in NSW wished to continue to take advantage of the findings of overseas investigations, data on local dust conditions had to be compiled in gravimetric form.
3. The success in combating pneumoconiosis in NSW had, in large measure, been due to the extensive use of continuous miners and a system of working in which all personnel were located in intake air. The increasing use of longwalls could lead to a significant proportion of the workforce being exposed to higher dust concentrations. Furthermore, in longwall mining, not only is the dust concentration higher owing to a more rapid excavation rate, but also the pattern of movement of men in relation to the machine is such that it is impossible to effectively monitor the dust level by using the Owens Jet Dust Sampler.

4. On a more practical level the increased size of the industry, together with the difficulty of obtaining spare parts for the obsolete Owens Jet Dust Sampler, had already created some difficulties and would eventually ensure that the particle-counting system could not continue indefinitely.

At this stage the desirability of changing to a gravimetric system was accepted by the Joint Coal Board's Standing Committee on Dust Research and Control, but with the reservation that any new standard adopted should be able to maintain the current low levels of pneumoconiosis. Subsequent to this decision, a research contract was placed by the Joint Coal Board with the School of Mining Engineering at the University of New South Wales, with the twin aims of determining the gravimetric equivalent to the particle-counting standard and of correlating the performance of various gravimetric instruments (Phillips,1984b).

Dust sampling in New South Wales was administered by four regional offices of the Joint Coal Board. Relative to Sydney these were North, North West, West and South. Thirty of NSW's 90 underground coal mines were chosen for inclusion in this research. These mines were selected to embrace as wide a range of seams, geographic locations and dust conditions as possible. This broad coverage was necessary in case the particle size distribution in the range 0-5 μm varied widely from seam to seam, thus affecting the correlation between particle counting and gravimetric sampling.

It was recognised from the outset that correlation of Owens Dust Counts with full-shift gravimetric samples was unlikely to succeed because the Owens method samples the dust concentration for only a very small percentage of the time. Even 100 Owens samples, each taking approximately 2 seconds to perform, would only cover 0.8% of a typical shift. Fortunately, however, a SIMSLIN was available for this study, which enabled both instantaneous and time-averaged readings of respirable dust levels to be obtained in mg/m^3. The SIMSLIN instrument is shown below in Figure 5.6

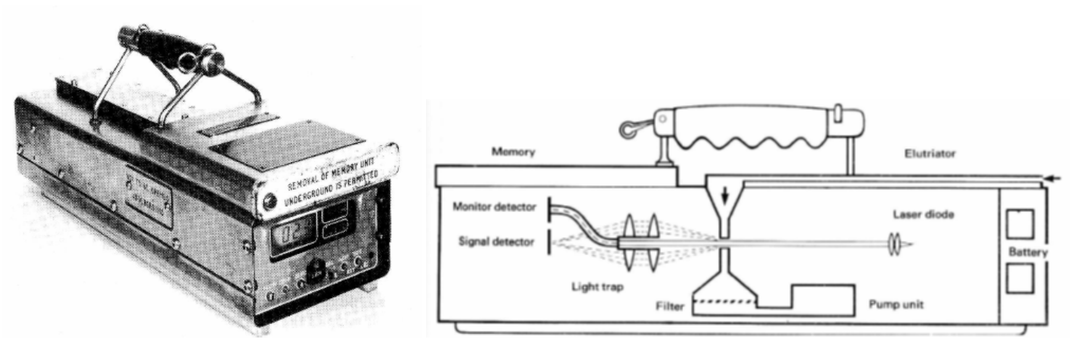


Figure 5.6 A SIMSLIN Mark 2 gravimetric sampler

At each mine the SIMSLIN, together with a MRE Type 113A area sampler and two personal samplers were assembled into a frame, as shown in Figure 5.7, and suspended approximately 1.75 m above floor level and in the centre of the chosen roadway. As the work progressed, it became obvious that the best location for sampling a wide variety of conditions during a single shift was in the return airways of development headings, at a distance of 50 to 100 m outbye of the machine.

Of the 30 mines originally included in this study, 29 were successfully sampled, with an average of 32 pairs of data points being obtained at each mine. The comparison between gravimetric and particle count samples was made by plotting the pairs of data points obtained at each site, the particle count being taken as the independent variable. This avoided the need to perform inverse regressions and allowed the lines of best fit to be established by use of a standard least-squares regression technique. In each case a linear relationship was found, giving an equation of the form:- Gravimetric equivalent = (*A* x Owens Dust Count) + *B*

where *A* and *B* were constants determined for each site.

Correlation coefficients greater than 0.5 were obtained in 23 cases, and for these sites this parameter had an average value of 0.81. The linearity of these graphs having been established, the data were then pooled for each seam, for each district and, finally, for the entire State.

The results showed good consistency over all the seams, which proved that particle size distribution within the respirable range does not vary greatly in the seams mined in NSW. Only in the North West district, where only three sites were sampled, was any significant variation noted. The results on a district basis are presented in Table 5.3. It can be seen that the gravimetric equivalent to the current standard for coal dust- that is, 175 ppcc in the range 1-5 µm was in close agreement for three of the districts and for the same analysis performed on the overall data.

The State-wide result suggested that the maximum permitted concentration of respirable dust, measured gravimetrically, should be set at 3.15 mg/m^3 or lower if the exposure of mine workers was to be retained at its present level. At the 3.15 mg/m^3 level only the North West district would be significantly affected, since this requirement would be somewhat more stringent than the present standard. However, dust counts in the North West were generally low and no difficulty could be foreseen in this district meeting standards set at this new level.

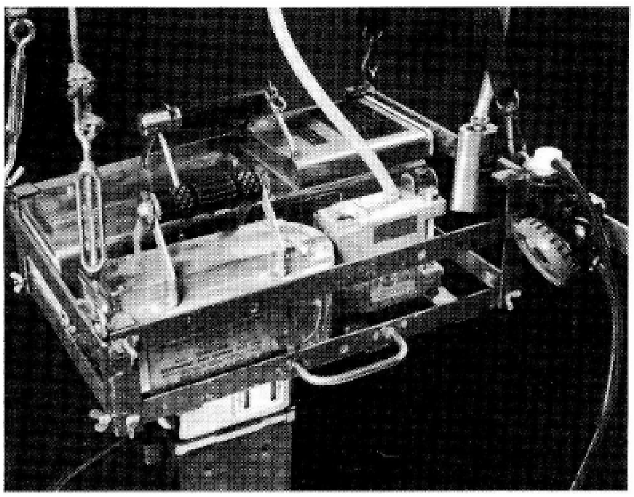


Figure 5.7 Frame containing gravimetric sampling instruments
(Source: Phillips, 1984b)

The research results presented in Table 5.3 were considered to be conclusive and legislation making gravimetric monitoring the prescribed method of undertaking respirable dust sampling in the coal mines of NSW was proclaimed in February 1984.

Table 5.3 Correlation of particle count and gravimetric sampling results on a district basis (Source: Phillips, 1984b)

District	No. of sites	No. of data pairs	Regression constants		Corr. Coeff.	Std. error of estimate	Gravimetric eq. of 175 ppcc in mg/m^3
			A	*B*			
North	7	321	0.014	1.121	0.83	1.491	3.52 ± 0.17*
North West	3	90	0.029	1.060	0.75	1.803	6.21 ± 0.77
West	5	124	0.015	0.192	0.85	0.847	2.86 ± 0.21
South	14	418	0.013	0.251	0.83	1.370	2.54 ± 0.13
NSW	29	953	0.013	0.804	0.78	1.661	3.15 ± 0.11

This legislation was incorporated into the NSW Coal Mines Regulations Act and specified that, for low-quartz dust, the maximum permitted concentration would be 3 mg/m^3 (i.e. for safety, a small reduction was made from 3.15 to 3 mg/m^3), whereas for high-quartz dust the maximum was set at 0.15 mg/m^3. The definition of 'high-quartz' referred to samples of respirable dust containing more than 5% by mass of quartz. The approved method of dust sampling, as specified in the legislation, involved sampling the dust exposure of at least five men at each working face. The men selected in bord-and-pillar mining should include the continuous miner driver and deputy. Dust-sampling equipment would be subject to the approval of the Chief Inspector of Coal Mines, and the initially approved apparatus should consist of an air-sampling pump capable of drawing 1.9 l/min through a hose, a pulsation damping device, a miniature cyclone and a membrane filter 25 mm in diameter. The dust standards also required that the dust-loaded filter would be subjected to an X-ray diffraction determination of quartz content, either as part of a random sampling procedure or on the advice of the dust- sampling officer.

In preparation for the change to gravimetric sampling it was felt by the Joint Coal Board that some of the more practical aspects of routine gravimetric sampling required further research and evaluation and so, in the period February-May, 1983, a small pilot study was conducted at three mines in the South district. Du Pont P 2500A pumps and Casella cylones were used to simultaneously sample five men at each face, all of which were working in the Bulli seam. At two of the mines both longwall and continuous miner units were sampled. This pilot study provided, for the first time in NSW, an indication of the relative exposure of different categories of coal face workers (Phillips, 1984b).

Although I left Australia at the end of 1984 to take up my appointment at the University of the Witwatersrand, I maintained an interest in technical developments within the New South Wales coal industry. The 1984 dust standards remained in place in NSW until changes, based on international on-going medical research, were introduced in the New South Wales Government Gazette No. 185 dated December 21st 2007. This changed the limit for coal dust to 2.5 mg/m^3 and for quartz containing dust to 0.12 mg/m^3.

5.3 An early example of gravimetric sampling in South Africa

Continuous miners were first introduced into the South African coal industry in the mid-1970s and the numbers in use grew rapidly. By 1979 there were 34 in use and by 1981 the number in use or on order was approaching 100. These machines replaced the drill-and-blast method of mining, where production personnel were, quite obviously, withdrawn from each heading at the time coal excavation occurred. In addition, before mechanisation the black workforce was usually employed on 6, 9 or 12 month contracts. This meant that although respirable dust levels were monitored and annual reports produced, there was no perceived threat of the emergence of large numbers of cases of coal workers' pneumoconiosis. However, the introduction of continuous miners fundamentally changed this situation. Firstly, the use of continuous miners necessitates workers being present in the production heading while coal excavation is taking place. Secondly, the time and effort required to train workers to be safe and effective operators of machines such as shuttle cars and continuous miners results in them becoming career employees likely to spend many years working underground.

During 1981, when I spent 6 months of my sabbatical leave working at the Chamber of Mines Research Organisation (COMRO), the Mining Operations Laboratory had been asked to investigate the effectiveness of dust suppression systems on continuous miners. Since I had gained experience in respirable dust monitoring using a SIMSLIN dust monitor (see section 5.2 above) this was one of the three research projects on which I worked and prepared a research report (Phillips, Pedroncelli and Ribbink, 1981).

The experimental work was conducted at two mines; Cornelia Colliery, near the town of Vereeniging and Matla Colliery in the Witbank coalfield. At Cornelia Colliery the effects of operating additional banks of dust suppression sprays at a number of different water pressures was investigated, while at Matla the influence of machine mounted fans and on-board booster pumps was evaluated.

The respirable dust concentrations were monitored at the continuous miner using three instruments: a Modified Thermal Precipitator, a M.R.E. type 113A gravimetric sampler and a SIMSLIN Mk II gravimetric and optical sampler. The two gravimetric samplers were well known instruments and have been described in section 5.2 above. However, the Modified Thermal Precipitator was an instrument unique to the South African coal industry and requires some description.

In the early 1950s the original Thermal Precipitator (TP) was seen as a possible replacement for the Witwatersrand konimeter, which had been in use since 1916. However, the TP was bulky, required considerable skill to operate and samples had to be counted under a microscope. Because of these limitations, the Chamber of Mines of South Africa developed a Modified Thermal Precipitator

(MTP), together with a photoelectric assessor to replace particle counting by a microscopist. Between its introduction in 1956 and the gradual introduction of personal gravimetric samplers in the late 1980s and early 1990s (Department of Minerals and Energy, 1994) the MTP, as represented in Figure 5.8 was the instrument used for dust sampling in coal mines.

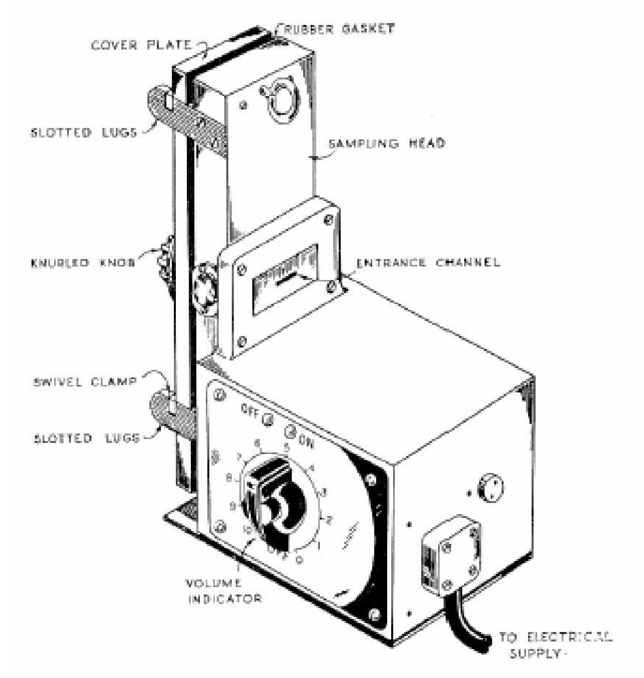


Figure 5.8 Modified thermal precipitator
(Source: MVSA, 1987)

“the MTP samples continuously at 10 ml/minute for a period selected by the observer ranging from one minute to 10 minutes. Air enters the instrument through a small horizontal elutriator then passes through a narrow gap between a glass slide and a thin nichrome wire stretched transversely across the slide and 0.2 mm from it. Respirable particles entering the zone of influence of the hot wire migrate towards and adhere to the cool surface of the slide. By moving the slide forward up to 12 samples can be deposited on the slide. Normally the sampling period is 10 minutes but longer or shorter periods can be used when lighter or heavier dust concentrations are anticipated” (MVSA, 1987).

The slides were evaluated using a photoelectric assessor, the general view and schematic of which are shown in Figure 5.9

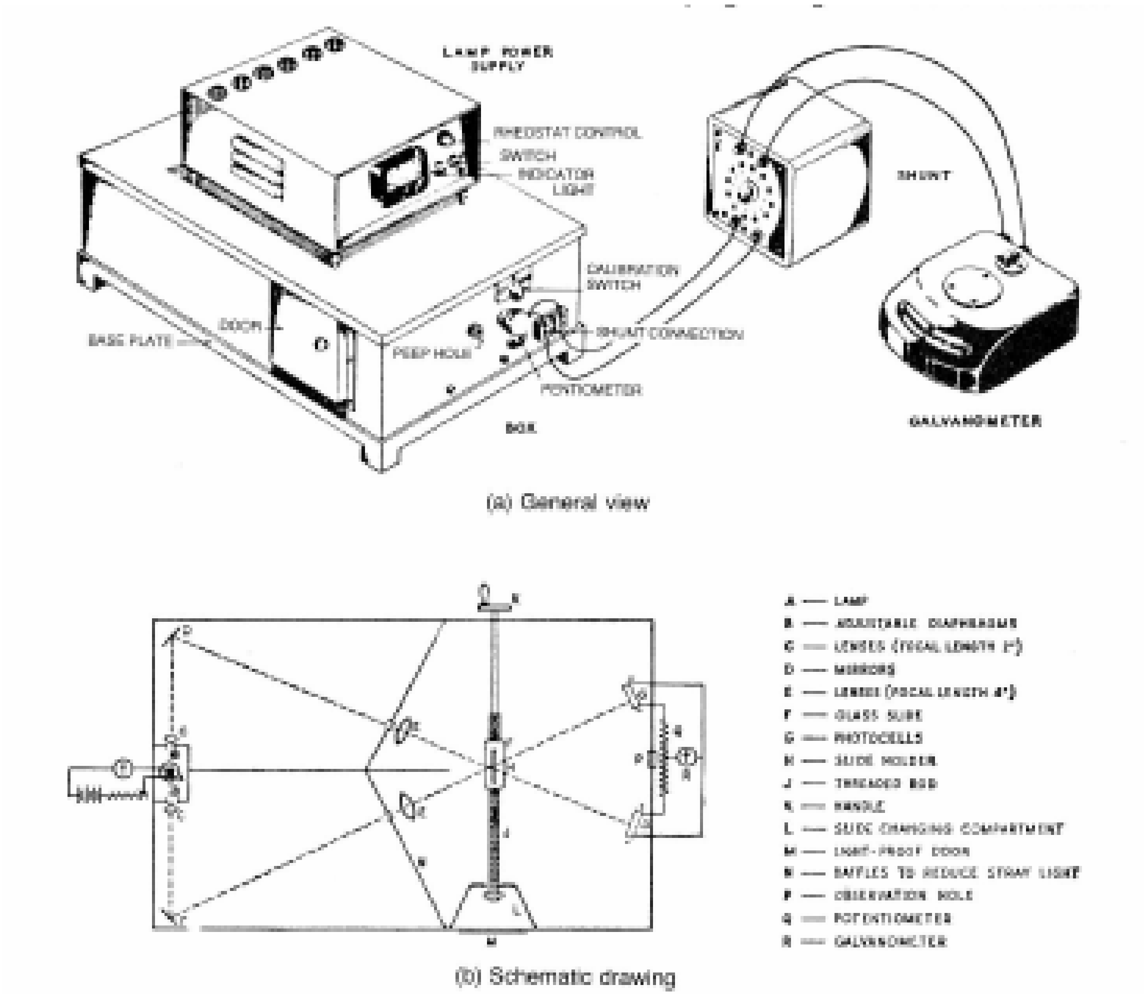


Figure 5.9 Photoelectric assessor
(Source: MVSA, 1987)

"The photoelectric assessor consists of a light-proof box in which two matched beams of light are beamed diagonally at photoelectric cells mounted at the far end of the box so that the beams intersect just before they pass through a MTP slide. One beam goes through clear glass while the other goes through one of the dust samples. The diminished intensity of one beam reduces the output of the photocell in question and the magnitude of this imbalance is measured with a galvanometer. After some adjustments this output is quoted as a dimensionless photoelectric reading (PER)" (MVSA, 1987).

Although the SIMSLIN had been purchased by the Collieries Environmental Control Service some time previously, it had never been used. The optical system of this instrument provides both an instantaneous readout and a shift average. In addition, there is a true gravimetric result obtained from pre- and post-weighed filters. The optical system needed to be calibrated for each of the two mines, since dust particle shape and reflectance varies from location to location. There was a calibration box that could be attached to the SIMSLIN to bring the optical shift average to the same value as the true gravimetric result. The calibration for Cornelia colliery is shown in Figure 5.10.

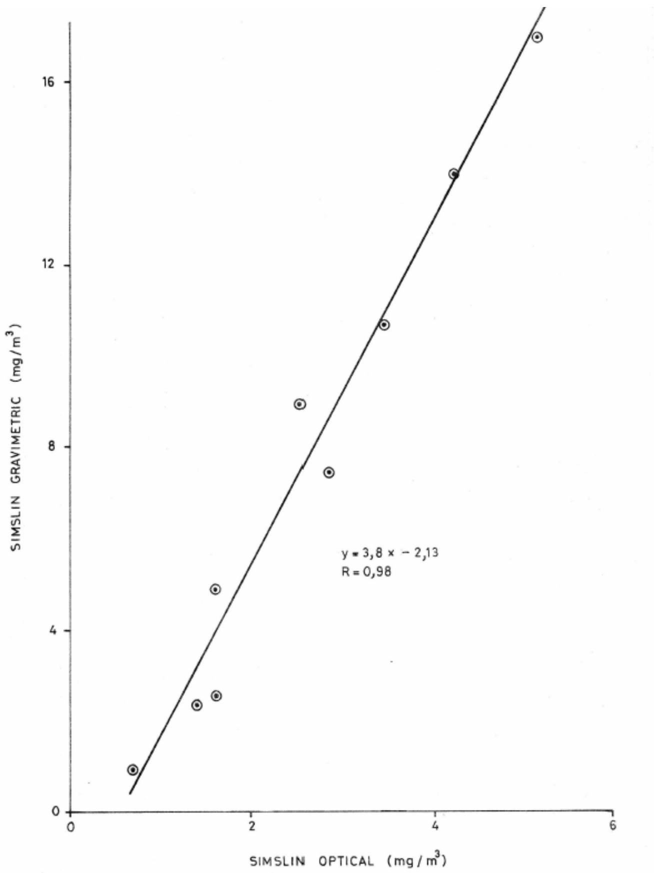


Figure 5.10 Calibration of the SIMSLIN for Cornelia colliery
(Source: Phillips, Pedroncelli and Ribbink, 1981)

The SIMSLIN was an exceptionally versatile instrument and it was possible to use the solid state memory to identify which of the activities in the heading created the most dust. Such a record is shown in Figure 5.11. However, it must be carefully interpreted since there is automatic scaling of the y-axis and the x-axis, which records time intervals, runs from right to left.

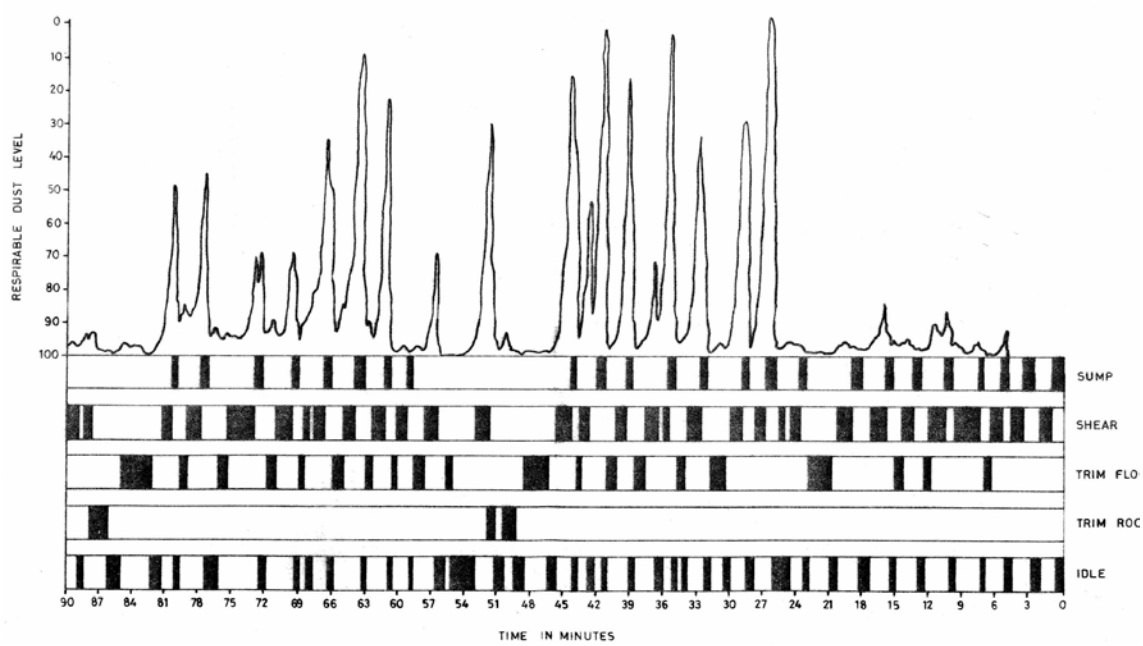


Figure 5.11 Record of the dust conditions at Matla – scrubber fans off
(Source: Phillips, Pedroncelli and Ribbink, 1981)

This investigation yielded valuable results, some of which reinforced the conclusions of other research workers regarding the effectiveness of water sprays, while other data identified issues specific to the South African coal industry. In general, sprays were found to be highly inefficient in suppressing dust once it is airborne. Only when water is applied at the point of dust generation or the coal is pre-wetted will dust levels be reduced.

The trials at Cornelia colliery showed conclusively (Phillips, Pedroncelli and Ribbink, 1981) that the only sprays affecting dust level were those acting on the picks before they entered the coal, thereby carrying water into the cut and preventing the dust from becoming airborne. An increase in water pressure was also shown to have a beneficial effect and two possible reasons for this were suggested. Firstly, at higher water pressures the droplet size generated by the nozzle will be smaller, while the flow rate and consequently the number of droplets will be higher. Hence the chances of capturing an airborne dust particle will increase. Secondly, the increased velocity of the sprays, particularly those directed forwards towards the cutting head, will tend to prevent the dust from moving away from the face towards the driver. This led to a recommendation that the use of venturi sprays, mounted on the cutter boom and directed towards the cutting head, should be investigated.

At Matla colliery, exhaust ducting was employed during the trials and the interaction of machine-mounted fans with the main ventilation system was investigated. Machine-mounted scrubber fans were found to improve visibility and reduce the concentration of coarse airborne dust without apparently changing the respirable dust level. Only by ensuring the exhaust of the machine-mounted fan was directed towards the main exhaust ducting, which should be of considerably higher capacity than the machine fans, could the possibility of recirculation be avoided.

The biggest problem with using the MTP and having data in PER was that very little was known about the correlation between exposure measured in PER and the probability of contracting coal workers' pneumoconiosis. Elsewhere in the world studies had involved gravimetric measurements. Although not strictly an objective of the trials at Cornelia, it was nevertheless possible to correlate the results obtained using the MTP with the data from the SIMSLIN. Figure 5.12 shows the PER values plotted against the calibrated SIMSLIN optical values.

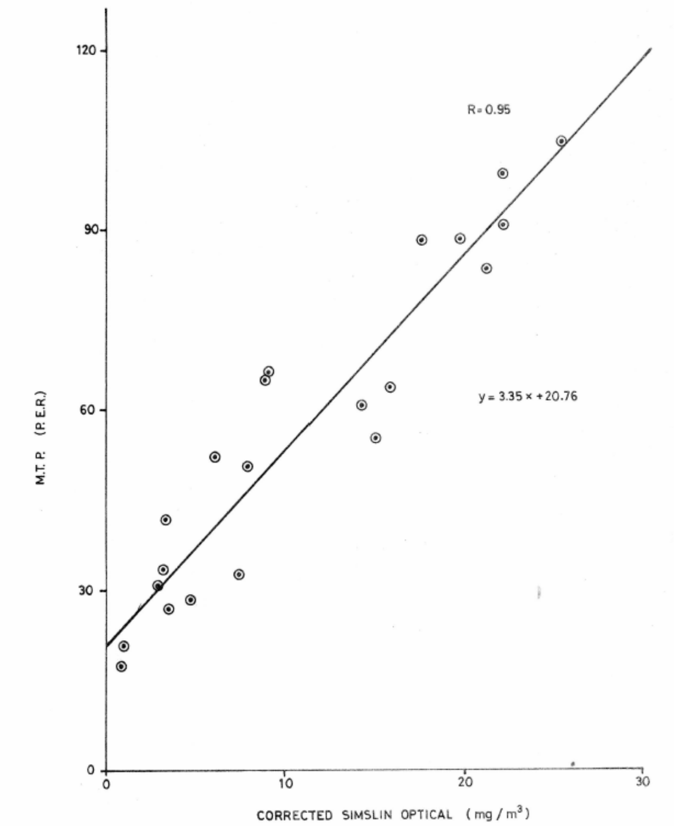


Figure 5.12 Relationship between PER and gravimetric results
(Source: Phillips, Pedroncelli and Ribbink, 1981)

It appears that the relationship is linear with a positive intercept, i.e. that at MTP readings of up to about 20 PER the SIMSLIN does not respond. The previous section of this Chapter describes the programme in NSW of relating particle counting techniques to gravimetric sampling (Phillips, 1984b) and a similar effect had been observed. It is therefore likely that the elutriator of the SIMSLIN has a characteristic curve which rejects more dust than that excluded by the MTP, particularly in the smaller diameter range. This means that the conversion from a PER value to mg/m^3 could not be achieved using a simple multiplying factor.

The relationship established during these tests in the No. 2 seam at Cornelia colliery was PER reading = 3.35 SIMSLIN reading +20.76.

The effect of the constant term is that the conversion factor depends on the dust concentration, as follows:

a) at 60 PER 5.1 PER = 1mg/m^3
b) at 80 PER 4.5 PER = 1mg/m^3
c) at 100 PER 4.2 PER = 1mg/m^3

The last comment regarding this work relates to the dust concentrations occasionally found during the test work conducted in 1981. It was observed that the workforce had very little knowledge regarding the effects of coal dust and continued cutting with blocked or missing water sprays and poor ventilation. Management also continued to argue that they had a workforce on short-term contracts, while this was very obviously changing rapidly. The highest dust concentration measured over a complete shift was 32.5 mg/ m^3, which was totally unacceptable when it was considered that the legal limit in New South Wales was being set at 3 mg/ m^3 and in the United States it was 2 mg/ m^3. This programme of research did much to stimulate the discussion which eventually led to the introduction of gravimetric sampling in South African coal mines.

5.4 More recent respirable dust research in South Africa

Late in 1999 Mr Bharath Belle, a mining engineer working at the Kloppersbos Fires and Explosion Centre, part of the CSIR Miningtek, approached me regarding the possibility of enrolling as a part-time post-graduate. He already had an M.Sc. from Penn State University and research experience with the respirable dust research group at NIOSH and, since the South African coal industry had concerns over respirable dust levels, this became his area of study. In 2000 we began a collaboration lasting to date, which led to Dr Belle's Ph.D. in 2004 (Belle, 2004) and the publication of 6 conference papers, 5 of which were re-written as journal papers. A further journal paper and a SIMRAC research report completed our joint work on respirable dust but we have worked subsequently on other issues.

The most pressing problem in 2000 was the industry's need to comply with the Directive issued by the then South African Department of Minerals and Energy (DME) to reduce the respirable dust concentration to below 5 mg/m^3 (for the sampling period) at the operator's cab position on continuous mining machines. In those days remote operation of machines was relatively rare and the Directive also specified the maximum advance in any cutting sequence should be restricted to 12 m. Whether this was to prevent the operator working under unsupported ground or it was to assist in effective dust control was never clear. The interpretation of the Directive was that the coal face being cut could never be more than 12 m ahead of the most forward point of auxiliary ventilation devices such as a jet fan or force fan ducting.

In order to determine whether or not the 12 m rule would reduce dust exposure, data was gathered from two bord-and-pillar sections, with the individual headings and splits divided into 12 m cutting blocks, as shown in Figure 5.13.

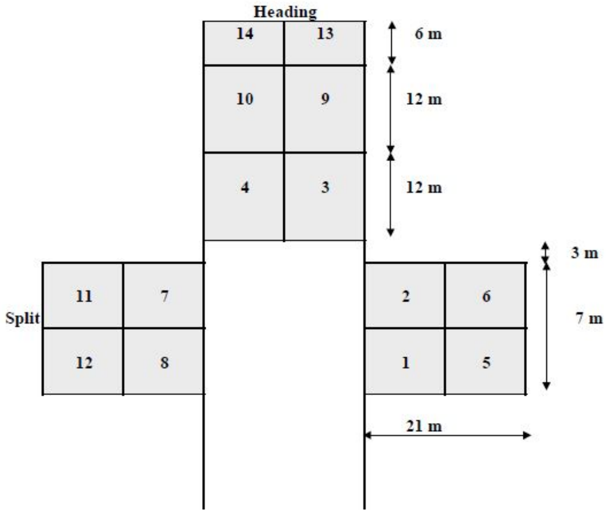


Figure 5.13 12 m cutting sequence in a bord-and-pillar section
(Source: Belle and Phillips, 2001)

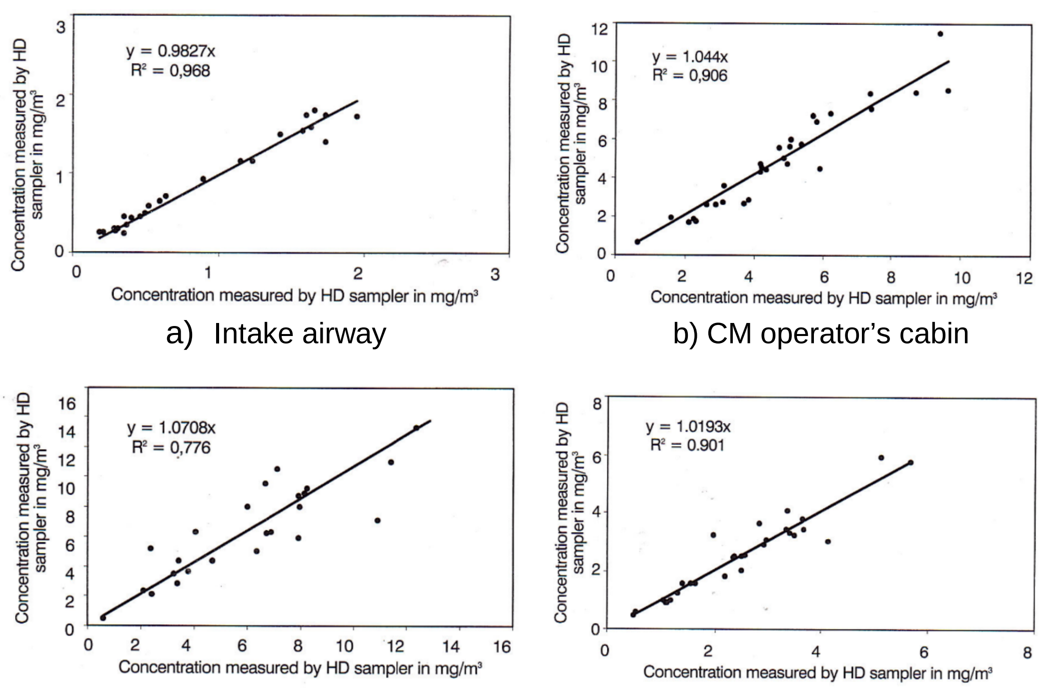
During the underground trials four different dust control systems were used – the half-curtain system, a retrofitted hood system, a double scrubber system and an integrated hood system. For each dust control system, respirable dust levels were measured and compared for 12 m and 24 m deep cuts, i.e. single blocks and double blocks. The sampling instruments used were two personal samplers drawing 2.2 ℓ/min of air through mini-cyclones and a Hund tyndallometer to give an instantaneous indication of dust levels. Results were obtained from over 60 underground sampling shifts and subjected to rigorous statistical analysis.

The conclusions of this study, (Belle and Phillips, 2003), were that a worker positioned inside the cabin of a CM during the cutting of a 24 m block is at a higher exposure risk than a worker when cutting a 12 m block and this was true for all four different dust control systems investigated. However, mere application of the 12 m rule on its own does not solve the respirable dust problem. Mean dust levels for the four dust control systems when cutting 12 m blocks were 4.2, 1.0, 5.5 and 8.5 mg/m^3. This was in sharp contrast to the call by the DME to reduce respirable dust levels to below 5 mg/m^3 and growing evidence from international studies that a level of 2 mg/m^3 should be regarded as the maximum exposure level if the most serious form of coal workers' pneumoconiosis is to be avoided. In addition, the analysis of the dust concentrations at the operator's position indicated the need for meticulous and regular maintenance, together with the application of available state-of-the-art dust control technologies, effective dust control strategies and best practices. In other words, there was an obvious need for a sustained research effort in South Africa to reduce worker exposure to respirable coal dust.

In order to assist mines in their efforts to reduce respirable dust levels by introducing new technologies, it was increasingly necessary to measure dust concentrations as accurately and in as practical a way as possible. At this time mines were required by the DME to submit "engineering samples" i.e. gravimetric samples collected from pre-determined operator positions on continuous mining machines and bi-annual “personal samples” from each section of the mine. Most mines made use of South African manufactured 10 mm plastic cyclones, with a 37 mm filter holder, connected to an air pump which draws 2.2 ℓ/min through the instrument. Measurement of the size-selection characteristics of these cyclones, as tested in the UK (Kenny et al, 1998), confirmed they are similar to those of the Higgins-Dewell (HD) designs commonly used in the UK and elsewhere in Europe.

Comments from industry indicated a scepticism that different results were being obtained from two identical samplers placed in close proximity and the differences were also dependent on the level of the dust concentration being measured. To address this question dust samples were collected in bord-and-pillar continuous miner sections under operational conditions (Belle and Phillips, 2002). Two samplers were positioned in the section intake (40 m outbye of the face), two in the operator’s cabin, two in the face area (far forward on the machine) and two in the section return airway (40 m outbye of the face). Paired samples were defined for this study as two HD type cyclones collecting the dust samples at the specified location for the same sampling period and positioned approximately 250 mm apart.

Sampling took place during 30 underground shifts but for a variety of reasons, e.g. failure of one pump, some data had to be excluded from the analysis. Figure 5.14 shows the raw pairwise data.



a) Intake airway

b) CM operator’s cabin

c. Face area

d) Return airway

Figure 5.14 Airborne dust concentration at four locations
(Source: Belle and Phillips, 2002)

A battery of statistical analyses was applied to the data, with the main conclusions being (Belle and Phillips, 2002)

1) the results of the paired t-test indicated that dust readings measured by two samplers side-by-side show no significant differences
2) it was found that dust concentrations in the sample population ranged from 0.18 to 17.78 mg/m^3
3) the variation in the pairwise results was at its greatest for samples taken close to the face, i.e. at the highest absolute values of respirable dust concentration
4) the perception of significant differences between pairs of side-by-side samplers remained. The bias between sample pairs was determined using the concentration ratio (larger to smaller data). The analysis indicated that the sample biases the measurement by 11.6%, 15.1%, 25,3% and 12% for dust samples taken at section intake, operator's cabin, the face area and the section return respectively, i.e. the higher the absolute measurement the higher the bias.
5) finally, from a practical standpoint the greater bias for high dust concentration values is not a matter of concern as these fail to meet the 5 mg/m^3 standard set by the Department of Minerals and Energy (now the Department of Mineral Resources). The only use of these results is to alert mine management, who would need to take remedial action

During the period 2001-2004 Dr Belle was busy, under my supervision, undertaking the research that would be presented in his Ph.D. thesis. The title of this work was “Development of a dust exposure level index (DELI) for South African underground coal mine workers”. The abstract of the thesis states “This research study had the objective of the development of a pragmatic diagnostic tool (method) to evaluate the exposure of workers. The tool incorporates a set of controllable parameters and influential areas and prioritises them to manage and reduce the worker exposure. The DELI model was to provide critical information as an index, i.e. to show whether the environment is dusty, borderline or relatively free of dust and the effectiveness of administrative and engineering dust control measures” (Belle, 2004).

It was recognised from the outset of this work that different coal types would have inherently different “dustiness” characteristics and this would be an important input parameter for the DELI. However, there was no facility in South Africa to assess this parameter and, in 2002, I obtained a SIMRAC grant to purchase equipment and build a suitable test rig. Since it was likely to be a dirty facility and unsuitable for a university campus, CSIR Miningtek agreed to host it at the Kloppersbos Explosion Testing Site. The laboratory test facility comprised a roll crusher (purchased using the SIMRAC grant to the School of Mining Engineering), located at the intake end of a 0.9 m high by 1.2 m wide, wood framed, hardboard sheet, rectangular wind tunnel 8 m long. An exhaust fan and a dust collector were located at the discharge end of the tunnel. The crusher was a 1.1 kW compact double roll-crusher (79.4 mm diameter rolls) operating at 70 rpm consistently with twenty-four 12.7 mm high staggered teeth on each roll. The roll crusher had a fixed gap of 28.57 mm between the rolls and was designed to produce a product size less than or equal to 15.88 mm (0.625 inches).

The research component of this contract involved collecting over 200 coal samples from 20 operating mines in Kwa Zulu Natal, Mpumalanga, Limpopo and the Free State. Every attempt was made to cover the different seams, which in the main coalfield in Mpumalanga are numbered 1 to 5 according to the geological formation (bottom-up). These samples were crushed under controlled conditions and the airborne respirable dust levels created were monitored at an approximate distance of 2m from the crusher (Phillips and Belle, 2003).

The dust sampling was carried out using a Hund real-time dust monitor and three gravimetric samplers operating at 2.2 ℓ/min according to the ISO/CEN/ACGIH respirable size-selective curve. The Hund real-time monitor continuously monitored the respirable fraction of the dust and was consistently positioned in the middle of the chamber facing the air stream for the duration of the tests. All the gravimetric dust sampler inlets were positioned facing towards the airflow. Preliminary crushing trials indicated that the dust chamber air velocity to maximise dust concentrations for quicker mass collection on the filters was 0.8 m/s and this was the velocity maintained throughout the tests.

A schematic diagram of the Inherent Respirable Dust Generation Potential (IRDGP) test facility is shown in Figure 5.15, while a photograph of the same facility is shown in Figure 5.16.

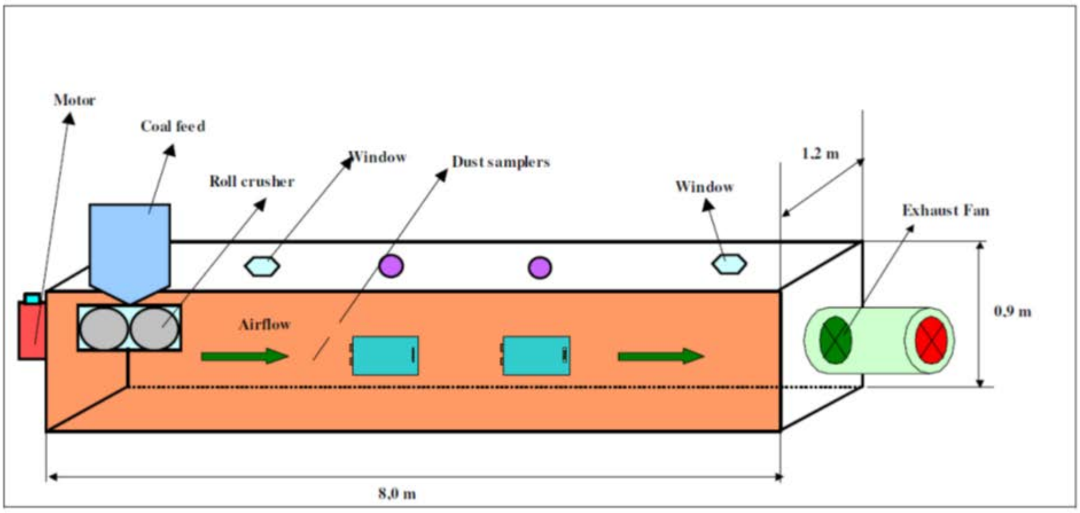


Figure 5.15 Line diagram of the IRDGP test facility
(Source: Phillips and Belle, 2003)

Figure 5.16 Photograph of the IRDGP test facility
(Source: Phillips and Belle, 2003)

“In summary the following conclusions were reached from the IRDGP data obtained from the test samples

- For the first time, a clear delineation of coal types (Bituminous and Anthracite) that possess the most inherent respirable dust generation potential was possible.
- Average coal crushing time of coal samples for the study indicated that the crushing time decreases in the order of seams 1 to 5. Kwa Zulu Natal coals took the highest crushing time during the tests when compared to the other coal seams and coal types. The reasons can be attributed due to inherent coal properties of high rank anthracite coals.
- Inherent silica content of South African coal seams indicated that average inherent silica for the test coals was 3.54%. Similarly, historical data from airborne coal dust samples when analysed for quartz has indicated that the results were below the limit of detection of X-ray spectrometer. However, caution must be exercised when assessing exposure specifically in the presence of sandstone bands and for roof-bolt operators.
- Based on the measured respirable dust data of the South African coals, it can be concluded that majority of the coal mining provinces such as Mpumalanga, Free State and Limpopo have on average similar IRDGP, while Kwa Zulu Natal coal samples which are semi-anthracite type coal have greater IRDGP” (Phillips and Belle, 2003).

Belle continued to develop his Dust Exposure Level Index and this involved a very significant programme of underground sampling, since the index required an understanding of peak exposure levels, dust levels in intake air and also in the return airways. As data became available several joint papers were written (Belle and Phillips, 2006), (Belle and Phillips, 2007a), (Belle and Phillips, 2007b) and (Belle and Phillips, 2009). However, my involvement in this work was that of an academic supervisor and my contribution to these four papers was, at most, 10%.

The final paper in this series dealt with the very practical problem of limestone dust contamination of respirable dust samples for exposure assessment (Belle and Phillips, 2013). Stone dust has been used for coal dust explosion prevention in South African coal mines since the 1970s and the use of stone dust will be discussed in detail in Chapter 6 of this thesis. Pulverized limestone dust (with less than 5% silica content is commonly used due to its wide availability, low cost and the fact that it does not appear to pose any significant health hazard, apart from skin irritation. The adopted TWA (8-h) value for stone dust, as published by ACGIH (2002) is 10 mg/m^3 provided the material contains no asbestos and less than 5% of crystalline silica. In South Africa the exposure limit for particles not otherwise classified (PNOC) has been set at a lower value, i.e. 3 mg/m^3.

Improvements in the methods used to apply stone dust resulted, in 1998, in new regulations regarding its frequent application to freshly exposed coal faces. Most coal mines apply stone dust in continuous miner sections prior to the start of the morning shift. However, due to logistical reasons, it is a frequent occurrence that stone dusting takes place at the start of the morning shift while preparing to cut or even during coal cutting. This has frequently resulted in stone dusting operations taking place during times when respirable dust sampling was also in progress, resulting in overestimation of exposure. In response to this issue, some quick and basic research was requested to evaluate typical levels of contamination and to provide guidance on the minimum time interval between stone dusting and sampling if this problem is to be avoided.

Since limestone dust does not pose a health hazard, its respirable dust levels have not been studied intensely so there was no known protocol or quick analytical method for quantifying the presence of stone dust in gravimetric samples. The method selected for this research was to use a three-member team with each person having access to a pDR (personalDataRam), an instrument that uses light scattering as a measurement technology to obtain instantaneous readings correlated with time and stored in internal memory. A time study of activities in the continuous miner section was also undertaken.

The personal samplers were switched on at the start of the approximately 12 minutes of stone dusting and the results obtained are given in Table 5.4. The sampling time, mass of respirable size dust and the calculated concentration for each of the three samplers are presented in Table 5.4 as “Original” results. The data were then further analysed to distinguish between the mass of coal dust collected and the mass of limestone dust and these results are reported as line items in Table 5.4.

The effects of the short period (approximately 12 minutes) of stone dusting at the start of a working shift were to overestimate coal dust exposure by 13.9% for sample A, 28,6% for sample B and 63.5 % for sample C and give an average exposure level of 12 mg/m^3 for those 12 minutes.

Table 5.4 Effects of stone dust on respirable dust sampling
(Source: Belle and Phillips, 2013)

Sample Type	Sample Parameter	Person		
		A	B	C
Original	Time, min Concentration, mg/m^3 Mass, mg	303 2.05 1.36	300 1.48 0.97	300 1.57 1.03
Stone dust	Time, min Concentration, mg/m^3 Mass, mg	9 10.23 0.20	11 10.56 0.25	12 15.22 0.40
Coal dust	Time, min Concentration, mg/m^3 Mass, mg	294 1.8 1.16	289 1.15 0.73	287 0.96 0.606

The instantaneous respirable dust levels as recorded by the three pDR instruments are shown in Figure 5.17. The high levels of limestone dust during the first 12 or 13 minutes of sampling can be seen, so the influence of stone dust contamination is obvious.

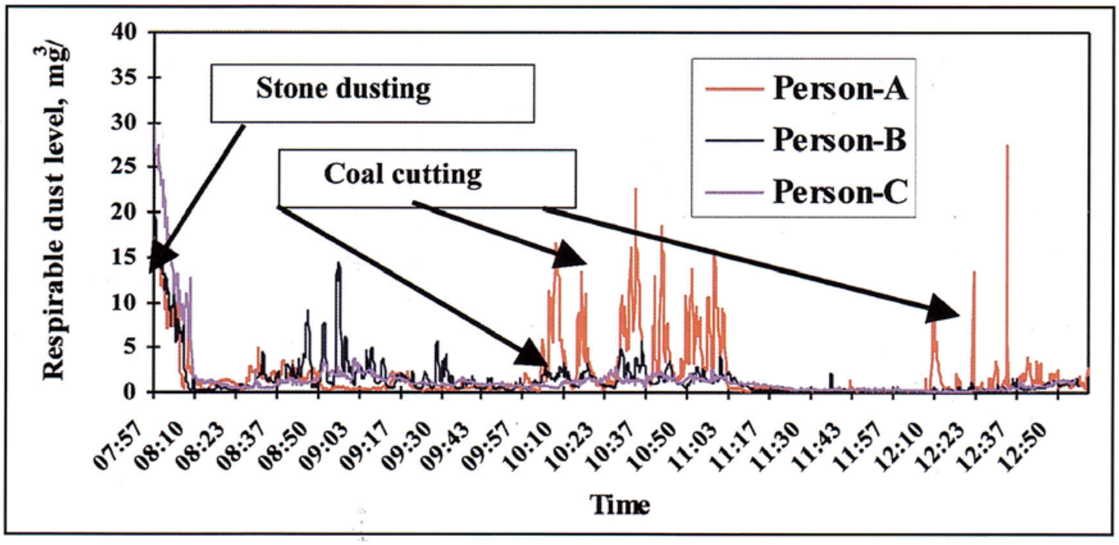


Figure 5.17 Instantaneous respirable dust levels

The paper (Belle and Phillips, 2013) also provided details of the size distribution of the limestone dust used underground and makes estimates of the settling rates of different size limestone particles. However, these are topics best addressed in Chapter 6, which deals with the prevention of methane and coal dust explosions.

CHAPTER 6 RESEARCH INTO METHANE RELEASE RATES IN COAL MINES, TOGETHER WITH THE PREVENTION OF METHANE AND COAL DUST EXPLOSIONS

6.1 My research in Australia

My appointment in late 1977 as a lecturer in the School of Mining Engineering at the University of New South Wales (UNSW) was to teach coal mining subjects and to conduct research in that specialised area. Until then the academic staff, except for the Head of School Professor Frank Roxborough, all had backgrounds in metalliferous mining. Barely 21 months after my arrival a methane explosion at Appin Colliery on 24th July 1979 killed 14 men (Kinninmonth, 1981). While I was not involved in the Appin inquiry, Kembla Coal and Coke (Pty) (Ltd), who had recently opened an adjacent mine (West Cliff Colliery), approached UNSW because of the severe methane outburst problems being experienced. These were later summarised in the recollections of a continuous miner driver of that time "We developed a code of precautions. We trained miner drivers to recognise such warnings as changes in the coal during mining, gas fluctuations on the monitors, a red tinge on the roof or "stretch marks" on the roof. These indicated a mylonite zone was coming. We added the precaution "If you are not sure, DON'T". It was considered the best thing to do if a miner driver was not sure about what was in front, was to get the specialised crew which later became known as the "bomb squad", to mine through the zone. In the big outburst, we loaded out 35 cars and still had more loose coal sitting there. Because the zone was straight across the heading, and with the miner head up, a lot of coal would blow under the head and push the miner back about 4m. In this case 9 brattice props were blown out. There was a compressed air blanket in the cab operated on demand by the miner driver" (Condran, 2009).

These were obviously extremely hazardous conditions which would continue until enough "pit-room" had been created to allow an extensive methane drainage scheme to be installed. To gain some understanding of the problem a recent mining graduate employed on the mine was tasked with undertaking research on a part-time basis and I became his supervisor (Hayes, 1983)

After a thorough review of the literature on the origin, composition and retention of gases in coal, followed by a literature review of the gas content of coal seams, gas emission into mine workings and the principles of gas drainage, the experimental work necessary to define variations in seam gas content across the accessible parts of the West Cliff lease was undertaken. The principal objectives of the laboratory work conducted by Hayes and myself were

a) to determine the equilibrium sorption isotherms for Bulli seam coal samples at a range of temperatures and moisture contents using both methane and carbon dioxide as the sorption gases and

b) to obtain some quantitative information on the rates of adsorption and desorption of methane and carbon dioxide for West Cliff coal.

The experimental work included carbon dioxide studies since West Cliff Colliery was reasonably close to mines such as Metropolitan Colliery, where violent carbon dioxide outbursts had occurred in the past.

In order to achieve these two objectives a single piece of equipment was designed and manufactured in the workshop of the School of Mining Engineering, UNSW. A schematic layout of the apparatus is shown in Figure 6.1.

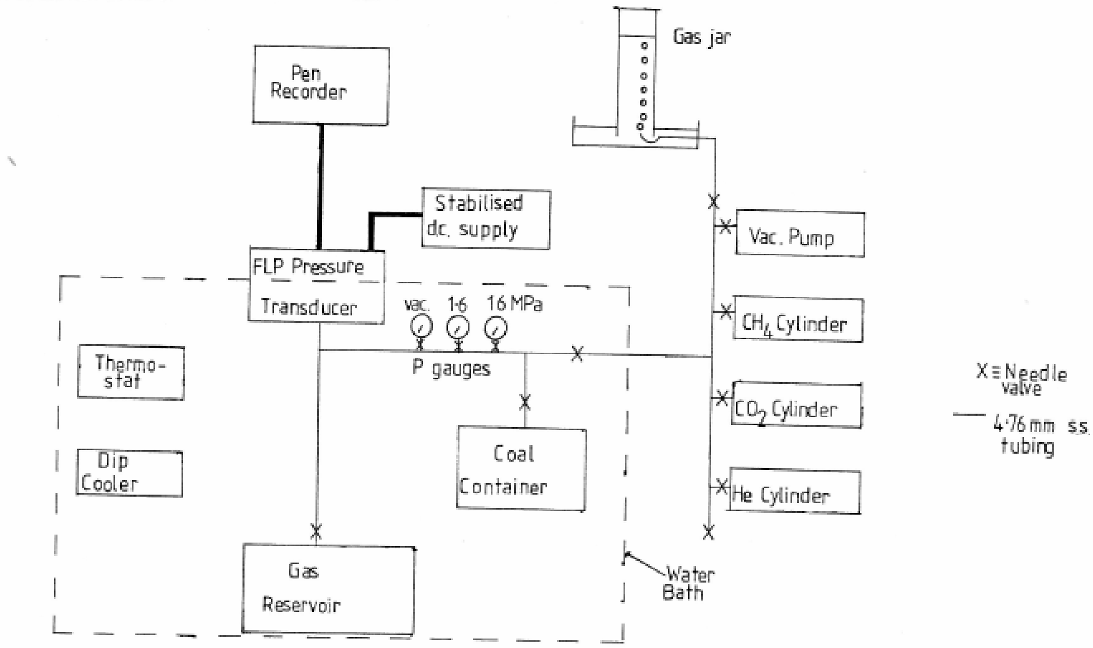


Figure 6.1 Schematic representation of experimental apparatus
(Source: Hayes, 1983)

The coal container had a volume of approximately 90cc, while gas cylinders had a capacity of 1 litre. The gas cylinders, coal container and gauge stems were all immersed in a constant temperature stainless steel water bath, with a useable temperature range from 5º C to 60ºC. Figure 6.2 shows the assembled apparatus before being placed in the water bath. The numbered components were :-

1. Coal sample container
2. Gas reservoir
3. Pressure gauges
7. Pressure transducer

At the time of this work there were only four development panels in the mine and a total of 40 samples were collected. Where possible strip samples were taken from the face and sealed in air-tight plastic containers. When this was not possible lump samples were taken at the face and enclosed in plastic bags until used. All samples were ground and sieved to obtain a range of faction sizes from -47 µm to 5 mm. After oven drying samples were placed directly in the coal container, while the correct quantity of water was added to obtain moist samples and this was taken up by the coal under vacuum.

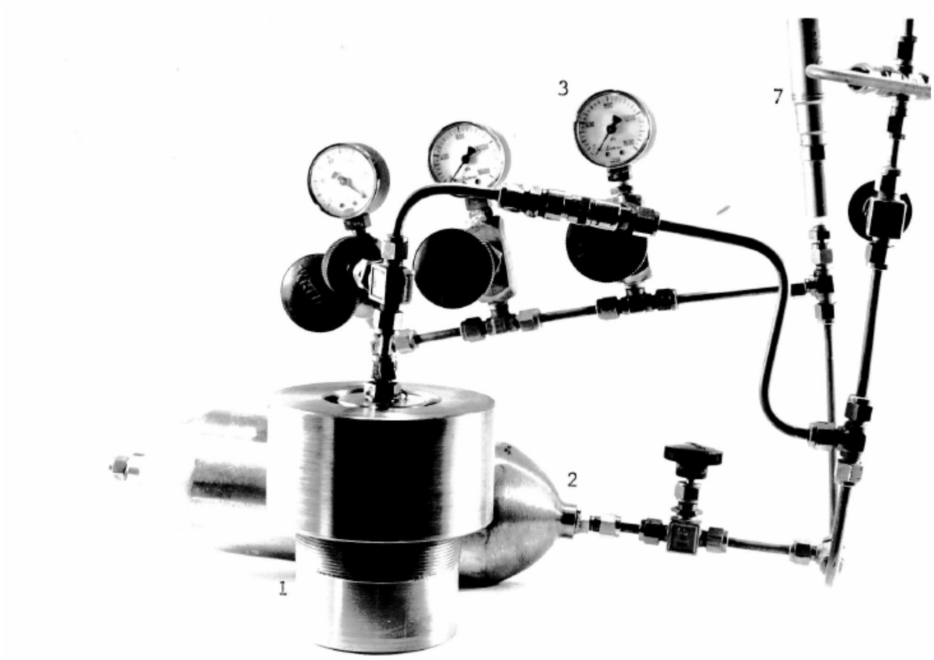


Figure 6.2 View of the apparatus

The experimental work allowed the Langmuir isotherms to be plotted and the equations which described them were obtained. This was done for CH_4/coal and CO_2/coal systems. The monolayer saturation volumes for the Bulli seam (the volume of gas adsorbed at infinite pressure) were calculated as 27.8 m^3/tonne and 47.6 m^3/tonne for CH_4 and CO_2 respectively.

The rate of sorption for coal from the Bulli seam depends principally upon the particle size of the sample and is relatively independent of gas pressure, the temperature within the narrow range of temperatures tested (10 to 30ºC), the moisture content and whether the process was adsorption or desorption. Indeed, particle sizes above 3 mm had little or no effect on the desorption rate of methane. From this it could be concluded that, for gas desorption, the seam can be considered as multiple basic units, each of 2-3 mm in size and that gas is desorbed from the basic unit into the micro-fractures surrounding it, thence into cleats and then into the mine workings. The limiting factor in gas flow from virgin coal is not the desorption process but the resistance to flow provided by microfractures and cleats.

Particle size dominates the desorption rate to such an extent that the outburst problem could be related to geological structures. Outbursts of coal and gas at West Cliff Colliery all coincided with slip-strike faults along which fine, powdered coal called “mylonite” is found. This is a general term for any gouged material pulverised by tectonic movement. The desorption of gas from these “mylonite zones” is very rapid due to the distribution of predominantly small particle sizes along the fault planes and their ability to desorb gas rapidly. This contributes to the mechanisms and intensity of outbursts.

While Hayes' work provided Kembla Coal and Coke (Pty) (Ltd) with data that proved invaluable to the design of a methane drainage system for West Cliff Colliery, it was recognised that this research had been basic and much more needed to be done. In 1980 I was appointed to the Australian Government Advisory Committee on Gas Related Problems in Coal Mining and I applied for funding from the Australian Research Grants Committee for a substantial grant. This allowed for the full-time appointment of Dr Phillip Lingard, a researcher with nearly 20 years of experience in rheology. Although most of his work had been concerned with the mechanisms of oxygen transport in blood, he had a broad understanding of the physics of sorption of gases. In addition, Professor Ian Doig, of the School of Chemical Engineering, UNSW joined the team because of his expertise in gas reservoir engineering. This collaboration led to the design of an automated five cell triaxial test bed capable of gas-mechanical measurements on coal and rock cores. Between 1982 and 1984 three papers were published detailing the results regarding the permeability of coal (Lingard, Phillips and Doig, 1982) and also the permeability and sorption characteristics of triaxially stressed coal samples (Lingard, Phillips and Doig, 1983) and (Lingard, Doig and Phillips, 1984).

During the time when the triaxial test rig was being designed and manufactured, some preliminary permeability studies were undertaken on samples obtained from three gas-troubled coal mines in Queensland and New South Wales. A simple laboratory permeameter, which accepted 38mm diameter by 40 mm long cores, was used. The core was contained within a thick rubber sheath and air pressure of 500 kPa was applied outside the sheath to provide suitable sealing between the rubber and the coal. A schematic of the apparatus is shown in Figure 6.3.

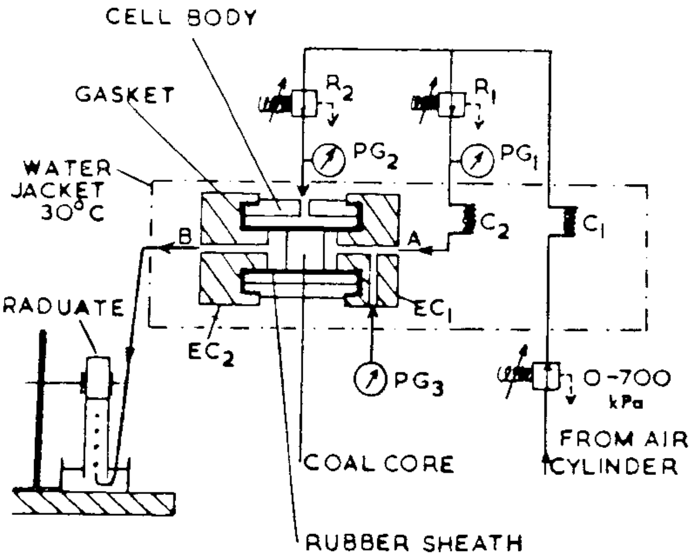


Figure 6.3 Simple laboratory permeameter
(Source: Lingard, Phillips and Doig, 1982)

Differential driving pressures up to 100 kPa were used and the airflow through the coal sample measured by displacement of a water column in an inverted graduated cylinder. Flow measurements, accurate to about 2% were achieved.

Coal lumps were cored both parallel and perpendicular to the bedding plane and discarded if obvious fissures were detected. The permeability of the coal samples tested ranged from 0.1 to over 100 milliDarcy. However, there were obvious differences in the results obtained using coal samples from different collieries. The average permeability increased in the order West Cliff, Appin and Leichardt Collieries. However, the single highest permeability came from West Cliff coal taken from a location immediately adjacent to the outbursting mylonite zone at the colliery (as described at the start of this section).

It was always recognised that permeability was being measured at low confining stress relative to the in-situ stresses underground and that the results obtained would indicate higher permeability than in-situ values. While a knowledge of permeability provides a better understanding of in-seam gas movement and the design parameters of any methane drainage scheme, the data obtained from this work was of a preliminary nature and was subject to many uncertainties. Most significantly the samples had been subjected to stress changes caused by excavation from a region subject to abutment stresses and had been allowed to de-stress. This made the need to develop a facility capable of measuring permeability and sorption characteristics under in-situ stress conditions even more pressing.

Because of the lengthy nature of both sorption and permeability tests it was decided to build a five-cell rig in order to maximise data recovery within the cost and available space constraints of the project. A comprehensive description of the test rig is provided in the literature (Lingard, Phillips and Doig, 1983) and (Lingard, Doig and Phillips, 1984). A general view of the test rig is shown in Figure 6.4.

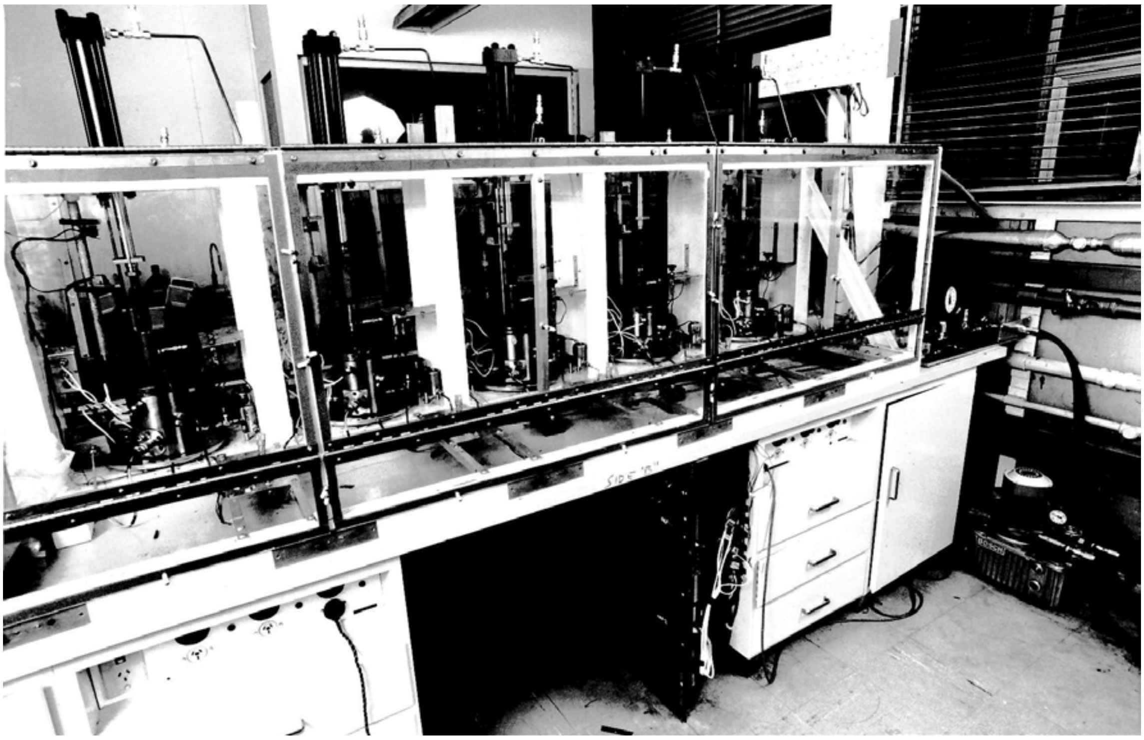


Figure 6.4 Overall view of test rig
(Source: Lingard, Doig and Phillips, 1984)

The heart of the system was the triaxial cell, which could place coal cores under an axial loading while a confining radial pressure was supplied by oil pressure within the cell. One of the five test cells is shown in Figure 6.5.

The design of the triaxial cell reflected the mutual incompatibility of the ideal conditions for sorption-capacity and permeability testing. The long diffusion times for sorption testing increase with the size of core so the smallest standard diamond core, an XR bit, was used to obtain 18.3 mm x 25mm samples. On the other hand, it was felt that permeability testing required a larger core e.g. an EX core of 37.98mm diameter and 50 mm length. The cells could accommodate both size cores with a simple change of platens, piston and outer flanges. The design was similar to the Hoek-Franklin cell with respect to confining the core in a polyurethane rubber sheath. However, it differed in being closed at the lower end, in the arrangement of porting of gas lines and pressure transducers and in the provision of a twelve-pin electrical contact port for interfacing strain gauges.

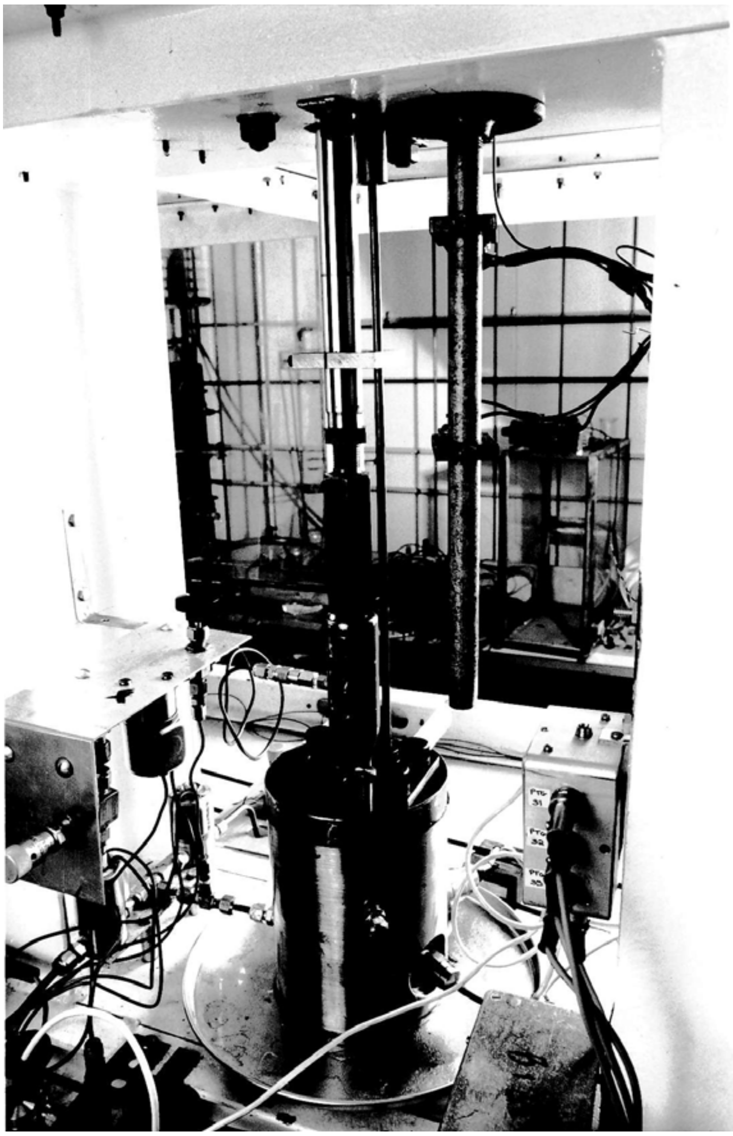


Figure 6.5 One of the five cells
(Source: Lingard, Phillips and Doig, 1983)

The hydraulic force management and gas management systems are described in detail in the paper (Lingard, Doig and Phillips, 1984). The complexity of the design is illustrated by Figure 6.6, which only shows the gas management system

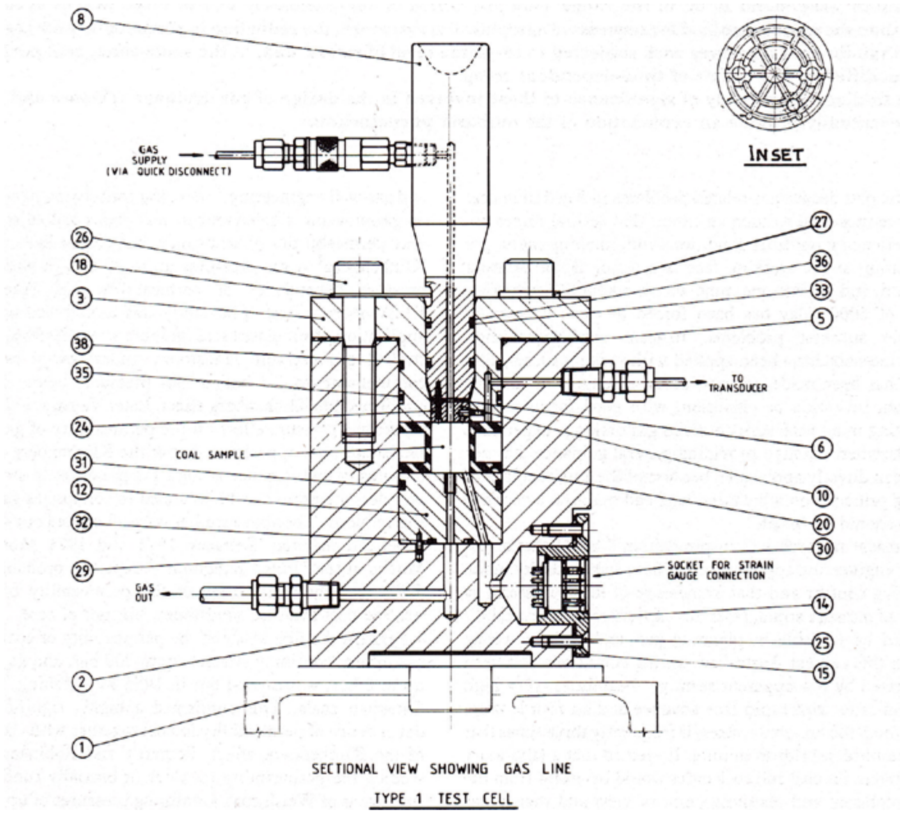


Figure 6.6 Test cell design showing gas distribution system
(Source: Lingard, Doig and Phillips, 1984)

Early experiments with this test rig showed permeability measurements for some coals at high simulated in-situ stress could take several days to pass sufficient gas to make a reliable flow rate determination. For other, more permeable samples or samples under low stress the test could be completed in a matter of minutes. The permeability was calculated using Darcy's equation

$$K = \mu LQ/(A\Delta P_g \cdot \bar{P}_g)$$

where μ is the gas viscosity, (centipoise)
L is the sample length, (cm)
Q is the volumetric flow rate, (ml/s)
A is the specimen cross-sectional area, (cm^2)
ΔP_g is the gas pressure drop across the sample, (bar)
and $\bar{P}_g$ is the mean gas pressure in the core, bar

The results obtained in the early days of using the five-cell rig are illustrated in Figure 6.7, which shows the Klinkenberg effect in coal samples subject to a uniform confining stress of around 100 bars and at mean gas pressures between 12 and 80 bar. From a study of the literature this pressure range exceeded that of previous research elsewhere.

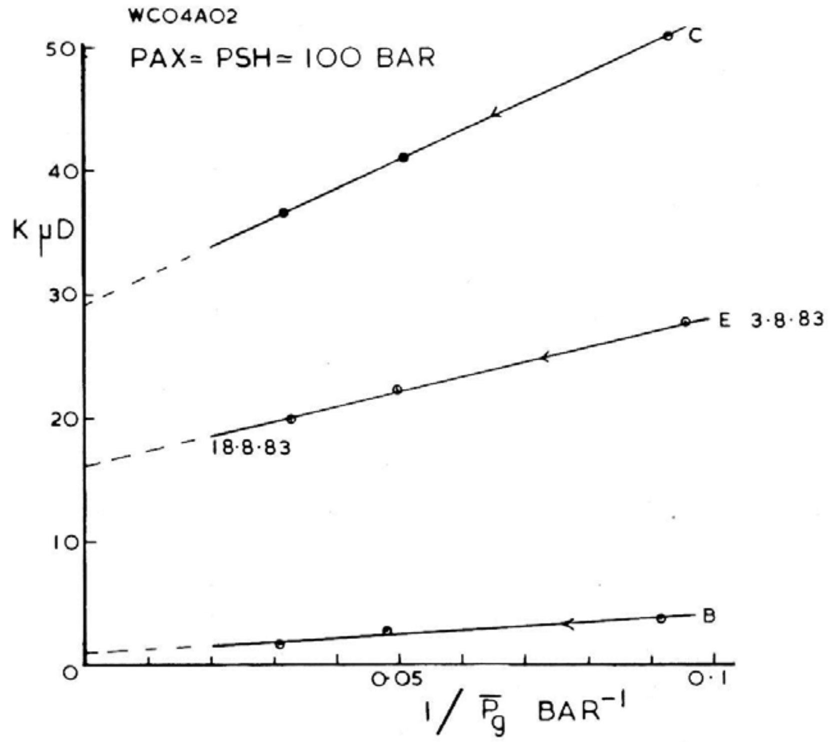


Figure 6.7 Klinkenberg plot of permeability against reciprocal of mean gas pressure immediately after application of stress
(Source: Lingard. Doig and Phillips, 1984)

The dates shown in Figure 6.7 become important because of the effects on permeability of time lapse. Figure 6.7 represents data obtained shortly after the first application of stress to the sample. Figure 6.8 represents data obtained approximately one month later. It is obvious that there is a reduction in the general level of permeability between the two series of experiments conducted on the same coal samples. One explanation of this time-dependent change is that creep with plastic flow has occurred and this was investigated in later experiments. Although this research continued until the end of 1986, I resigned from the University of New South Wales at the end of 1984 and the remainder of the programme was conducted by Lingard and Doig.

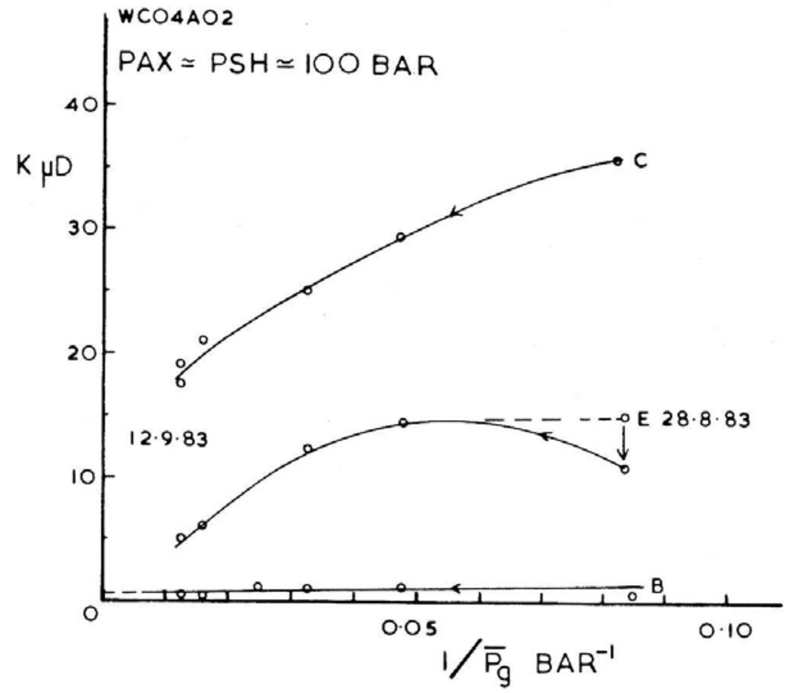


Figure 6.8 Klinkenberg plot of permeability against reciprocal of mean gas pressure one month after application of stress
(Source: Lingard. Doig and Phillips, 1984)

6.2 Mine safety in South Africa

The South African mining industry has a truly horrendous history of mining accidents, with the worst ever being the Coalbrook disaster on the 21st January 1960 when a collapse of room and pillar workings extended over an area of some 300 hectares resulted in the loss of 437 lives (Phillips, 1999). However, it was not only major incidents that contributed to the death toll. In 1970 more than 1000 mineworkers died as a result of workplace accidents and even in 1985, the year I arrived in South Africa, 710 fatalities occurred. Table 6.1 shows that despite improvements a total of 6523 men were killed in the decade 1985-1994 (Phillips, 1996a) and that gold mining contributed over 75% of total fatalities, while coal mining contributed 10%.

Table 6.1 Summary of mining fatalities and persons employed
(Source: Phillips, 1996a)

		1985	*1986*	*1987*	*1988*	*1989*	*1990*	*1991*	*1992*	*1993*	*1994*
Fatalities	*Gold*	540	709	547	509	557	531	461	407	426	372
	Coal	93	67	123	55	54	51	43	46	90	55
	Platinum				38	35	25	37	41	29	24
	Diamond	3	8	12	15	21	19	12	29	21	18
	Other	74	46	52	42	50	38	35	27	19	17
	Total	**710**	**830**	**734**	**659**	**717**	**664**	**588**	**550**	**585**	**486**
Employment	*Gold*	527 947	551 935	553 549	532 558	516 077	489 560	438 990	411 686	394 508	391 447
	Coal	120 959	120 214	114 022	108 584	106 640	103 531	96 197	75 858	61 163	57 430
	Platinum				85 322	86 833	97 019	104 152	103 871	98 781	92 913
	Diamond	18 352	18 474	19 346	19 823	22 017	22 982	21 516	19 654	14 880	15 556
	Other	104 413	91 226	110 562	5 727	10 768	22 017	62 045	54 073	22 017	43 512
	Total	**771 671**	**781 849**	**797 479**	**752 014**	**742 335**	**735 109**	**722 900**	**665 142**	**591 349**	**600 858**

When the data for gold and coal for the period 1988 to 1994 was analysed separately, it was found that four broad groupings predominated, i.e. falls of ground, machinery/transport, fires/explosions and general. However, as Figure 6.9 shows, there were very significant differences between gold and coal mining accident patterns. For example, gold mining, with an average stoping depth of nearly 2000m in 1994 and with the deepest workings at nearly 4 km below ground (see Table 4.1), experienced a very large number of fatal accidents caused by falls of ground and rockbursts. On the other hand, coal mining being more mechanized and prone to methane emissions, had a more even distribution between the major categories of accidents.

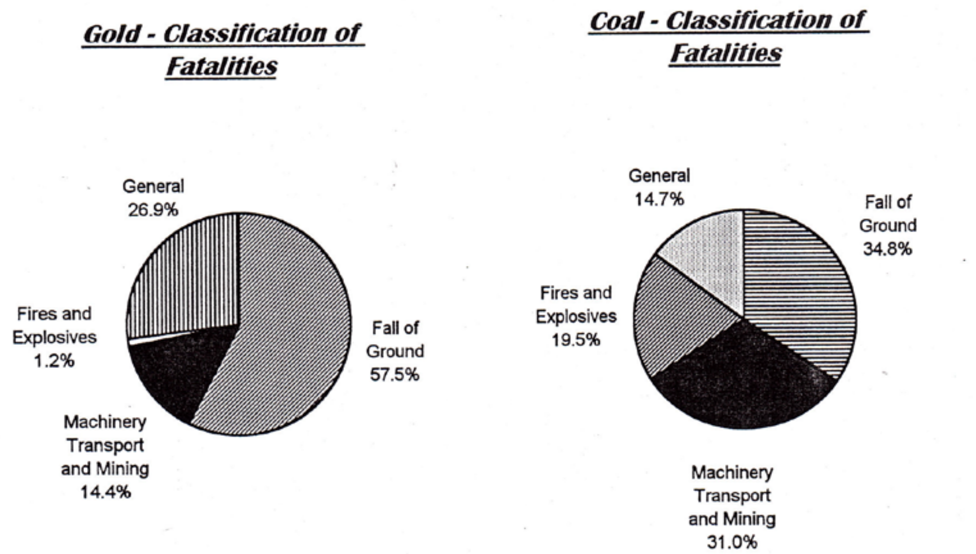


Figure 6.9 Causes of fatalities at both gold and coal mines (1988-1994) (Source: Phillips, 1996a)

The decade covered in Table 6.1 also saw the legalization and growth of the black trade union, the transformation of South African society and an increasing reluctance by politicians and the general public to accept that mining accidents are inevitable. During 1994 a Commission of Inquiry into safety and health in mining (Leon, 1995) reviewed the situation and since then a great deal of attention has been focused on health and safety in the South African mining industry.

It would be wrong to conclude this section without providing more recent statistics showing the vast improvement that has taken place. However, Table 6.2 shows there is still a long way to go before the vision of zero harm is achieved.

Table 6.2 Mining Fatalities 2014-2016

	2014	2015	2016
Gold	44	33	30
Platinum	14	22	27
Coal	9	5	4
Other	15	17	12
Total	82	77	73

6.2.1 History of coal mine explosions in South Africa

The first recorded explosion in a South African coal mine occurred in 1891 at the Elandslaagte Colliery in Natal. Between then and 1992 more than 330 explosions caused the death of 1036 coal miners, while a further 532 were injured (Landman, 1992). From the available data the ten most significant coal mine explosions in South African history have been identified and are listed in Table 6.3.

Table 6.3 The ten worst explosions in South African history
(Source: Phillips, 1995)

COLLIERY	DATE	KILLED	INJURED
Durban Navigation Collieries	08/10/1926	125	0
Natal Navigation Collieries	16/05/1923	78	4
New Marsfield Collieries	31/07/1935	78	0
Hlobane Colliery	12/09/1983	68	8
Hlobane Colliery	12/09/1944	56	13
Middelbult Colliery	13/05/1993	53	0
Glencoe Colliery	13/02/1908	50	5
Burnside Colliery	20/05/1930	38	0
Ermelo Mine Services	09/04/1987	35	11
Middelbult Colliery	12/08/1985	34	7

Table 6.4 Ignitions and explosions (1984–1993)
(Source: Phillips, 1995)

YEAR	IGNITIONS		EXPLOSIONS		
	NO	INJURED	NO	INJURED	KILLED
1984	5	0	2	0	6
1985	6	1	1	7	34
1986	2	-	0	-	-
1987	1	-	1	11	35
1988	1	1	1	5	0
1989	4	0	1	4	1
1990	7	0	2	13	0
1991	5	1	1	17	1
1992	5	1	1	2	6
1993	4	0	1	0	53
TOTAL	40	4	11	59	136

The Hlobane Colliery explosion in 1983, which killed 68 mine workers, was followed by a decade in which there was an average of 4 reported ignitions and an average of over 1 explosion per year. Table 6.4 shows the frequency of these events over the decade 1984 to 1993. More significant is the fact that, in eight of the ten years, explosions caused fatalities and a total of 136 men died over this period.

All ignitions, fires and explosions result from a supply of fuel being mixed with sufficient oxygen and being in contact with a source of sufficient energy for the process to start. In coal mining the fuel is almost invariably methane or a mixture of methane and coal dust, while oxygen is supplied by the air circulating through the underground workings. The causes of explosions are, therefore, categorised according to the source of the ignition energy.

Sources of ignitions can be divided into three major categories, namely frictional, flame and electric ignitions. Frictional ignitions include ignitions due to the cutting picks of continuous miners, road headers, coal cutters or longwall shearers contacting quarzitic rock and ignitions due to stone on stone or stone on steel friction. Flame ignitions include those due to blasting, spontaneous combustion or even heated surfaces. Electric ignitions include ignitions due to electric sparks and lightning forming stray currents.

In order to determine research priorities in the early 1990s, data for the previous three decades was analysed and a summary is presented in Table 6.5.

Table 6.5 Source of ignitions in South African collieries 1984-1993
(Source: Phillips, 1995)

IGNITION SOURCES	**PERIOD**							
	1960s		**1970s**		**1980s**		**1990 to 1992**	
	I	**%**	**I**	**%**	**I**	**%**	**I**	**%**
CM Picks	0	0	1	5,5	14	29,5	6	25,5
CC Picks	1	4,5	3	17	11	22,5	0	0
Shearer Picks	0	0	0	0	1	2	0	0
Stone on Stone	0	0	0	0	4	8	1	4
Blasting	14	64	6	33,50	5	10	1	4
Spon Comb	0	0	0	0	0	0	2	8
Heated Surface	1	4,5	1	5,5	1	2	0	0
Naked Flame	2	9	1	5.5	0	0	0	0
Electricity	2	9	2	11	5	10	0	0
Lightning	2	9	4	22	3	6	0	0
UNKNOWN	0	0	0	0	5	10	14	58,5
TOTAL	22	100	18	100	49	100	24	100

The very obvious change was the decline in ignitions associated with drill-and-blast operations. In the 1960s coal cutters and blasting accounted for nearly 70% of ignitions but, because of mechanisation through the introduction of continuous miners, this had fallen to 4% in the 1990s. Conversely the number of incidents caused by frictional ignitions, while using continuous miners, had risen and many of the 58.5% ignitions classified as "unknown causes" occurred in the face area, leading to a reasonable assumption that these were also caused by continuous miner picks.

6.3 Administration of explosion prevention research

Shortly after my arrival in South Africa, I was invited to join the Explosion Hazards Advisory Committee (EHAC), a sub-committee of the Coal Mining Research Controlling Committee (CMRCC) and I remained a member until the Safety in Mines Research Advisory Committee became operational in 1992. Because of my membership of EHAC, I was asked by the Government Mining Engineer (GME), in August 1985, to visit the Harpur Hill, Buxton laboratory of the Safety in Mines Research Establishment, UK. The purpose of this visit was to investigate whether they would be prepared to test the explosibility of South African coals and to negotiate the costs involved. On my return to South Africa I was informed by the GME, Mr G P Badenhorst, that the CMRCC had decided not to send coal samples overseas but to establish test facilities in South Africa and I was invited to join the Steering Committee. After an extensive search, a suitable site was identified 40 km north of Pretoria in a military area. The design of a 200m long, 2.5m diameter explosion gallery was undertaken by Deutsche Montan Technologie in Germany and constructed from 19mm thick mild steel. This could resist 16 MPa of internal pressure and 21 MPa of total pressure. The closed end was built on a reinforced foundation rigidly fixed to the ground, while the remainder of the gallery was mounted on free-sliding foundations to allow for temperature-dependent expansion as well as expansion during explosion testing.

Figure 6.10 View of the 200m test tunnel
(Source: Du Plessis, 2000)

Funding for the construction of the Kloppersbos site, together with on-going support for its operation came from the Department of Minerals and Energy. By an agreement with the CSIR, management of the facility, including the recruitment and employment of staff, would be undertaken by the CSIR. Following the commissioning of the gallery, a Research Advisory Committee, chaired by Mr Dieter Krueger of CSIR Enertek was formed and I became a member.

Figure 6.10 shows the tunnel during construction in 1987, while Figure 6.11 shows a coal dust explosion exiting the tunnel. The gallery was instrumented to record the static and dynamic pressures of an explosion, as well as the extent of the flame. Twenty instrument stations were incorporated into the design and each was equipped with a piezoelectric pressure transducer to measure the static pressure and a photoelectric sensor to record the passage of any flame.

Figure 6.11 Unsuppressed coal dust explosion
(Courtesy of Mr Isaac Mthombeni)

The South African Minerals Act (Act 50 of 1991) allowed for the creation of the Mine Health and Safety Council, together with a Safety in Mines Research Advisory Committee (SIMRAC) to advise the Council on the need for research based on the safety risk of mines. One of the first tasks of SIMRAC was to identify and quantify the safety risks and, jointly with a colleague Gys Landman, we produced a discussion document on the safety related research needs of the coal mining industry (Phillips and Landman, 1994).

As a consequence of the creation of SIMRAC, the CMRCC and its sub-committees fell away and in 1992 a SIMRAC sub-committee, the Collieries Environmental Engineering Advisory Group (COLEEAG) was formed and my membership was confirmed. This committee had a very wide remit and it was felt that the important topic of coal mine explosions was not receiving the necessary attention. At the COLEEAG meeting on 14th July 1993 this issue was discussed and a decision recorded.

> "DECISION 12/JUL Prof Phillips to convene and chair a Special Interest Group on Explosion Hazards (SIGEH), with membership as agreed at the COLEEAG meeting of 14 July 1993".

Shortly after SIGEH was established I was asked by the committee to prepare a motivation for the funding of a facility to test and develop active on-board suppression systems for continuous miners and roadheaders. The motivation was to be based on preliminary designs and cost estimates. This tunnel will be discussed in Section 6.4 below.

I continued to chair SIGEH until in 1996 when SIMRAC became a true tripartite organization of state, labour and employers and independent membership of committees was no longer deemed acceptable.

6.4 Supervision of explosion prevention research by postgraduates

In the sixteen years, from 1986 to 2001, I successfully supervised 10 research students in the general field of explosion prevention. These 7 M.Sc. and 3 Ph.D. students are listed in Table 6.6

Table 6.6 Research students working on explosion prevention

Name	Degree	Year	Name	Degree	Year
Billenkamp	M.Sc.	1988	Landman	Ph.D.	1992
Stripp	Ph.D.	1989	Cook, A	M.Sc.	1999
Flint	M.Sc.	1991	Genc	M.Sc.	2000
Searra	M.Sc.	1991	Van Dijk	M.Sc.	2000
Cook, P	M.Sc.	1992	Du Plessis	Ph.D.	2001

6.4.1 Laboratory and desktop based studies

My aim with Billenkamp and Stripp was to establish a "methane laboratory" to determine seam gas content and permeability of South African coals and to generate data for the South African coal mining industry. This was a continuation of my work in Australia, as described in Section 6.1 above. However, although the Coal Mining Research Controlling Committee made some funds available, these were insufficient to replicate the sophisticated 5 cell rig built at the University of New South Wales. The laboratory equipment developed was described in a joint paper in 1987 (Phillips, Billenkamp, Wait and Stripp, 1987).

There are two methods of determining the quantity of gas held in a sorbed state within the coal seam. The direct method relies on capturing a "fresh" coal sample, enclosing it within a container and measuring the release rate of gas against time since capture. Since the process of obtaining the sample will, by definition, change the state of the sample, the in-situ value must be determined by extrapolating gas content levels back to a time when the sample was still undisturbed. Although useful results can be obtained by this method, it requires the logistical capabilities of a commercial enterprise or a large research organization to collect samples from underground mines and obtain an accurate estimate of the time that the sample was first disturbed by the excavation process. The alternative or indirect method requires that seam gas pressure and temperature are accurately measured in-situ and that the adsorption isotherms for that particular coal are determined in the laboratory. This was the method chosen for this research programme and involved the re-pressurising of crushed coal samples, at constant temperature, under a predetermined pressure of methane. The heart of this apparatus is the “cell” or coal container shown in Figure 6.12, which was immersed in a temperature controlled water bath.

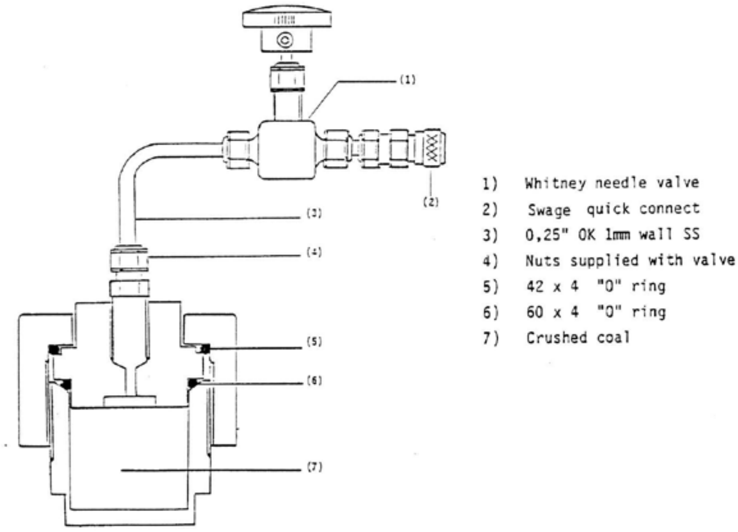


Figure 6.12 Gas sorption cell

Billenkamp tested some 90 samples from 20 mines and obtained the sorption isotherms (Billenkamp, 1988). Billenkamp also wrote software to allow the experimental data to be converted to isotherms and for the easy use of the isotherms to predict real-life methane make. However, this required a knowledge of local in-situ gas pressure and temperature conditions. Many researchers had used hydrostatic head as a proxy for seam gas pressure but in South Africa this was not always valid. In the Witbank/Highveld coalfield the seams are frequently intersected by igneous dykes and frequently capped by dolerite sills, both of which can significantly affect seam gas pressure.

The third aspect of modelling methane flow relies on an understanding of the permeability of the coal seam. This is best determined by in-situ studies but with the limited resources available it was decided to undertake laboratory studies on cores recovered during exploration drilling. Since these samples were de-stressed during the core recovery process, permeability studies must involve the re-stressing of samples to approximately the in-situ condition. Studies in Australia had shown that failure to do so can lead to permeabilities three orders of magnitude greater than those that occur in-situ (Lingard, Doig and Phillips, 1984). In order to achieve triaxial confinement of the sample using simple apparatus, the permeameter developed for the tests was a modified Hoek cell capable of accepting 30 mm diameter specimens and applying radial pressures of up to 10 Mpa and axial pressures of up to 140 MPa. A controlled gas pressure could be generated across the specimen using specially modified platens in the Hoek cell and this ability to control gas pressure allowed the pressure dependence of permeability (the Klinkenberg effect) to be studied. A schematic layout of the apparatus is shown in Figure 6.13.

Initial results using coal cores from the Ermelo, Phoenix and Piet Retief collieries indicated that the bituminous coals were highly permeable, while the anthracites were generally much tighter and had a lower permeability. Typical figures for 'liquid' or true permeability of the bituminous coals ranged from 2 to 3 milli Darcy, while the anthracites gave values of between 1 and 10 micro Darcy at confining pressures of 5 Mpa. As expected, the bituminous coals also showed a high degree of directional permeability, being more permeable in the direction of the bedding plane. These coals also showed that provided a minimum confining pressure of about 3 Mpa is exceeded, the Klinkenberg effect was pronounced, with permeability significantly decreasing at high gas pressures.

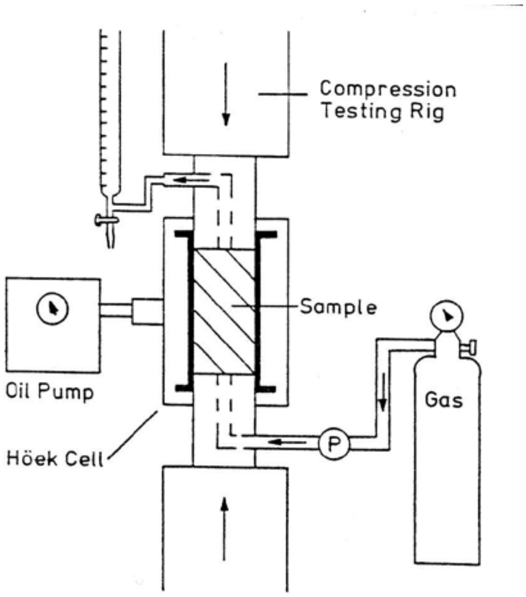


Figure 6.13 Schematic of simple permeability apparatus

With the research group established, further work came from an unexpected source. Following the explosion at St Helena Gold Mine in August 1987, in which 63 people lost their lives, the gold mining industry also showed interest in the explosion prevention research being conducted at Wits. Mines in two of the country’s goldfields (Evander and Welkom) were classified as “fiery mines” but the origin of the gases encountered had never been fully understood. Searra conducted a literature review of both the history of flammable gas occurrences in Witwatersrand gold mines and the influence of geological factors in the Witwatersrand Supergroup (Searra,1990). Several possible sources of methane, hydrogen and helium were identified but the most likely source was the coal measures, sometimes several hundred metres above the gold reefs, in both Evander and Welkom. Searra investigated what structures and mechanisms could be responsible for the migration of methane, against the background that several authors had argued that the natural buoyancy of both methane and hydrogen would prevent them from migrating downwards, while ventilation specialists had spoken about the high negative pressures generated by typical gold mine ventilation systems. However, underground methane occurrences in Witwatersrand gold mines are almost invariably associated with fissure waters and Searra elected to measure methane solubility in 14 water samples collected from 6 mines, together with a control sample of distilled water.

Chemistry textbooks either state that methane is not soluble in water or that it is almost insoluble. While this is true at atmospheric pressure, Figure 6.14, which reflected the solubility of methane in water from the Bracken gold mine, shows that solubility (m^3/tonne of water) increases linearly with pressure but is less pronounced at higher temperatures. The similarity of results obtained for all 14 samples of mine water and distilled water indicated that the composition of the water does not affect methane solubility.

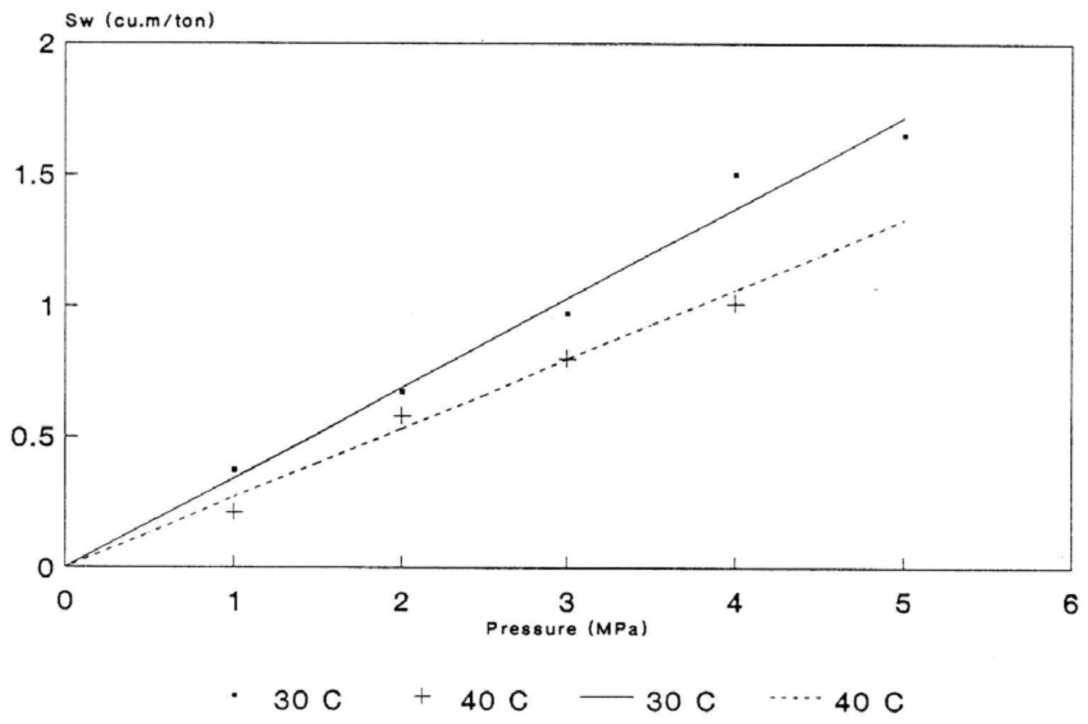


Figure 6.14 Methane solubility in mine water
(Source: Searra, 1991)

By the late 1980s the major coal mining companies were becoming increasingly concerned as the number of ignitions and explosions increased, primarily due to frictional ignitions associated with the use of high productivity continuous miners. In particular, the Anglo American Coal Corporation, led by Mr Dave Rankin, wanted a greater understanding of the causes of ignitions in seams which could be classified as having “low seam gas content” by international standards. A senior Consulting Engineer at Anglo Coal, John Flint undertook this task and registered as an M.Sc. student under my supervision (Flint, 1990).

A major problem facing Flint was that, although there had been 155 known incidents since the first explosion in a South African coal mine in 1891, the reports of Inquiries undertaken by the Mines Inspectorate were confidential and not released to the industry. “It is the author’s view and that of almost every Mining Engineer in the Republic that, following the example of the United Kingdom, the outcome of such serious incidents should be made known to the Mining Industry in an attempt to prevent such incidents occurring again. Unfortunately, this is not the case in the Republic of South Africa and frequent requests to the Mines Inspectorate have met with little success” (Flint, 1990). However, in the late 1980s attitudes were changing, perhaps with a view to future politics but certainly due to the rise of the black trade union (National Union of Mineworkers) which was committed to improving mine safety. This resulted in Flint gaining access to many of the Reports of Inquiries and undertaking a thorough analysis of several past incidents.

Flint’s dissertation runs to over 800 pages and is structured according to the cause of the initial ignition, i.e. frictional heat and sparking, lightning and stray currents, blasting operations and explosives, electrical sparking including faulty electrical equipment and ignitions by flames. He also included a chapter on the relatively rare phenomenon in South Africa of methane outbursts. Each of the chapters had a final section containing the conclusions Flint drew from his study of several Reports of Inquiries. As a person with many years of practical and managerial experience he was able to identify both the root causes of the explosions and the control measures that could have prevented the occurrence of these incidents.

6.4.2 Field based studies

Two postgraduate students undertook fieldwork as part of their research. After recovering cores from a number of South African collieries and subjecting them to adsorption and permeability tests, Stripp concentrated his research on two problem mines, Ermelo Colliery and Majuba Colliery. In order to use the laboratory data, the in-situ temperatures and seam gas pressures needed to be measured.

While temperature readings posed no great problem, gas pressure measurements must be made by sealing a section of an in-seam borehole using inflatable packers. Since the retrieval rate for packers is always less than perfect, i.e. there is a significant risk of losing the packer should the borehole deform, and in view of the high cost of commercially available units, a low cost device was designed by the research group and manufactured in the Department of Mining Engineering’s workshop from locally available components. As the hydrostatic head in coal mines increases at the rate of approximately 1 MPa for each 100 m of cover and overseas experience has shown the ratio of peak seam gas pressure to hydrostatic head is of the order of 0,7, the prototype packers were designed to measure pressures of up to 1400 kPa, i.e. they are suitable for use in mines with up to 200 m of cover. After insertion, gas pressure builds up behind the packer as gas desorbs into the borehole. This continues until equilibrium is reached and seam gas pressure and borehole pressures are the same. A nylon hose, threaded through the packer insertion tubes allowed this pressure to be measured at the collar of the hole. These packers were used extensively at Ermelo Mines (Stripp, 1989) and packers modified to measure higher pressures were used by the Chamber of Mines Research Organisation at the ill-fated Majuba Colliery. This mine was developed to excavate a 3.5 m high seam at a depth of approximately 300 m and to supply 15 Mtpa of coal to the adjacent 4200 MWe Majuba power station. Unfortunately, feasibility studies had missed the frequent dolerite intrusions which resulted in an inefficient mining layout. Strata control problems and high seam gas content exacerbated the situation and the mine was closed before it came into full production.

One of the packers is shown in Figure 6.15, while the installation process, including the nitrogen bottle used to inflate the packer, is shown in Figure 6.16.

Figure 6.15 Complete packer assembly
(Source: Stripp, 1989)

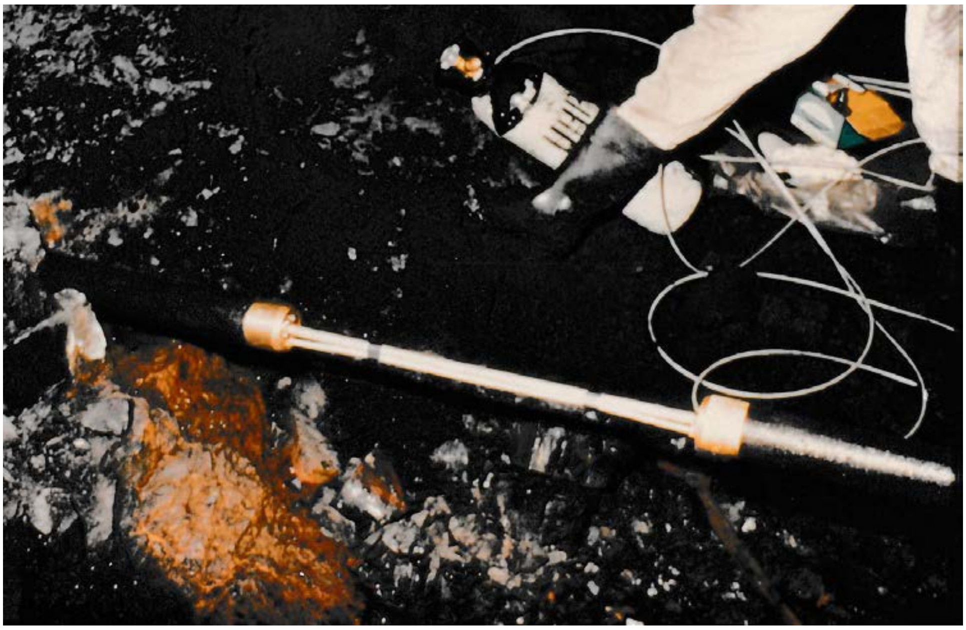


Figure 6.16 Installation of the packer in borehole
(Source: Stripp, 1989)

The results obtained at Ermelo showed the virgin seam gas pressure was approximately 70 kPa and the pressure gradient was such that this was reached at about 22 m into the rib-side. In comparison, the seam gas pressure at Majuba Colliery was often measured at values in excess of 600kPa. Overall, Stripp found that in-seam gas pressure measurements were significantly lower than could be expected by studying the findings of European and United States researchers. This he attributed to the higher in-situ permeability of the coal seams he studied, which would cause exaggerated degassing effects.

The specific interest in conducting investigations at Ermelo Colliery was the direct result of an explosion there in April 1987 in which 35 mine workers perished. The mine owning company, Trans-Natal Collieries (part of the GENCOR Group), supported this research in order to understand more fully the risk of methane accumulations occurring. One particular concern was the influence of main fan stoppages, since the explosion occurred shortly after a fan maintenance period. As part of Stripp's field work he made use of the "METHOBAR", a purpose-built system designed and constructed by GENCOR's Group Electronics Department. This portable system was designed to automatically read and record the following;

1. absolute atmospheric pressure
2. methane concentration of the ventilating air
3. the time and date of each reading

This portable device is shown in Figure 6.17

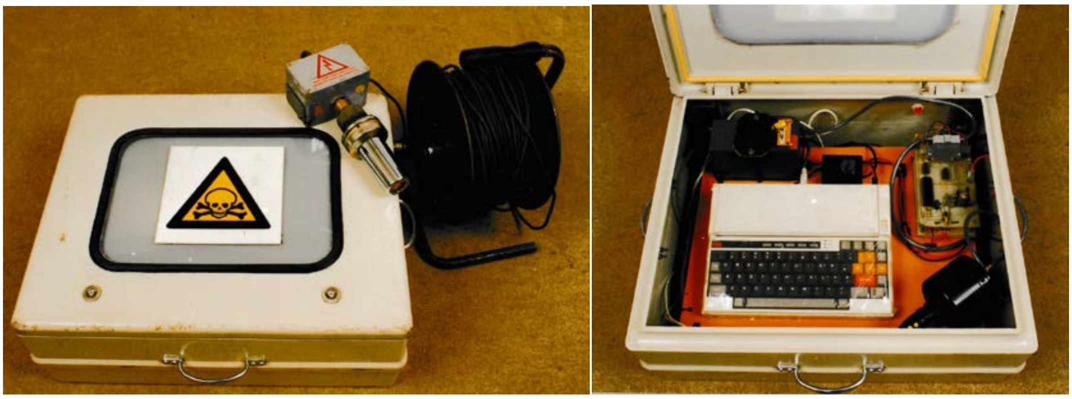


Figure 6.17 The GENCOR made METHOBAR
(Source: Stripp, 1989)

Five sites were chosen to monitor return airflows during non-production shifts. These included panel returns, returns from pillar extraction panels and returns adjacent to partially sealed goafs. The parameters measured additionally allowed the methane make inbye of the instrument to be calculated by multiplying the air quantity by the % CH_4.

The results illustrated how rapidly underground pressures change when a main fan is stopped, being typically 1 -1.9 kPa/hour as compared to variations of 0-150 Pa/hour due to natural variations in surface air pressure. The immediate warning that came from this work was that "although South African collieries are relatively shallow and contain low in-situ gas content seams, rapid build-up of general body methane can occur after fan stoppages. This is probably encouraged by the permeable nature of South African coal seam strata" (Stripp, 1989).

The final contribution made by Stripp was to develop a simple, one-directional model of gas flow through coal. Initially the model made unrealistic predictions but this was overcome by replacing the porosity term with a pseudo-porosity considerably greater than the actual in-situ porosity. The value of this term was determined empirically and took account of the high degree of absorbed gas present within the coal. "the model showed that laboratory permeability measurements are unsuitable in any prediction of methane flow rates. This stressed the need for reliable in-situ permeability measurements in future approaches to methane emission prediction" (Stripp, 1989).

In May 1993 an explosion at Middelbult Colliery killed 53 mineworkers. This methane explosion is described in section 6.7 and it can be said with some certainty that it involved a large quantity of methane. Following the internal inquiry, the owning company, Sasol Mining (Pty) Ltd., awarded a research contract to Itasca Africa (Pty) Ltd. to investigate the quantity of methane likely to accumulate in goafs during pillar extraction operations. This work was conducted by Mr Alan Cook and he used his results to prepare an M.Sc. dissertation under my supervision (Cook, 1999).

A goaf is created whenever one of the high extraction coal mining methods, i.e. longwall, shortwall or pillar extraction is practiced. Once the coal seam is removed the immediate roof strata collapses with subsequent collapse of overlying strata. Stress relaxation in the floor can also result in fracture and heave of the floor. The goaf can, therefore, be categorised as a complex zone comprising of fractured rock, with considerable volumes of void between the rock fragments and the strong possibility of a significant cavity forming above the broken rock. Methane from adjacent seams and remaining portions of the mined seam can collect in these voids. Since methane is lighter than air it will rise in the goaf until capped by any impermeable layer. During mining this gas concentration is controlled by ventilation or drainage with the intention of either diluting and removing the methane by airflow across the goaf or by keeping a high and inert methane concentration away from the working area and accepting that there will be an explosive fringe.

Cook investigated goaf conditions at two of the Sasol Mines, Twistdraai and Middelbult, over a two-year period. Since any goaf is totally inaccessible, gas monitoring was conducted using boreholes drilled from surface before pillar extraction commenced. Permanent 6mm diameter gas monitoring tubes were installed in each borehole, as shown in Figure 6.18, with a backup system of a single tube that could be raised or lowered to "fish" for high gas concentrations. Sufficient slack tube was left on surface to permit movement during strata collapse. Samples drawn from the tubes were passed through four sensor heads (low range methane (0-5%), high range methane (5%-100%), oxygen and carbon monoxide.

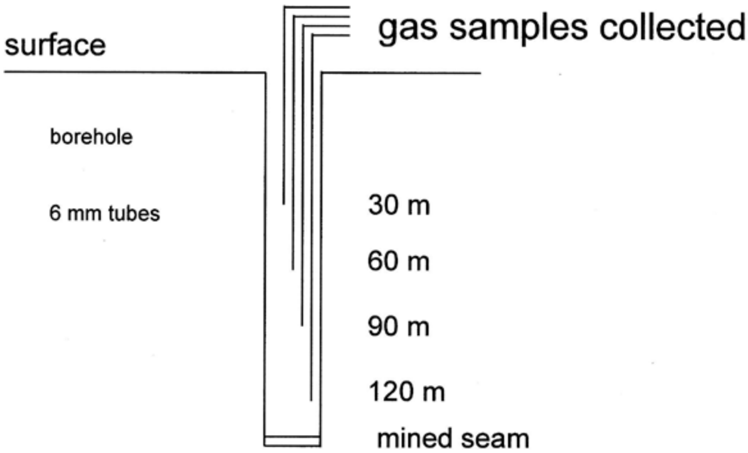


Figure 6.18 4-tube bundle installation at Twistdraai Colliery
(Source: Cook, 1999)

What became abundantly clear is that gas concentrations in the goaf vary spatially and also move with time making any study complex. “At Twistdraai methane concentrations peaked at above 50% within 24 hours of goafing and at less than 20m into the caved strata. Concentrations then decreased to around 30 to 35% before declining over a period of several weeks”. (Cook, 1999). This ageing effect is due to dilution as caving increases the ratio of goaf volume to methane volume.

Cook also showed conclusively that a large volume of methane/air mixture within the explosive range existed in all the goafs he studied but its location moved with time. At Twistdraai there were sufficient boreholes to produce gas plots both along the centre line of the goaf (c/l) and across the goaf. As an example of the movement of the high methane region, centreline plots for the 22nd March and the 15th May are shown in Figures 6.19 and 6.20 respectively. On both days approximately 50% of the goaf contained areas of high methane concentration but this region has moved, over two months, to the other end of the goaf.

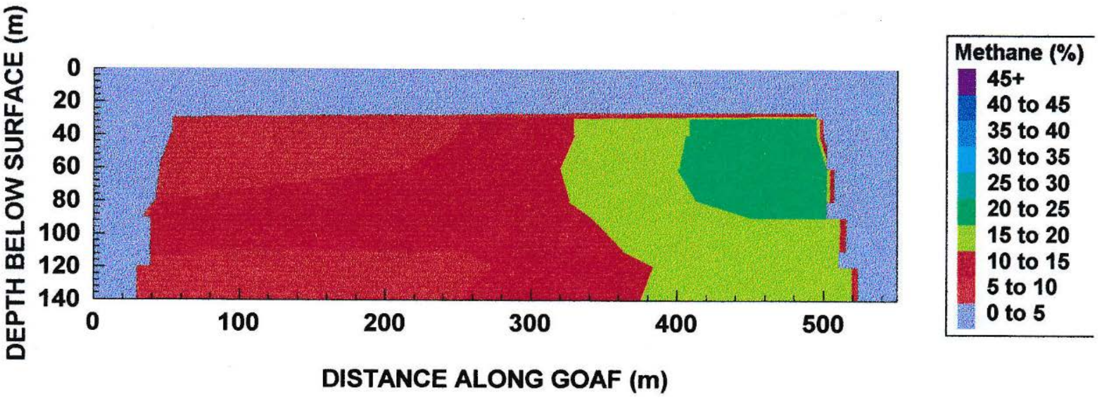


Figure 6.19 Methane concentrations measured on c/l of Twistraii goaf 22/03/99
(Source: Cook, 1999)

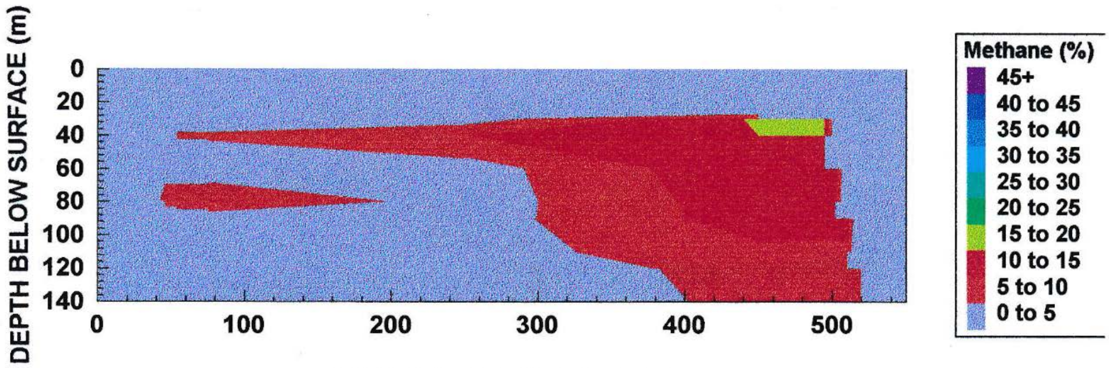


Figure 6.20 Methane concentrations measured on c/l of Twistraii goaf 15/05/99
(Source: Cook, 1999)

By superimposing the methane and oxygen contours and using the Coward Explosibility Triangle, the zone within the goaf lying within the explosive range of atmospheres could be plotted. An example of this is shown in Figure 6.21.

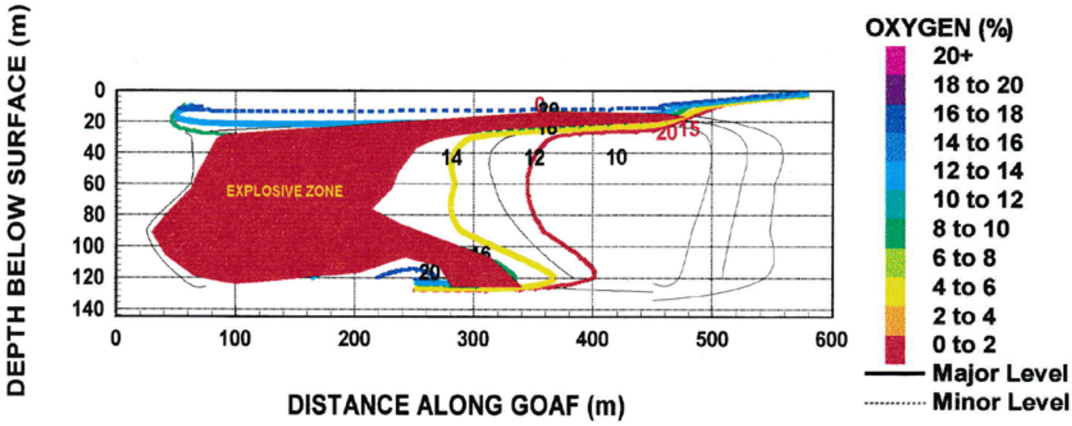


Figure 6.21 Explosive zone on c/l of Twistraii goaf 22/03/99
(Source: Cook, 1999)

At the same time that Cook was working on the gas problem within the goaf, SASOL was undertaking its own research on the Rock Engineering of total extraction mining systems. This work led to estimates that the total final volume (fractured rock and voids) in the goaf being monitored at Middelbult was approximately 70 000 m^3. If the goaf void space is as little as 3%, i.e. 2 100 m^3 and the explosive zone is about 30 % of the total, then the explosive methane/air volume would be about 700 m^3. If exposed to an ignition source this would result in a major explosion and this is further discussed in Section 6.7.

6.4.3 Research at the G.P. Badenhorst Test Facility (Kloppersbos)

As described in section 6.3, I was a member of the Kloppersbos Steering Committee during the planning and construction phases and I was also a member of the Kloppersbos Research Advisory Committee (RAC) for several years after the 200 m gallery was completed in 1987. Although the positions of labourers, a supervisor, an instrument technician and a manager were filled by the transfer of CSIR personnel, the RAC recognised the need for a full-time researcher to be appointed. A young graduate Mining Engineer, Patrick Cook, was available and was employed at Kloppersbos to commission and calibrate the tunnel. In order to maintain a close contact with him, he also registered as an M.Sc. student under my supervision (Cook, 1992).

Since precise dust size distributions are essential in the explosibility testing of coal dusts, Cook's first task was to commission the coal milling plant, which was capable of delivering 150 kg/hour of coal dust. The flow diagram for the coal milling plant is shown in Figure 6.22.

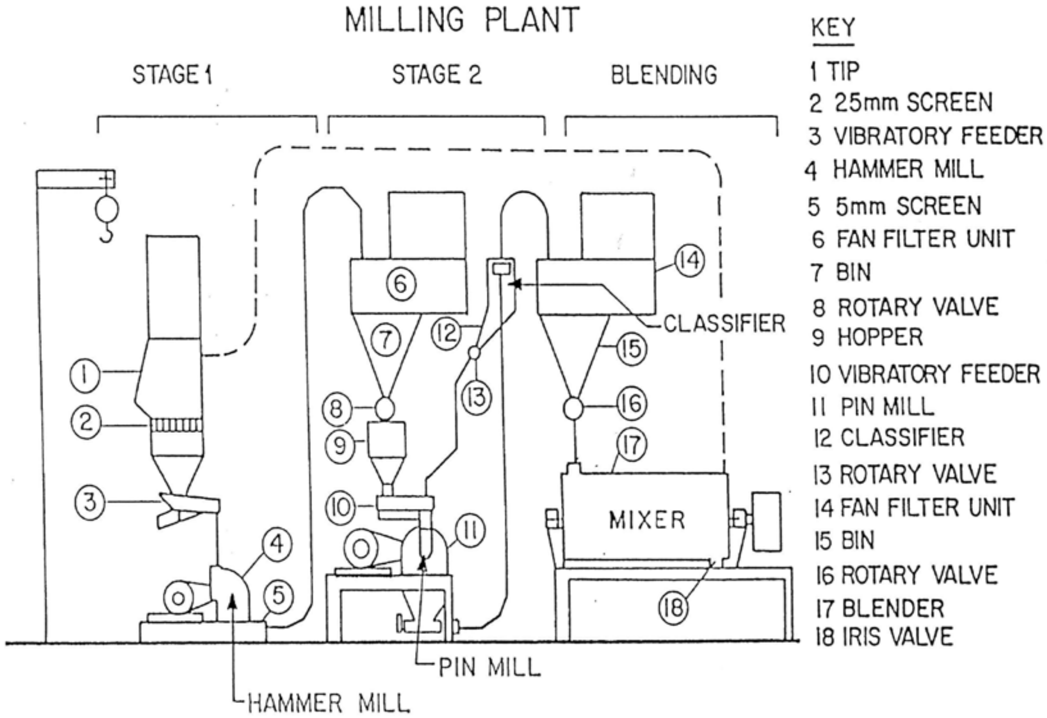


Figure 6.22 Flow diagram of the Kloppersbos milling plant
(Source: Cook, 1992)

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In order to replicate the dust found in coal mine roadways two criteria need to be met

a) in order to produce a natural size distribution the breakage of the coal must conform to a natural breakage, i.e. that achieved during coal winning
b) the median particle size must be 20 µm, with a top size of 150 µm ± 15 µm

Cook achieved the desired size distribution in a range of coals by reducing the feed rate to 125kg/hour to ensure consistent results, by optimising the speed of rotation of the pin mill and by tuning the classifier by balancing the quantity of wash air and speed of rotation.

Once the tunnel instrumentation was commissioned Cook began a series of tests to calibrate the tunnel. The variables available were:-

1. magnitude of CH_4 explosion (volume of CH_4, concentration of CH_4, igniter)
2. length and mass of coal dust deposit
3. placement of coal dust on the floor or on shelves
4. the introduction of a non-dust zone to create a double explosion

These variables were illustrated by Cook, as shown in Figure 6.23

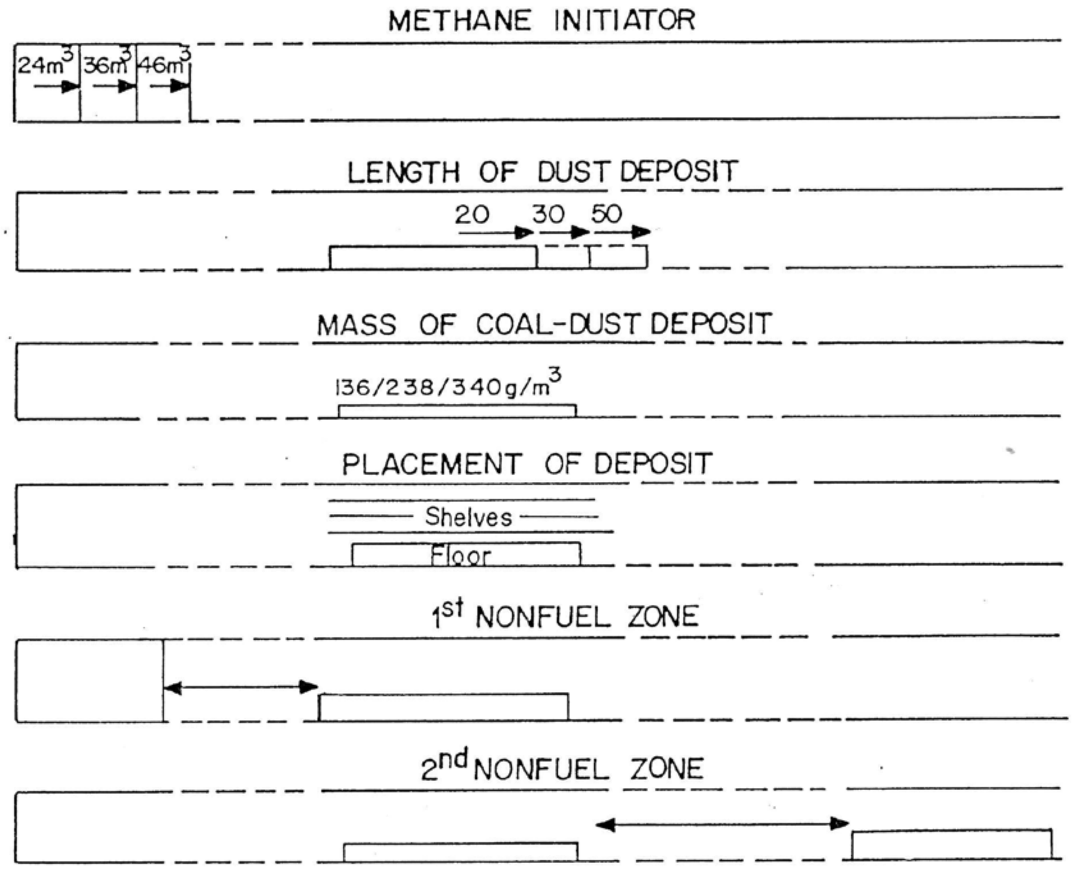

Figure 6.23 Variables within the gallery that required investigation (Source: Cook, 1992)

Because of the location of the tunnel, noise was not a problem and the inherent strength of the tunnel allowed strong explosions to be undertaken. The calibration exercise resulted in the development of 6 explosion protocols, as shown in Figure 6.24, covering virtually all possibilities. This allowed the then applicable stone dusting standards in South Africa to be examined.

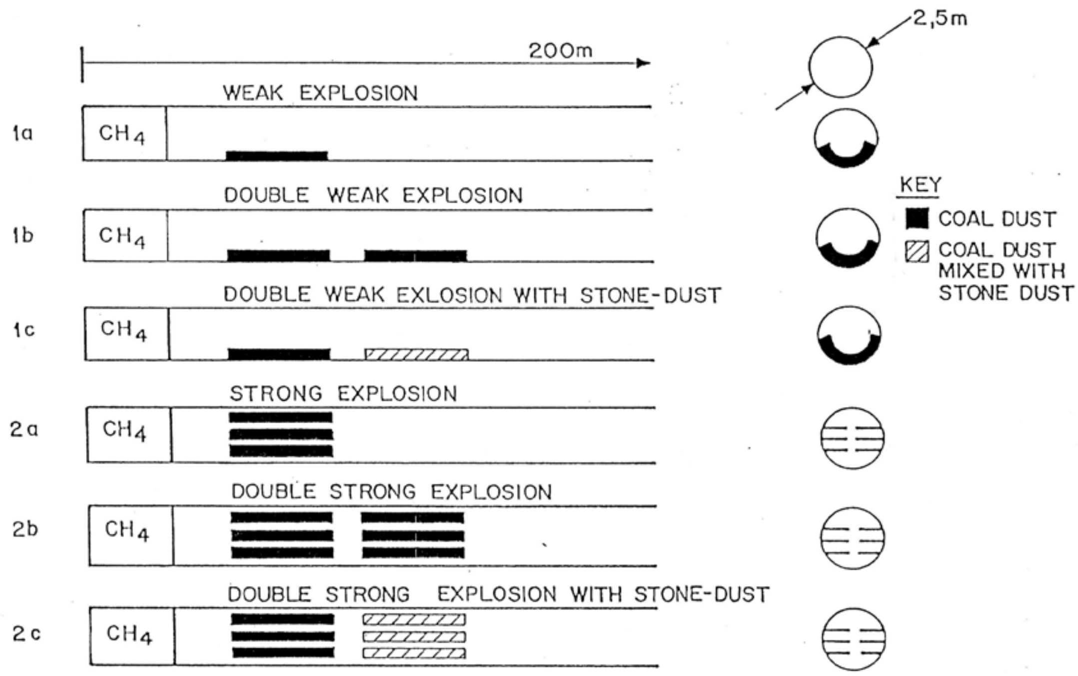

Figure 6.24 Explosion protocols for the 200m gallery
(Source: Cook, 1992)

Stone dusting as a precaution against coal dust explosions was only made mandatory in South Africa following the Wankie Colliery disaster in June 1972. There a series of explosions had destroyed the Number 2 Colliery resulting in the loss of 426 lives. Although this mine was in Zimbabwe (then Rhodesia) it was managed by a South African company. The response from the Transvaal Coal Owners Association was a reluctant acceptance of the 1958 United Kingdom standards, based on the volatile matter content, as shown in Table 6.7

Table 6.7 Stone dusting standards (Regulation 10.24 of Minerals Act)

Percentage by mass of volatile matter content, in the coal being mined	Minimum percentage by mass of incombustible content
14<VM%<20	50
20<VM%<22	55
22<VM%<25	60
25<VM%<27	65
27<VM%<30	68
30<VM%<32	70
32<VM%<35	72
35<VM%	75

The effectiveness of applying these stone dust standards was evaluated at Kloppersbos using the 6 test protocol. Weak and strong, single coal dust explosions were used to establish the baseline for each coal tested, while the introduction of stone dust in the quantity specified in Table 6.7 into the second fuel zone measures its effectiveness. It was found that for many coals used in the investigation the specified level of stone dust did not inhibit explosion development nor did it suppress explosion propagation. This resulted in the Department of Minerals and Energy setting up a Task Group on Coal Mine Explosions. This Group prepared a “Guideline for the Compilation of a Mandatory Code of Practice for the Prevention of Coal Dust Explosions in Underground Mines” (Task Group, 1997). I had oversight of this Guideline to ensure its recommendations complied with the research findings and was asked to sign-off on the final document.

In these Guidelines the concept of a sliding scale, i.e. that shown in Table 6.7, was abandoned and all underground coal mines in South Africa are considered to be at risk of coal dust explosions. The underground workings are considered to be divided into 8 areas, each with significantly different potential for experiencing a coal dust explosion. The minimum percentage by mass of incombustible matter content is 80% in the high risk areas and 65% in all other areas (Task Group, 1997)

6.4.4 Research using the 20 m tunnel

As described in section 6.3, I was involved in the decision to construct a facility at Kloppersbos to be used for the development of on-board active suppression devices for continuous miners and roadheaders. In order to allow full scale tests to be conducted, the dimensions of a typical heading had to be replicated and actual continuous miners were to be used in the tests. The tunnel, as constructed, was 20 metres in length and 7 metres wide, with a variable height roof to simulate seam heights of 2 to 6 metres. A full description of the tunnel, together with its instrumentation, has been provided elsewhere (Oberholzer, 1995). Figure 6.25 is a schematic view of the tunnel. To cater for the conditions that could exist after a continuous miner has complete the first lift, the tunnel had a removable shoulder. This allowed tests simulating cutting with a continuous miner with a 3.5 m wide cutter drum or a full face machine cutting a roadway 7m wide.

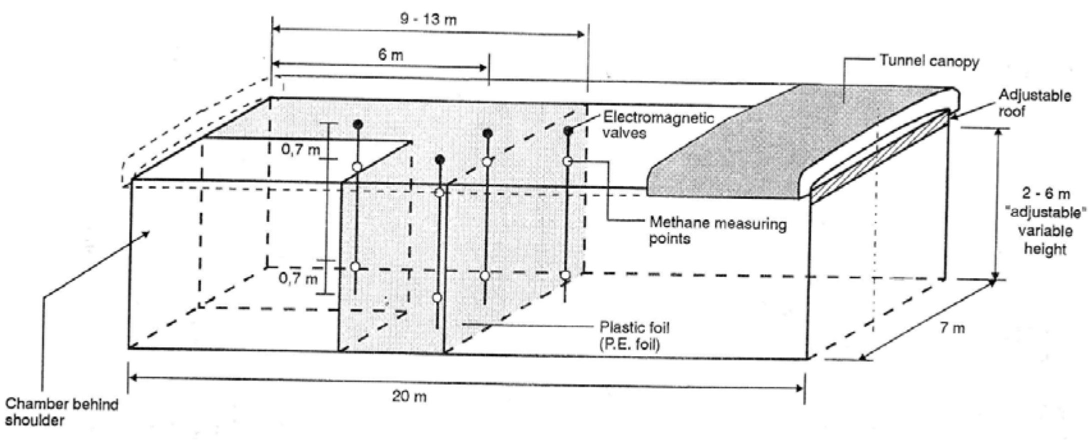

Figure 6.25 Schematic of the Kloppersbos 20m tunnel
(Source: Oberholzer, 1995)

This facility was used by three different companies, all involved with developing active on-board explosion suppression systems. In addition, work undertaken on behalf of the French mining industry involved a simulated road heading machine working in a roadway with an inclined roof (van Dijk, 2000).

Figure 6.26 shows a continuous miner about to enter the tunnel, while Figure 6.27 shows a night-time demonstration of a methane explosion in an empty tunnel.

Figure 6.26 CM about to enter 20m tunnel
(Courtesy of Mr Isaac Mthombeni)

The tunnel was fully instrumented with static and dynamic pressure transducers, optical sensors to detect the extent of flame travel and temperature sensors to determine the continuous miner driver's risk of sustaining burns. The location of the different sensors is shown in Figure 6.28. Before the start of testing the Special Interest Group on Explosion Hazards (SIGEH), which I chaired, asked the CSIR Minintek to produce a protocol for testing active on-board suppression systems (AOSS). The criteria set out by SIGEH were based on the fact that in the 1990s nearly all CM operators sat in a cab on the machine, rather than using remote controls. The criteria were

- flame propogation from a face ignition must be limited by the AOSS to such a distance that the operator is not in danger, i.e. the flame should be extinguished before reaching the operator's position in the cab of the machine
- the shock wave from a methane explosion in the face should be within human tolerance levels at the operator's position
- the temperature rise at the operator's position should not lead to severe burns, i.e. not to exceed 100ºC for more than a few milliseconds

- no false activations of the AOSS should occur as a result of other equipment in use in an underground mine
- the system must be so designed that after operation it must be capable of being made fully operational again with 8 hours.

Figure 6.27 Unsuppressed methane explosion in empty 20m tunnel
(Courtesy of Mr Isaac Mthombeni)

The protocol was revised several times and the final version became available in 2001 (Du Plessis, 2001). In order to obtain the experimental data required to see if an active on-board suppression system achieved the results demanded by the protocol, the tunnel was fully instrumented as illustrated in Figure 6.28.

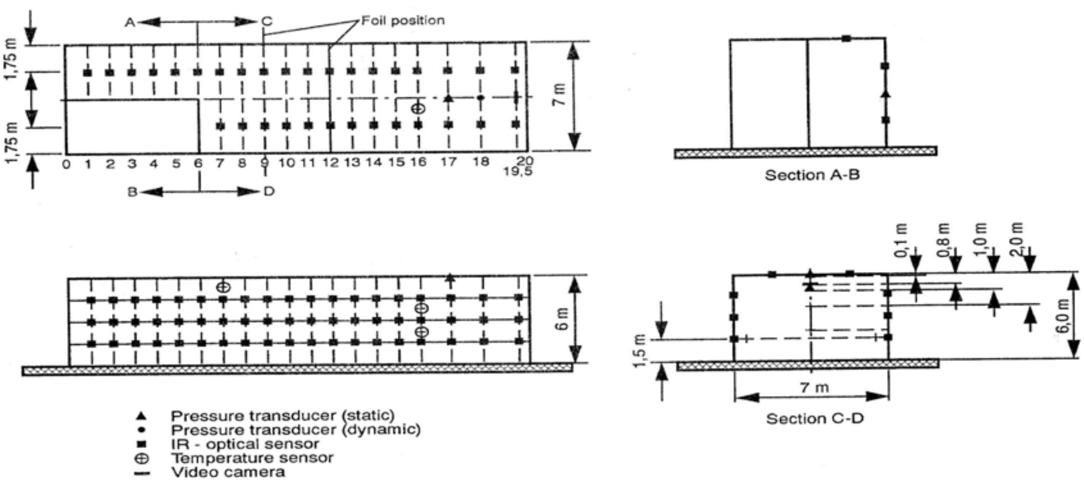

Figure 6.28 Position of sensors in 20m tunnel
(Source: Oberholzer, 1995)

Van Dijk was employed by Centrocem Projects (Pty), one of the companies involved in developing an active on-board suppression system. He registered for 0999999999999999the location of the nozzles of the Explo-Stop system on an actual Joy 14CM9 continuous miner and a model of a Dosco 1300H machine, in each case determining the quantity of suppressant powder required. The Explo-Stop system was originally developed by DMT in Germany as a static roadway suppression system and Centrochem were attempting to re-engineer it in order for it to be machine mounted.

The CSIR undertook 33 explosion tests, with Van Dijk's involvement, using a Joy 14CM9 in various positions in the face (flat face, 1st cut with shoulder, 2nd cut with empty ½ heading, etc), a further 10 tests were undertaken using a full-size model of a CM. This research was funded by SIMRAC. A further 27 tests were undertaken with a model of the Dosco 1300H road heading machine and with the tunnel roof set at a slope of 15º. This contract research was undertaken at the request of INERIS (France) on behalf of the Houilleres de Bassin du Centre at du Midi and replicated the mining conditions in a dipping seam.

With so many inter-connected variables in the system, e.g. density of the suppressant curtain is a function of location of nozzle, mass of powder released and distance of nozzle to furthest point of roadway perimeter, the raw data was not very illuminating. Only when the data was rationalized by grouping data did the picture emerge and conclusions could be drawn. For example, the volume of methane/air mixture divided by (spray to wall distance multiplied by powder mass) is shown plotted against temperature at the operator's position (Figure 6.29).

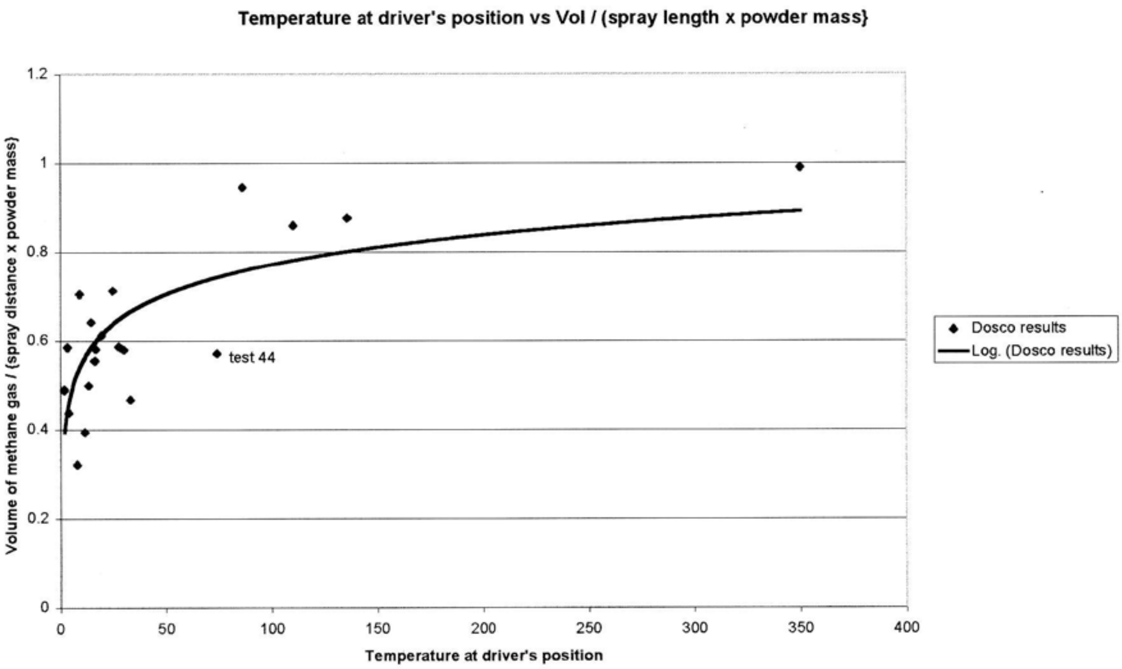

Figure 6.29 Temperature at CM operator's position
(Source: Van Dijk, 2000)

From this figure it can be seen that in order to maintain temperatures below 100ºC the volume of methane/air mixture (Van Dijk often refers to methane gas when in fact it was a 9% mixture) that can be protected against is a function of mass of suppressant powder dispelled and location of the nozzles relative to the periphery of the excavation. "From a purely geometrical analysis one should be able to calculate the volume in front of the powder line as well as the average powder travel lengths. Purely from these calculations it should be possible using Figure 6.30 to calculate the expected temperatures at the operator's cab position" (Van Dijk, 2000)

In summary Van Dijk's major finding was "the volume of the methane/air mixture in front of the powder line has the greatest influence on the way that an AOSS behaves. The limiting volume is around 120 m^3 of a homogeneous 9% mixture. The fact that underground there will never be conditions of a fully homogeneous 9% mixture in front of the mining machine – and the Explo-Stop system- is in itself a safety factor for the system" (Van Dijk, 2000)

A member of staff of the School of Mining Engineering at Wits, Mr (now Professor) Bekir Genc, also used the 20m tunnel tests to prepare a M.Sc. dissertation under my supervision (Genc, 2000). Genc had access to the raw data from the 70 tests conducted between July 1995 and January 1999. The raw data was being interpreted by various researchers in different ways and some considerable confusion existed, mostly due to the very large quantity of data produced by each test. The location of the 76 flame sensors, the dynamic pressure sensor, the static pressure sensor and the temperature sensor have been shown in Figure 6.28. Each sensor reported to a data acquisition system, with the capability of monitoring 128 channels at 30 kHz per channel for a maximum period of 2 seconds.

In gaining an insight into how a methane ignition in the face spreads around the machine and is potentially extinguished by an AOSS, the accurate measurement of flame speed was crucial. The flame sensors were located at 1m intervals and at each interval there were four sensors, two on the roof and one on each side of the tunnel. Although the data acquisition system was able to provide information as to when the flame activated any particular group of sensors, the calculation of flame speed had been undertaken in four different ways. When flame speed from the point of ignition (i.e. the face) was considered then the flame speed could be calculated as the average of the speed measured at each 1m interval or by dividing the total distance by the total time. However, the ignition of the methane air mixture was not taking place "in the face"- it was in a volume contained within a plastic membrane and the interaction between the initiator and the mixture caused abnormalities within this volume. To avoid potential errors, the same two calculations of flame speed could also be conducted for distances measured outwards from the membrane.

In reality all four ways of calculating flame speed were just simple methods of dealing with vast quantities of data and did not fully use the available data. Genc had a strong IT background and used Microsoft Excel, Microsoft Access and the Microsoft SE Cad package to evaluate all of the data and then display the results using Bentley's Microstation CAD software.

The flame arrival times at each of the 19 groups of flame sensors, each spaced 1m further from the closed end of the tunnel, are show in the illustrative example given in Figure 6.30. Genc's contribution to this programme was to show that large quantities of data could be used in their entirety and that simple calculations of parameters such as flame speed could be misleading when trying to determine the success or failure of an AOSSS (Genc, 2000) and (Genc, 2002)

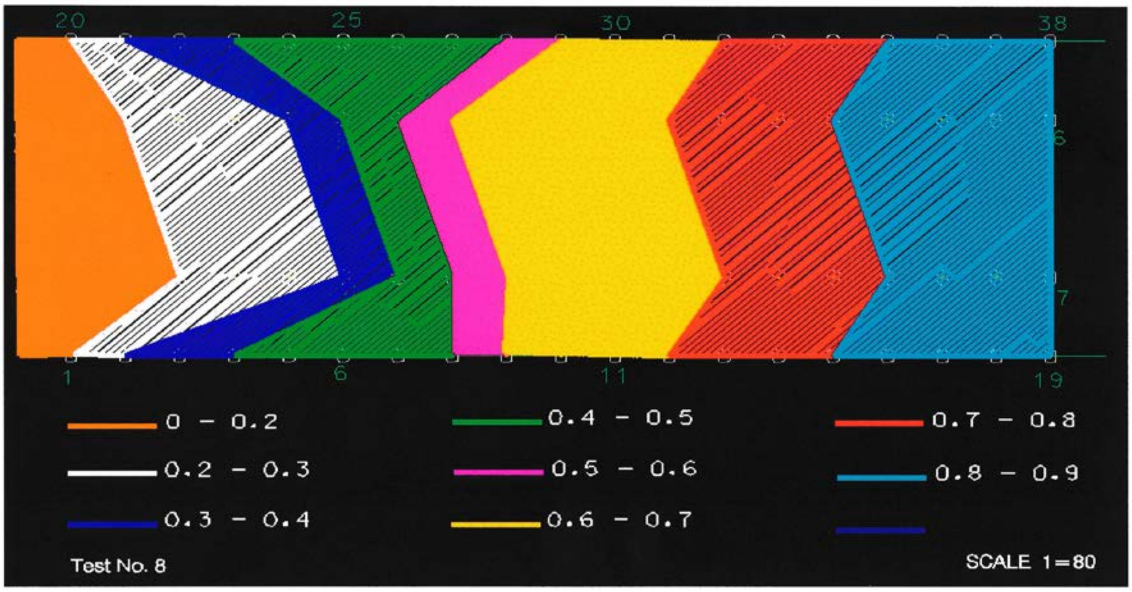

Figure 6.30 Flame propagation inside the tunnel in seconds
(Source: Genc, 2000)

6.5 Collaborative research

Of the 22 doctoral students I have supervised over the years, several produced truly original work of value to the mining industry. In the field of explosions research I was privileged to work closely with two such individuals; Dr Gys Landman and Professor Jan Du Plessis. In addition, on several occasions, I was able to draw in the expertise of well-known international and local researchers such as Dr David Creedy of Wardell Armstrong Consultants in the UK and Dr A.J. Kielbloch of the CSIR

6.5.1 Explosibility at the coalface

This topic was explored by Landman, who had obtained his M.Sc. at Wits in 1988 while working for ISCOR Mining. I supervised his very impressive project report which dealt with thin seam extraction at Durban Navigation Collieries (Landman, 1988). In 1989 he joined the staff of the School of Mining Engineering at Wits as a lecturer. At that time, I was a member of both the Explosion Hazards Advisory Committee and the Research Advisory Committee at Kloppersbos and both committees were concerned with the low utilization of equipment transferred from the Fuel Research Institute. Landman had worked for several years at Durban Navigation Collieries (DNC), which had a long history of explosions, including the

1926 event in which 125 people died. Within ISCOR Mining there was also an acute awareness of the risk of explosions since the sister mine to DNC, Hlobane Colliery, had suffered an explosion as recently as 1983 in which 68 mineworkers had died. With this background Landman was happy to undertake his Ph.D. studies in the field of explosion prevention.

Landman was able to draw on the work of Flint (Flint, 1990) and then conduct a detailed analysis of the explosion history of the South African coal industry. In particular, he estimated the exposure of a South African coalminer to the risk of death in an explosion in the 1980s was 1 in 3000, a figure much higher than that in comparable mining countries. He partially explained this by considering the method of working, which in South Africa was predominately room and pillar.

Landman also started to identify those areas and activities in the local mines which presented the highest risk “It is most important to point out that of the 47 face ignitions between 1980 and 1991, only the Bosjesspruit explosion on 30 March 1984 and the Durban Navigation explosion on 5 September 1991, both in longwalls, were in other than pillar type of mining systems. The difficulties of controlling the mining environment in a heading again exhibits itself in this observation” (Landman, 1992). This led both Landman and myself to concentrate our attention on the conditions at the coalface and to consider whether the increase in ignitions and explosions was directly a consequence of introducing continuous miners. “The mechanised excavation of gassy coal seams inevitably leads to the simultaneous dispersion of methane and coal dust into the environment surrounding the mining machine” (Phillips and Landman, 1992).

Research in South Africa had concentrated on determining the risk of a coal dust explosion propagating. The Fuel Research Institute had acquired and commissioned a German manufactured 40 litre explosion chamber in 1980 and had researched the explosibility of South African coals using the well- known K_{ex} Index. This index had been developed in Germany, where the pressure increase in an enclosed spherical vessel of 40 litres capacity had been compared with results obtained from explosion testing in a 200 m gallery. Two factors combine to make a coal dust explosion propagate. Firstly, the rapid expansion of gases during combustion must create a sufficiently strong “wind” ahead of the flame front to raise a cloud of dust and secondly the dust must itself be sensitive enough that heat radiating from burning particles can ignite the dust cloud quickly enough to sustain the reaction.

When the pressure/time curve for an explosion in a 40 litre vessel is examined these two components can be isolated. Both of these elements are present in the K_{ex} Index. Figure 6.31 shows a typical result for a coal dust explosion in the 40 litre explosion chamber. Following extensive testing in Germany it was found that coals with a K_{ex} Index value of below 7 MPa/s did not propagate a coal dust explosion when tested in a large gallery.

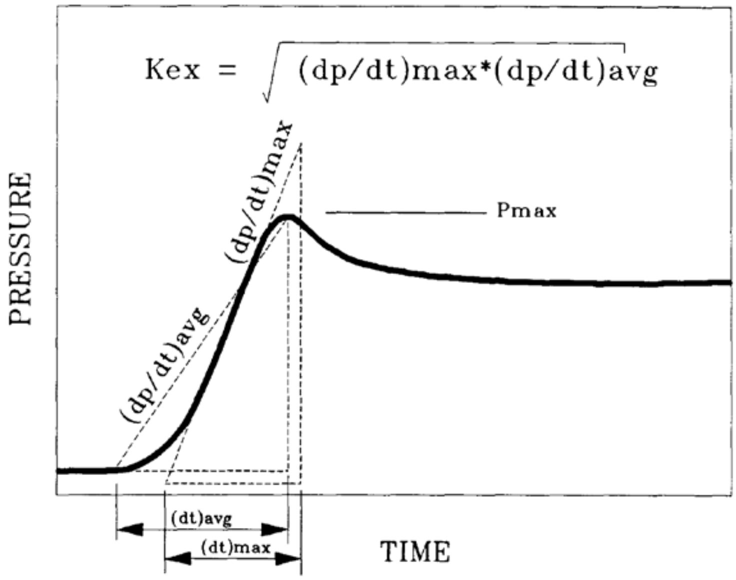

Figure 6.31 Typical pressure/time curve from which K_{ex} is derived (Source: Landman, 1992)

The K_{ex} Index is expressed as

$$K_{ex} = \sqrt{\left(\frac{dp}{dt}\right)_{max} x \left(\frac{dp}{dt}\right)_{avg}}$$

where $\left(\frac{dp}{dt}\right)_{max}$ indicates expansion rate
and $\left(\frac{dp}{dt}\right)_{avg}$ represents swiftness of ignition

Landman's studies were in two, interrelated parts

a) to determine the gas and dust concentrations conditions in and around the cutting drums of continuous miners
b) to study the interactions between methane concentration, dust concentration and ignition energy using the 40 litre explosion vessel

There was already a body of work available regarding methane release rates (Stripp, 1989 and Joubert, 1969). Suffice it to say that in even low gas content seams values up to 4 % in the general body of air have appeared as short duration, severe spikes. On the other hand, what research has been undertaken regarding dust at the face, has been aimed at measuring the respirable fraction and has excluded particle sizes greater than 7 µm. Since it was known that particles up to 1 mm can participate in an explosion Landman used rudimentary apparatus (a pole 4m long with a dust collection pot at the end, a filter with a pore size of 1 µm and pumps to achieve a flow rate of 7 l/min) to sample dust concentrations 0.5 m from the cutting drum of continuous miners. This was repeated 3 times at four different mines. The average concentrations for the four mines were 124, 160, 157 and 135 g/m^3 of particles less than 600 µm in size, with the respirable fraction being 15 to 20 per cent of the mass collected (Phillips and Landman, 1992).

The 40 litre vessel was designed to evaluate the inherent explosibility of coal dust and Landman needed to modify it so that a known quantity of methane could be introduced and to provide the option to replace the original chemical igniters with an energy controllable, electric igniter. Landman illustrated these three-dimensional experimental conditions in Figure 6.32

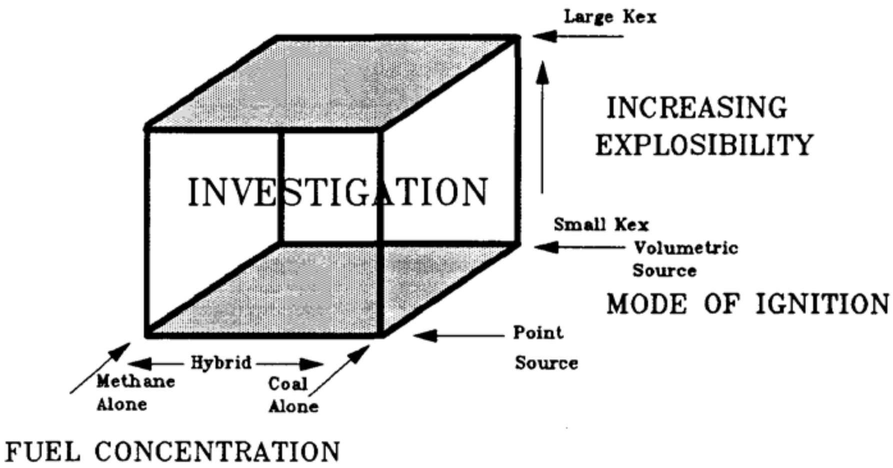

Figure 6.32 Illustration of 40 litre experimental programme
(Source: Landman, 1992)

Landman conducted several hundred tests and was able to define a three-dimensional contour within the above cube, below which the hybrid mixture remains non-explosive at a particular level of ignition energy. By using graphs of % methane concentration against dust concentration for each ignition energy a series of two-dimension illustrations could be made. A typical example is shown in Figure 6.33

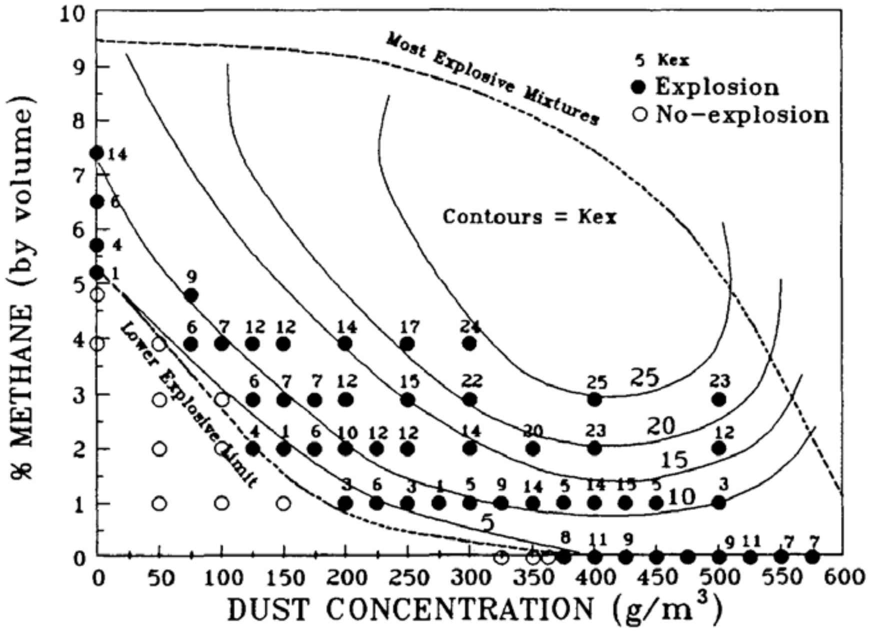

Figure 6.33 The K_{ex} blanket for washed Ermelo coal dust hybrid mixture (Source: Landman, 1992)

The main findings were published at an international conference and created considerable interest (Phillips and Landman, 1992) and it is worth repeating the conclusions here.

"Increased mechanisation and concentration of production at fewer coal faces has undoubtedly led to higher methane and dust concentrations which, in turn, have contributed to an increase in the number of ignitions at working faces, particularly those caused by friction between cutting picks and quartz-rich strata. It has been shown conclusively that the presence of airborne dust, in concentrations currently found at coal faces, can have a significant effect on the ignition of methane/air mixtures. The dangers associated with hybrid mixtures of methane and coal dust have been quantified in three ways: their sensitivity to ignition, their explosiveness and their potential to serve as an initiator for a much bigger coal dust explosion. The influence of dust on the lower limit of ignition of methane is extremely disturbing" (Phillips and Landman, 1992).

These conclusions opened a debate as to why more ignitions did not occur and the role that water sprays play in preventing frictional ignition of hybrid mixtures. This led to two further research contracts (Phillips, 1996b and Phillips, 1997c).

6.5.2 The bagged stone dust barrier

As a member of the Collieries Environmental Engineering Advisory Group (COLEEAG) and subsequently chair of the Special Interest Group on Explosion Hazards (SIGEH), I was aware of the work being undertaken by the CSIR at Kloppersbos. Part of this was in response to a request from industry to devise a replacement for the traditional Polish stone dust barriers. Relocating these barriers in a fast moving room and pillar section was labour intensive and created a steady demand for fresh stone dust. Du Plessis had been busy for several years perfecting the design of stone-dust filled plastic bags capable of being suspended from an integral closure mechanism/plastic hook. Testing in the 2.5m diameter, 200m long explosion gallery had been very promising but the difference in cross-section between the 4.9 m^2 of the gallery and typical underground roadways of between say 18 m^2 to 30 m^2 cast some doubts and questions on how an effective barrier could be constructive. To answer this further test work was conducted at the large cross-section galleries at the Tremonia Experimental Mine in Germany, where horse-shoe shaped cross-sections ranging from 7.5 m^2 to 22 m^2 were available.

“The objective of the work (at Tremonia) was firstly to ensure the correct placement and distribution of the bags in relation to the cross-sectional dimension of the gallery. Part of the work was therefore aimed at optimising bag placement so as to ensure optimum operational effectiveness when subjected to weak dynamic pressure explosions. The final set of tests was aimed at proving the effectiveness of the bagged stone dust barriers, both concentrated and distributed, in inhibiting flame propagation when subjected to a weak coal dust explosion in a large cross-sectional gallery” (Du Plessis, 2000). Despite early problems, the work at Tremonia showed that both distributed and concentrated barriers consisting of bagged stone dust could perform to the same standard as Polish light and heavy barriers. However, the more conservative or perhaps cautious industry representatives at the Safety in Mines Research Advisory Committee (SIMRAC) would not allow bagged barriers to replace traditional barriers, since there were still unresolved doubts regarding their effectiveness in bord-and-pillar sections.

The only facility capable of explosion testing in bord-and-pillar workings was the Lake Lynn Experimental Mine under the control of the Pittsburg Research Centre (PRC) of the National Institute of Occupational Safety and Health (NIOSH) of the United States Department of Health. A plan of the facility is shown in Figure 6.34.

The experimental mine was part of a closed bord-and-pillar limestone mining operation with an almost uniform height of 2m throughout.

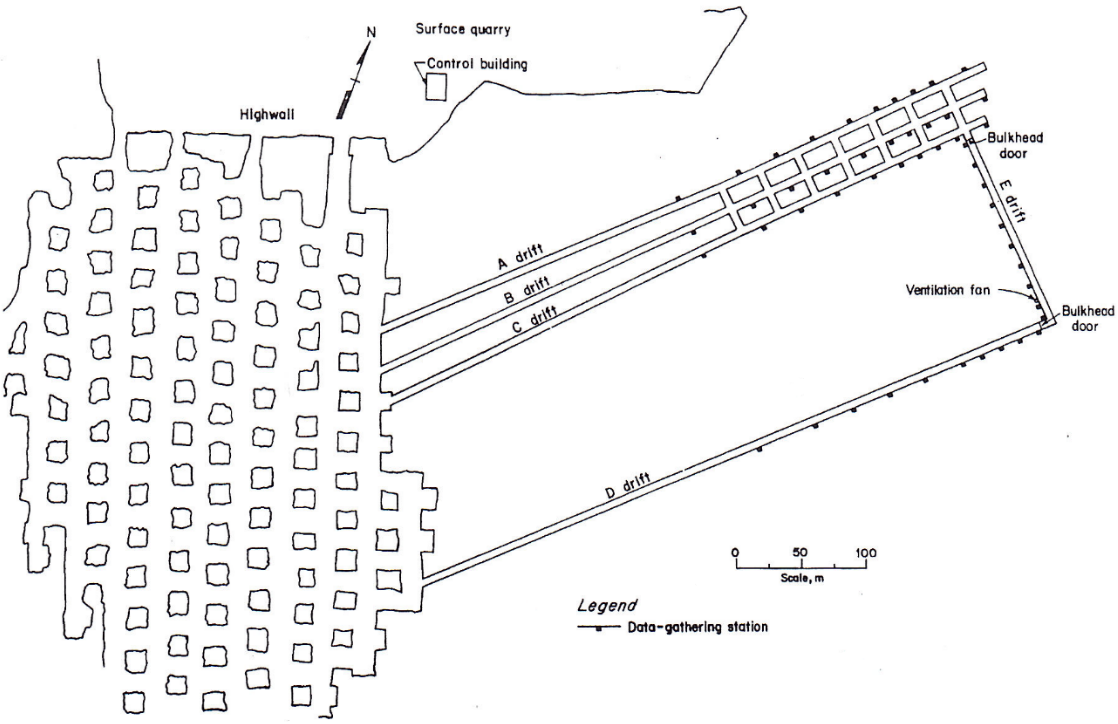

Figure 6.34 A general plan of the Lake Lynn experimental mine

In February 1998, Du Plessis and I travelled to Pittsburg with the aims of :-

a) persuading the PRC of the value to coal mining of undertaking this research
b) defining a suitable programme of experiments
c) determining a work schedule and cost
d) drafting a memorandum of agreement

This was achieved and presented to SIMRAC in the form of a report (Du Plessis and Phillips, 1998). The decision was that this work should go ahead at a cost of US$305 000 and that Du Plessis should be present during the two periods when testing would take place. As an example of a test a concentrated barrier at Lake Lynn is shown below in Figure 6.35.

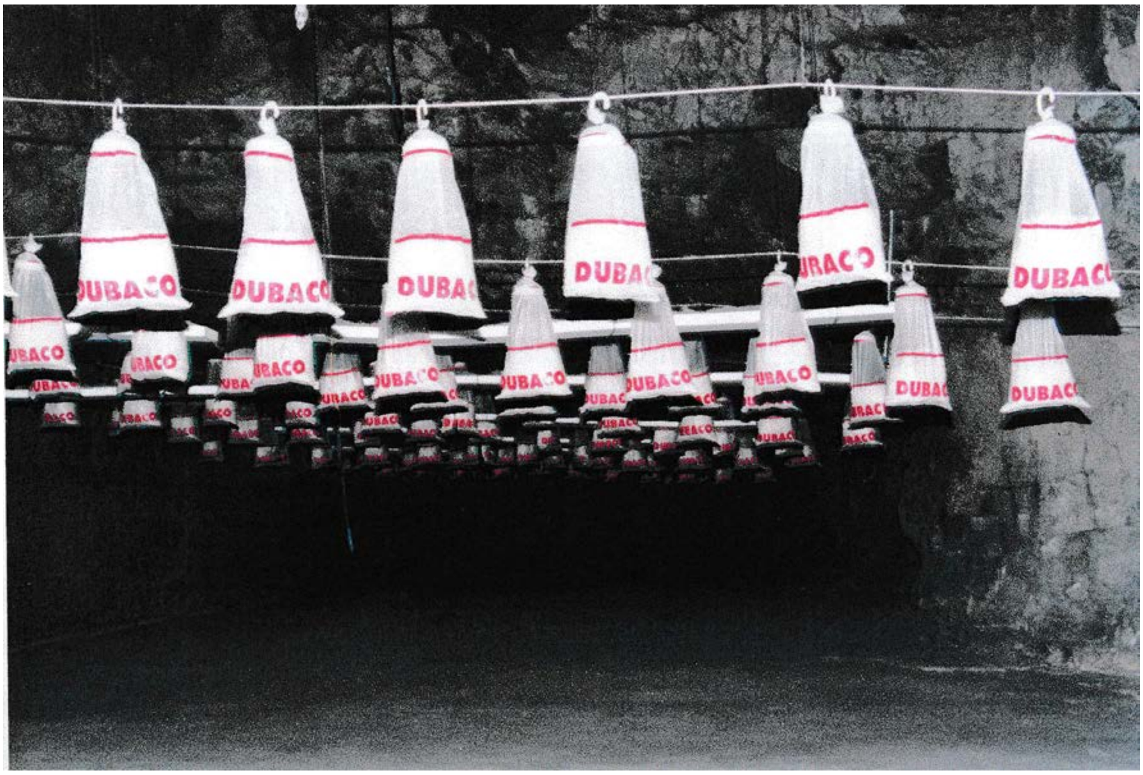

Figure 6.35 Bagged barrier installed for testing at Lake Lynn
(Source: Du Plessis, 2000)

In 1998 Du Plessis registered at the University of the Witwatersrand to undertake Ph.D. studies with myself as his supervisor. The whole development of the bagged barrier is discussed in his thesis (Du Plessis, 2000) and a summary was presented at an international conference (Du Plessis and Phillips, 2000).

The evidence presented (Du Plessis, 2000 and Du Plessis and Phillips, 2000) showed that from work at Kloppersbos the distributed bagged barrier was the barrier best suited for use in bord-and-pillar mining layouts, since it is most likely to operate optimally under conditions of low dynamic pressure. At Tremonia the distributed bag barrier was successfully tested against a fully developed coal dust explosion with the flame being extinguished inside the barrier. The necessary stone dust concentration of 1 kg of stone dust/m^3, which had been determined in tests at Kloppersbos, held true at Tremonia in the 20 m^2 cross-section gallery. Other results obtained by testing this barrier at Tremonia against a full coal dust explosion were

- “flame propagation decrease by at least 100m
- there was a large decrease in temperature
- static pressure decreased to less than half the original pressure
- dynamic pressure decreased to almost a quarter of the original pressure at the fourth sub-barrier position
- the barrier operated within an acceptable time” (Du Plessis and Phillips, 2000)

At Lake Lynn the tests were conducted in the three roadway bord-and-pillar part of the mine and again showed that, despite the attenuation of the pressure wave at each split, both the distributed and concentrated barriers were successful in stopping the flame propagation of a coal dust explosion within the barrier zones.

Bagged stone dust barriers are now used throughout the coal mining industry in South Africa and extensively in Australia. There is now also interest from Poland and the United States.

6.5.3 Post explosion survival

Following the Kinross gold mine fire in 1986 in which 177 workers died, the Government Mining Engineer moved quickly to make the carrying of body worn self-contained, self-rescuers (SCSRs) mandatory in coal mines and those areas of other mines where a risk assessment indicated they should be used. Following their introduction there were a number of fatal incidents in which rescue teams found evidence that individuals had used their SCSR but had been unable to find refuge within the 30 minutes during which these devices supported life.

Together with a junior member of staff I undertook a review of underground communication methods, as well as the available passive and active, low visibility guidance systems. This work formed the basis of a SIMRAC Research Report (Phillips and Gouws, 1995) and was also published in the proceedings of an international conference (Gouws and Phillips, 1995). This work was basically the prelude to a much more detailed research effort involving myself, Dr Kielblock (an authority on the physiology of workers under stressful conditions) and Mr Harry Rose, the Group Environmental Engineer for a major coal mining company.

This collaborative research effort looked holistically at strategies for surviving in low visibility, irrespirable atmospheres following an explosion or a severe fire (Phillips, Kielblock and Rose, 1997). Firstly 10 years of experience with the use of the 125 000 SCSRs introduced into the South African mining industry was reviewed. These sets were predominately of the potassium peroxide (KO_2) type with a nominal duration of 30 minutes at a ventilation rate of 30 litres per minute. However, in low seam heights or inclined escape routes ventilation rates have been measured at in excess of 52 ℓ/min. The results of these tests indicated quite clearly that the time available for escape using a standard SCSR can be considerably less than 30 minutes and that locomotion speed is very significantly reduced in low visibility. When 8 escapees were tested, their average speed of locomotion was 1.44 m/s in normal visibility, falling to 0.54 m/s in zero visibility. Their progress when holding a cable lifeline improved to 1.09 m/s, the best achieved with any of the guidance systems.

6.5.4 Methane layering

One aspect of concern raised by the Middelbult explosion of 1993 was the conclusion that a methane layer in the roof of a heading had acted as a slow-burning fuse (see Section 6.7). While there were very unusual circumstances attached to this, doubts were raised as to whether sufficient was known about the potential for methane layering in the South African coal industry.

In order to understand this issue, SIMRAC awarded a contract to Wardell Armstrong, a consulting firm in the United Kingdom, to undertake this work with the assistance of the Department of Mining Engineering at Wits. The work was undertaken by Dr David Creedy, an international expert in coal mine gases and myself and resulted in a SIMRAC Report (Creedy and Phillips, 1997). The gist of the Executive Summary of our report is reproduced below.

The report reviewed the state of knowledge on the occurrence, investigation, detection, monitoring, prevention and dispersion of methane layers in coal mines. Mining practice throughout the world in respect of methane layering was found to be generally reliant on the results of fundamental work completed in the 1960's in the UK. These studies involved simple tunnels, consistent with longwall situations, and their applicability to bord-and-pillar arrangements is often assumed but had not been tested. It was recommended that experimental work, using scale models validated by selected underground tests, should be undertaken to close the knowledge gap.

The fundamental problem this report identified was that flammable gases could accumulate and remain undetected in extensive areas of bord-and-pillar workings outbye of the face where ventilation could be problematic. Although airflow quantities could appear to be sufficient, the large cross-section of roadways in seam heights of up to 5m could lead to lower than desirable velocities. Due to the relatively high incidence of ignitions at the working face, it is important to prevent any layering in the face area providing a fuse which could transmit a flame to an explosive accumulation of gas elsewhere. A high standard of ventilation along the full length of the last-through-road will prevent such an eventuality. Ventilation monitoring in the last-through-road is therefore important.

The results of this research project, including findings from the literature survey, practical experience, underground observations in South African collieries and industry consultation (via discussions and an industry seminar at Kriel), provided a basis for preparing draft practical guidance on the control of methane layering risks in collieries. The most effective preventative measure against the formation of methane layers is adequate and continuous ventilation. A guidance philosophy was, therefore, proposed which placed the emphasis for minimising layering hazards at working faces on ventilation standards and on gas monitoring in non-working sections, where operational constraints do not allow air quantities to be maintained.

6.5.5 Sealed areas in coal mines

The most common method of dealing with mined out room-and-pillar panels in South African coal mines was to build brick walls across each roadway and ensure an airtight seal by using foam too close off any obvious gaps. A sampling pipe would be built into one of the walls and the atmosphere in the sealed-off area behind the walls sampled at regular intervals, usually about every three months.

In countries with gassy mines the atmosphere in sealed areas usually passes through the explosive zone in a matter of days or weeks. However, in the low gas regime in most South African coal mines the lower explosive limit is never reached.

Once mines began to undertake formal risk assessments in the late 1990s two concerns emerged

1. the sampling tubes typically extended 10 to 20m into the sealed-off panel and there was no information concerning large areas behind the walls
2. should an explosion take place somewhere in the sealed-off area brick walls would not provide any protection to the working parts of the mine

Because of these concerns SIMRAC awarded a research contract to the School of Mining Engineering at Wits and the work was undertaken by Mr D R Hardman, a very experienced mining engineer and Senior Lecturer at Wits, Dr D P Creedy of Wardell Armstrong in the UK and myself. A final report was submitted to SIMRAC in June 2001 (Hardman, Creedy and Phillips, 2001)

The report confirmed that many South African coal mines contain large areas sealed off from the mine ventilation. These areas contain an atmosphere sampled only at or near the seals. The research examined whether the current methods of monitoring sealed off areas were adequate, how South African practice compared with the methods used in other countries and whether alternative monitoring systems existed in 2001.

The research showed that monitoring deep into sealed areas is not normally a major concern world-wide and that periphery or near-seal sampling is normal. The situation in Europe and the United States was examined by Dr Creedy, while I used my contacts in Australia to obtain the necessary information.

The Australian mining industry had developed a risk based approach so that when circumstances indicated the need for multi-point sampling in sealed areas, the accepted method was the use of the tube bundle system, where samples are drawn through small diameter tubes to a central (usually surface) installation and the atmosphere from each tube is sampled sequentially prior to being analysed for the presence of a range of gases. Although the report identifies a number of factors which increase the risk of the occurrence of non-homogeneous atmospheres in a sealed-off area, the one tube bundle installation in South Africa for which data had been analysed indicated a remarkably uniform atmosphere, where all the sampling points gave readings very similar to standard near-seal observations. The alternative to the tube bundle system is to use standard, fixed-point gas sampling apparatus. Because these instruments need a power supply, routine maintenance and regular calibration, their use in a sealed off area could not be recommended. No new or novel devices were identified during this study but two concepts, both using infrared detection of flammable gases, appeared at that time to have potential but would require considerable research and development before they would become viable alternatives to the tube bundle system (Hardman, Creedy and Phillips, 2001).

One member of the SIMCOL committee had visited a mining research facility in Russia where they were advocating the use of methane-oxidising microbes to control the methane level in sealed areas. SIMRAC then placed a contract with Wardell Armstrong to review the existing literature. This identified several research organisations word-wide, including CSIRO in Australia, who were working on the use of microbes. The finding of this study were that biotechnology was being increasingly used in the mining industry and there were many examples of bioprocesses and biotreatments used commercially in mines throughout the world. However, most of the scientific work on methane had been focused on methane production from raw wastes (for energy purposes) rather than methane removal. However, there have been some laboratory experiments and fields trials using methane-oxidising microbes to lower methane concentrations, i.e.

$$CH4 \rightarrow CH3OH \rightarrow HCHO \rightarrow HCOOH \rightarrow CO2$$

Significant quantities of methane can be oxidised by these methanotrophs. Methane oxidation rates are dependent on various factors including the type of methane oxidising microbe, pH, moisture, oxygen availability and methane availability. Laboratory studies had demonstrated methane oxidation rates by pure cultures of methanotrophs from 10 to 31 mmol CH_4/g/h. In comparison, methane oxidation rates measured in natural samples such as soils, peat and sediments range between 0.002 and 10 mmol CH_4/g/h. Specific methane emission rates vary according to type of sample (wetland sediment, sediment, peat, lake soils) and generally vary from 0.1 to 100 mg CH_4/m^3/d. Conditions in South African collieries are favourable for the growth and propagation of methane oxidising microbes and their use in controlling methane in sealed areas. Temperatures lie typically in the range 15 – 24ºC, humidities are from 80 – 95 per cent and methane emission rates are relatively low.

A second contract was let with Wardell Armstrong, together with the professor of microbiology at Wits and myself as partners.

The objectives of this second phase were to:

- “Obtain mine water samples from South African collieries.
- Isolate and grow suitable organisms in the laboratory.
- Design a practical method for implementing the biotechnological solution.
- Clarify operational, safety and health risks associated with the proposed application.
- Specify the requirements of a pilot trial.
- Prepare a Phase Two report and an industry briefing note” (Creedy et al, 2000).

Laboratory work was undertaken at the Department of Molecular and Cell Biology (Microbiology) at the University of the Witwatersrand under the direction of Professor Alexander. Methane-oxidising micro-organisms were isolated from two South African mine water samples and cultures were grown in air containing 2.5%

methane by volume. The presence of methane-oxidising micro-organisms was established but the growth rate was slow. More active organisms were isolated from stagnant pond water. Importantly, the laboratory testing demonstrated that methane-oxidising microbes could “hibernate” in the absence of methane. It is therefore likely that the organisms could survive the lack of methane during the sealing-off process and become active if a build-up of methane occurred.

The report concluded that sufficient technical information and data had been gathered to indicate the feasibility of applying biological methods to reduce methane risks in sealed areas of South African coal mines. The proposal was that a field trial should be undertaken to demonstrate the practicality of the approach and provide the basis for designing an operational system. However, this was rejected by SIMRAC and this work was terminated.

6.5.6 Recent publications

I have continued to collaborate with two of my former Ph.D. students, in both contract research and consultancy. In 2014, I co-authored the chapter on “Explosible Dusts and Mine Explosion Prevention” in the Ventilation and Occupational Environmental Engineering in Mines textbook with Du Plessis, Lebecki and Belle (Du Plessis et al, 2014). More recently I have co-authored papers with both Belle and Du Plessis.

It was felt by both Du Plessis and myself that it was time to again remind the international mining community of the inherent risk of explosions in coal mines. With two of the largest explosions (excluding China) of recent times having been in the United States (Sago in 2006 and Upper Big Branch in 2010) the decision was to prepare a paper for the 16th North America Ventilation Conference (Du Plessis, Saleh and Phillips, 2017). The contents of the paper are given succinctly in its abstract.

“Many coal mine disasters were caused by methane and/or coal dust explosions. This paper lists a number of related coal mine disasters. It further reviews the development of controls to mitigate the risk of explosions. This review emphasises the serious consequences of such events. The controls discussed are classified as preventative measures (against methane and coal dust accumulations) and constructive measures (passive and active explosions barriers). It concluded that active barriers may provide additional benefits over passive barriers and that the successful operation of any barrier still depends on the type and strength of the explosion to be extinguished” (Du Plessis, Saleh and Phillips, 2017).

A second paper involved Belle as a co-author (our 12th joint paper and 15 years after our first publication). This paper (Spaeth, Belle and Phillips, 2017) was predominately the work of Spaeth, the CEO of ExploSpot Systems Pty Ltd, a company which already supplies active on-board suppression systems.

The concept the paper describes is the use of an active, conveyor mounted suppression system to protect conveyor roadways from a developing or propagating coal dust explosion. Each element of the proposed system has been tested on numerous occasions at Kloppersbos and elsewhere, and the technology described in the paper is in common use on-board continuous miners. However, each new application should undergo stringent testing under realistic conditions before being certified for use. Unfortunately, the experimental mine at Tremonia has been closed for some time and, more recently, the Lake Lynn experimental facility also closed. In the absence of a suitable test facility, a demonstration of this system was undertaken at Anglo Thermal Coal's New Denmark mine.

The active explosion roadway barrier (ARWB) is expected to be conveyor mounted and move forward with each conveyor extension. The test was designed to be as realistic as possible with the system mounted on a panel conveyor. An electronic signal activated the system, in place of an explosion triggering it, and two high speed cameras recorded the event. The quantity of the safe, suppressant agent, Mono Ammonium Phosphate, used was calculated using the formula

$$\mathrm{M} = 8\left[\frac{\mathrm{kg}}{\mathrm{m}^2}\right] \text{ x K1 x A}[\mathrm{m}^2]$$

Where M is the total mass of suppressant in the system
A is the cross-section area of the roadway
K1 is a constant depending on the angle of the system to the direction of the roadway.

The value of K1 has been determined in trials conducted by the manufacturer and the values are given in Table 6.8.

Table 6.8 K-factor influencing the amount of suppressant agent to be used in an ExploSpot AESS (Source: Spaeth, Belle and Phillips, 2017)

Distance behind workface	3 - 10m	10 - 30m	30 - 60m	60 - 100m	100 - 120m
Flame speed m/s	50	160	120	225	380
k1 for nozzle bank at 0°	1	1	1	1	1
k1 for nozzle bank at 15°	1.1	1.1	1.2	1.2	1.2
k1 for nozzle bank at 30°	1.2	1.2	1.32	1.32	1.32
k1 for nozzle bank at 45°	1.4	1.4	1.8	2	2.2

At New Denmark the conveyor belt was in-line with the roadway and the system nozzles were set at right-angles to the conveyor, resulting in a K factor of 1. With a roadway width of 5m and a height of 2.5m, the mass of suppressant agent used was 100 kg. Two high speed video cameras (2400 fps) were installed, one on each side of the conveyor belt and these were triggered at the same time the system detected the mock explosion.

The results of the test were reported in Spaeth, Belle and Phillips, 2017 and can be summarised as

a) all but 2.28 kg of the suppressant powder was distributed, i.e. a 98% efficiency
b) from an analysis of the video material it could be seen that the powder barrier filled the entire cross-section of the roadway within 270 milliseconds and in this same time also had a forward motion along the roadway of 5 m.

While this demonstration of the capabilities of the system proved successful, it cannot be compared with the years of test work in developing the Polish stone dust barriers at Experimental Mine Barbara or indeed the years it took to develop the bagged barrier (see Section 6.5.2). Consideration must now be given to how to approve any new system without the necessary facilities to test a barrier against a real explosion

6.6 Personal research

As Chair of the Special Interest Group on Explosion Hazards (SIGEH) I spent time reviewing the recent history of coal mine explosions and identifying root causes where research could possibly help reduce the risk of a recurrence. In addition, in 1993, Landman and I were requested by the Chamber of Mines to prepare the industries position paper, with regards to coal mine explosions, for submission to the Leon Commission of Inquiry (Phillips and Landman, 1993). I also provided testimony at the Inquiry, including cross-examination.

In order to inform the industry of the situation with regards to the explosion hazard and the work of SIGEH I presented three papers at relevant conferences reviewing the situation (Phillips, 1995, Phillips and Brandt, 1995 and Phillips, 1997b). What became abundantly clear was the introduction of continuous miners, starting in the late 1970s had reduced the number of methane ignitions caused by blasting but created the new hazard of frictional ignitions by continuous miner picks. In the period 1990 to 1993 80% of the ignitions recorded by the Department of Minerals and Energy as unknown were in the face area and so it can be assumed that they were also frictional ignitions.

Table 6.9 The changing pattern of ignition sources
(Source: Phillips, 1995)

IGNITION SOURCES	PERIOD							
	1960s		1970s		1980s		1990 to 1993	
	I	%	I	%	I	%	I	%
CM Picks	0	0	1	5.5	14	29.5	6	25.5
CC Picks	1	4.5	3	17	11	22.5	0	0
Shearer Picks	0	0	0	0	1	2	0	0
Stone on Stone	0	0	0	0	4	8	1	4
Blasting	14	64	6	33.5	5	10	1	4
Spon Comb	0	0	0	0	0	0	2	8
Heated Surface	1	4.5	1	5.5	1	2	0	0
Naked Flame	2	9	1	5.5	0	0	0	0
Electricity	2	9	2	11	5	10	0	0
Lightning	2	9	4	22	3	6	0	0
Unknown	0	0	0	0	5	10	14	58.5
Total	22	100	18	100	49	100	24	100

When the findings, as shown in Table 6.9, were considered by the coal mining committee of SIMRAC, i.e. SIMCOL, I was asked to source and review as much information as was available internationally and prepare a report identifying methods to reduce the risk of explosions and fires caused by frictional ignitions.

6.6.1 Review of frictional ignition research

The limitations in time and resources meant that only publications in English or translated into English were used and it was also realised that the topic of frictional ignitions had been extensively researched, so realistically the material sourced represented only a fraction of the material written on the topic. Nevertheless, the report prepared for SIMRAC was based on 155 references (Phillips, 1996b). Hard copies of 100 of the references were obtained and lodged in the library at Kloppesrbos and also scanned electronically. The remainder were consulted on microfiche or in summaries. Contact was made with five research centres and a visit was made to the Health and Safety Executive (U.K.) laboratory at Buxton, where a great deal of frictional ignition research had been conducted.

The report found that while metal on metal or rock on rock ignitions are still hazards in modem coal mining, the vast majority of frictional ignitions are situations where small pockets of methane in air are ignited by cutter picks on a cutting machine. The main results of the very considerable body of research that has been devoted to cutter pick ignitions indicate three very important conclusions:

1) "For a frictional ignition to occur cutter picks must come into contact with strata containing either quartz or iron pyrites. Quartzitic rock must contain at least 30% of fairly coarse quartz grains before the risk of ignition becomes significant. It is, therefore, possible to use petrographic and other analytical techniques to undertake a risk assessment of any site where a machine is to be deployed.

2) The sources of frictional ignitions are not sparks but smears of melted quartz that are left of the rock surface as a result of rubbing friction. The risk of frictional ignition rises rapidly when worn tools, which need higher cutting forces and more energy of cutting, are used. It follows from this that one of the most effective precautions against frictional ignitions caused by shearers, continuous miners and coal cutters is to maintain picks in good condition and to change them before they wear excessively.

3) The most effective means of preventing frictional ignitions during cutting is to spray water directly behind the pick, parallel to its direction of travel. This quenches the hot spot within the lag time for methane ignition, provided such important criteria as quantity, droplet size and to a much lesser extent minimum water velocity are met. Research has produced proven designs of suppression systems to be used in conjunction with all common types of pick. For many years cutting drums on longwall shearers and some road headers have been equipped with phased water supplies, to supply nozzles in front of and behind individual picks. However, wet-heads were not yet in

common use as an option for continuous miners because of technical problems with large-diameter rotary water seals" (Phillips, 1996b).

In addition a number of recommendations were made, including the following :-

1. Joy Mining Machinery in the United States was at that time developing a wet-head cutter case for their machines. While wet-heads for continuous miners were considered in the late 1970s and experimental heads manufactured in the 1980's, potential customers were unwilling to pay the additional cost necessary to provide a wet-head option on a machine. However, in 1995 Joy manufactured a prototype wet-head, which experienced teething problems due to the porosity of the castings involved and also with the seals. The latest news (in January 1996) was that a new housing and seals had been obtained and testing of the system would begin in early February 1997. Since water sprays behind the cutter picks are undoubtedly the most effective suppression technique, **it is recommended that interested parties monitor the progress of this development.**

2. Research has shown it is extremely difficult, if not impossible, to cause a frictional ignition when using new and undamaged cutter picks. If a risk assessment shows the site to be either a medium or high risk environment, the choice of pick design and the replacement criteria should be based on their potential to cause an ignition. While sophisticated instrumentation is available to analyse cutter head vibrations and warn of damaged or worn picks, a more practical solution in the mining environment is the routine and frequent inspection of the cutter head by well trained and motivated workers. **It is recommended that this aspect of work procedures be reviewed by mines and appropriate attention devoted to this aspect of ignition prevention.**

The report on frictional ignition research was also summarised and updated in a paper published in in 1997 (Phillips, 1997a) and was also used by Oberholzer to develop information and training aids to assist all employees to recognise frictional ignition hazards (Oberholzer, 1998). These aids were distributed to all coal mines and are still used by their Training Centres today.

6.6.2 Wet-heads for continuous miners

The recommendation to monitor progress on the development of wet cutter heads for continuous miners was accepted by SIMRAC and the Department of Mining Engineering at Wits was awarded the contract to undertake this work. It involved visiting or contacting the following companies and organisations

- Consol Coal Inc (Pittsburg, USA)
- Deutsche Montan Technologie GmbH (Essen, Germany)
- Eickhoff Bergbautechnik GmbH (Bochum, Germany)
- Hydra Tools International (Pty) Ltd (Rotherham, UK)

- Joy Manufacturing (Franklin, Pa., USA)
- Powercoal Pty Ltd (Sydney, Australia)
- Pittsburg Research Centre, NIOSH (Pittsburg, USA)

From these contacts it was discovered that “wet-head continuous miners i.e. with water available on the rotating cutter head, were first manufactured in the 1970's and tested as part of a research programme to reduce respirable dust levels. The work was sponsored by the United States Bureau of Mines and largely undertaken by that agency. Joy, Jeffrey and Lee Norse continuous miners, Jeffrey and Wilcox auger continuous miners and longwall shearers were all subjected to underground trials in the period 1974-76. Although it was recognised that water supplied directly behind the picks could significantly reduce the risk of frictional ignitions, the main thrust of the programme was concerned with dust reduction” (Phillips, 1997c).

The report to SIMRAC had the following conclusions

1. Research and the development of frictional ignition prevention devices for longwall shearers had been undertaken for at least 30 years. A range of suitable devices for ventilating the cutting drum and providing water for both dust suppression and pick back flushing was available and shearer manufacturers could offer machines and cutting drums specifically designed to minimise the risk of frictional ignitions. More recently work (1997) in Germany had produced wet-head systems which, either alone or in combination with compressed air, can give an equal level of protection for roadheading machines. Current developments in this area involved further refining of these systems and the remote (surface) monitoring that the devices were, in fact, operating.

2. The parallel development of wet-heads for continuous miners was delayed by about two decades because of the need to use large diameter rotary seals to transfer water to the cutting drums. Although seals were available they were found in practice, to be incapable of withstanding the effects of machine vibration and poor water quality. This problem had been solved although the seals, at a cost of up to $25 000 (in 1997 money) per pair, were expensive. Three machine manufacturers (Joy, Eimco and Voest-Alpine) at that time offered wet-head continuous miners, while Hydra Tools International provided the option of a retrofit wet-head for any continuous miner. Although operational experience with all four systems had been limited to a few machines and only a few years, confidence that reliable wet-head continuous miners could be made available was growing

3. Interest in the use of wet-head machines had been detected from Australia, the United States, the United Kingdom and Russia, as well as from the South African industry. Once machines are equipped with wet-heads, water can be used both for ignition suppression or, equally importantly, for the reduction of respirable dust levels. It appeared that no

further research was required into the use of wet-heads for frictional ignition prevention. The basics had all been thoroughly researched and progress in the design and improved reliability of wet-heads for continuous miners would only be achieved if more machines were purchased and used.

Following the acceptance of this report by SIMRAC several mines purchased wet-heads for retro-fitting onto existing continuous miners. SASOL Coal used these heads for several years with limited success. The greatest problem was the quality of service water. In-line filters, if too fine, became quickly blocked and, if more course filtration was used, dirt particles passing through the rotary seals caused premature failure. Several ignitions occurred while wet-heads were being used but it has been impossible to ascertain whether the pick box sprays were all working at the time, since it is understood they frequently became blocked. The latest information is that wet-heads are very rare in the South African mining industry.

6.7 Investigations of multi-fatality explosions

Between 1987 and 2013 I was involved, in different capacities, with five multi-fatality explosions, involving a total of 186 fatalities. My contribution to the five investigations is listed in Table 6.10.

Table 6.10 Involvement with multi-fatality explosions

Mine	Date	Fatalities	Involvement
Ermelo Mine Services	09/04/1987	35	Advisor to the mine's legal team during criminal trial
St Helena Gold Mine	31/08/1987	63	In loco inspection and expert witness in trial of Shaft Manager and Env. Engineer
Ermelo Mine Services	7/11/1992	6	In loco inspection and advisor at the inquest
Middelbult Colliery	13/05/1993	53	Chair of internal inquiry and advisor at inquest
Pike River (N Z)	19/11/2010	29	Advisor to management's legal team throughout Royal Commission and pre-trial hearing

In a keynote address to the Australian Mine Ventilation Conference in 2015 I used the last four of the incidents listed above as case studies to draw out the lessons learnt, since this, unfortunately, was also a form of practical research (Phillips, 2015). These case studies and the conclusions to the paper are reproduced here with only minor changes.

6.7.1 St Helena Gold Mine #10 Shaft – 1987

Although not a coal mine, St Helena was classified as a fiery mine. Ten shaft at St Helena gold mine was sunk during the period 1983 to 1987 and was still not fully commissioned at the time of the accident. The shaft was 9m in diameter, reached a depth of nearly 2000m and had a brattice wall, with 67% of the cross-sectional area for downcast air and 33% for upcast air. An intermediate pump station (IPS) had been excavated while shaft sinking and this was at a depth of 695m below collar. Two coal seams had been intersected at 330 and 360 metres below collar. Methane had been encountered at a depth of 462 metres below collar but was sealed by grouting. Figure 6.36 is a plan view of the IPS. The excavations were large, with the height throughout being about 7m and the width varying between 6 and 8m. The reason for these large dimensions was the designed capacity of the water storage dams. Work on equipping the IPS began about 4 months prior to the explosion and, while the major construction work, such as building the dam walls, was complete, cutting and welding of pipework was still in progress on a daily basis resulting in checks for methane on a regular basis.

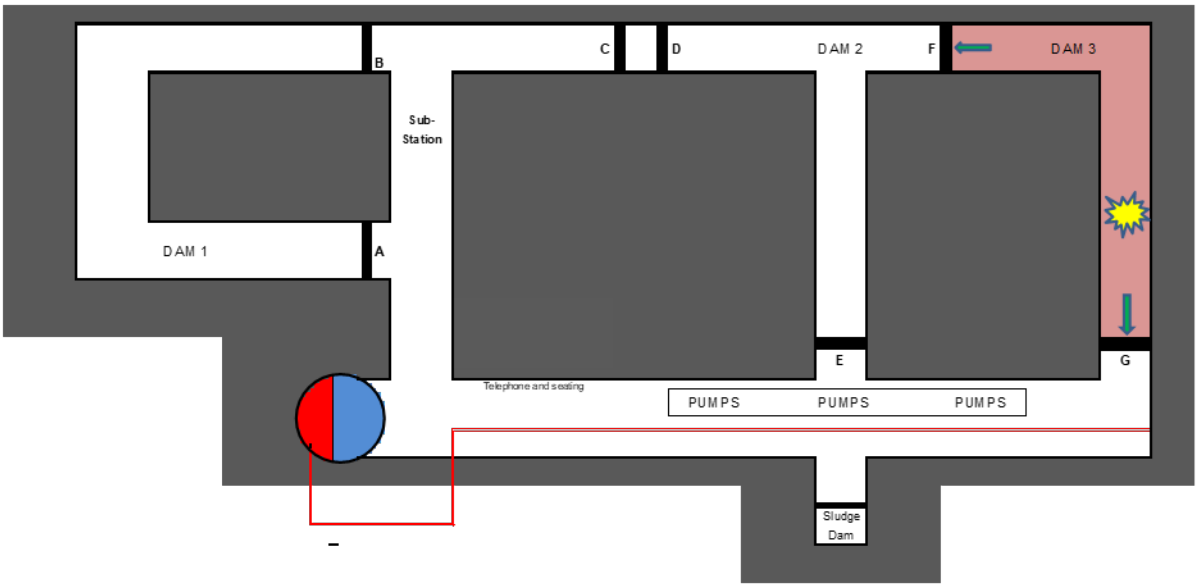

Figure 6.36 Plan view of IPS, #10 shaft St Helena gold mine

The IPS was ventilated by a 406mm pipe and used the negative pressure of the upcast shaft to draw about 8 m^3/s of air from the downcast side of the shaft and through the IPS. At the time of the explosion the intake side of the shaft was carrying 150 m^3/s of fresh air, while the upcast side was carrying 300 m^3/s of return air (150 m^3/s from # 10 shaft and 150 m^3/s coming from underground connections to other shafts). At 5:15 am on Monday 31st August 1987, during man riding in the shaft, a group of 9 artisan's assistants was conveyed to the IPS to await the arrival of the artisans on a later cage. Approximately 30 minutes later a full cage, travelling below the IPS was lost, the two # 10 shaft fans tripped and dust and gases permeated the working levels. The Mary Ann cage was slowly lowered but the shaft foreman reported that there was extensive damage to the shaft steelwork at a point 30 m above the IPS. When he returned to surface he also confirmed that the brattice wall had disappeared and he had heard screaming from the IPS. A mine overseer, who was also a member of a Proto team volunteered to be lowered in a hastily constructed bucket suspended, without guides, 40m below the Mary Ann. Despite the uncontrolled spinning of the bucket he was repeatedly lowered to the IPS until all 9 burnt personnel were rescued. One, who also had crush injuries from being trapped under a falling wall, unfortunately died in hospital.

It took a shaft sinking company 3 months to make the shaft safe down to the level where the delicate task of removing the accumulated debris at the shaft bottom could begin. After many months the bodies of all 62 persons in the cage were recovered. However, two months after the explosion it was possible to safely access the IPS by kibble and to begin the investigation. The thick dam wall at position G was blown over. Since this wall was approximately 6m wide, 6m high and the reinforcing bars were connected to drill steels anchored in the sidewalls, it was obvious considerable force had pushed it from the empty dam in an outwards direction. Similarly, the wall at F had cracked and bulged outwards. The paint had blistered off all the steelwork, which in the intervening two months had become very rusty. There was also a layer of soot (very fine carbon) over some surfaces and the sheet metal of the electrical control panels and switchgear had bulged outward or been “blown-off”. Initially the inspectorate investigators put forward a theory that the operation of a switch had triggered the explosion. However, the overwhelming evidence was that the explosion occurred in the dam between walls F and G and that the switchgear survived the initial over-pressure because of its construction and internal cross-bracing, while the vacuum following the explosion had sucked the sheet metal outwards. During the investigation several cigarette butts were found despite it being a fiery mine in which smoking was prohibited. The most likely cause of the explosion is that one or more of the artisan’s assistants smoked while unsupervised at the IPS. To dispose of the butt ends one of them climbed the ladder built into the dam wall and flicked the butt over the top while still alight, causing the ignition of gas. Because the walls blocked air movement inside each of the dams and because of the difficulty of inspecting behind the walls, methane had built up undetected inside the dams. Following the experimental work of a postgraduate student (see Section 6.4.1). it was determined that at the depth of the overlying coal seams the hydrostatic pressure would have allowed approximately 1m^3 of methane to dissolve in each tonne of water (Searra, 1990). When this water seeped through fissures lower down in the mine, methane “boiled” from the water at atmospheric pressure. This was why this gold mine was classified as fiery.

6.7.2 Ermelo Mine Services (Pty) Limited – 1992

On Saturday 7th November 1992, an explosion occurred in a drill and blast, room and pillar section of Ermelo Mine Services coal mine. While the loss of life was small in comparison the other explosions listed in in Table 6.10, there are some important lessons that emerged from the investigation.

The affected panel, shown in Figure 6.37, was at right angles and immediately adjacent to a previously worked panel, part of which had been sealed with brick walls at A1-A6 and the sealed area was known to contain significant quantities of methane. Before production commenced the seals were examined and the highest concentration of methane detected in Roadway A was 0.7%. The original plan was to split the air to have return airways on both sides of the panel. On the right hand side this would be Roadway A. However, because part of the original barrier pillar obstructed Roadway A, the panel was ventilated as a temporary measure by coursing the air as depicted in Figure 6.38. Production started on October, 28th and continued for 9 working days before the explosion. Simultaneously with the start of production, a small team was assigned to the task of drilling and blasting an airway through the offending pillar.

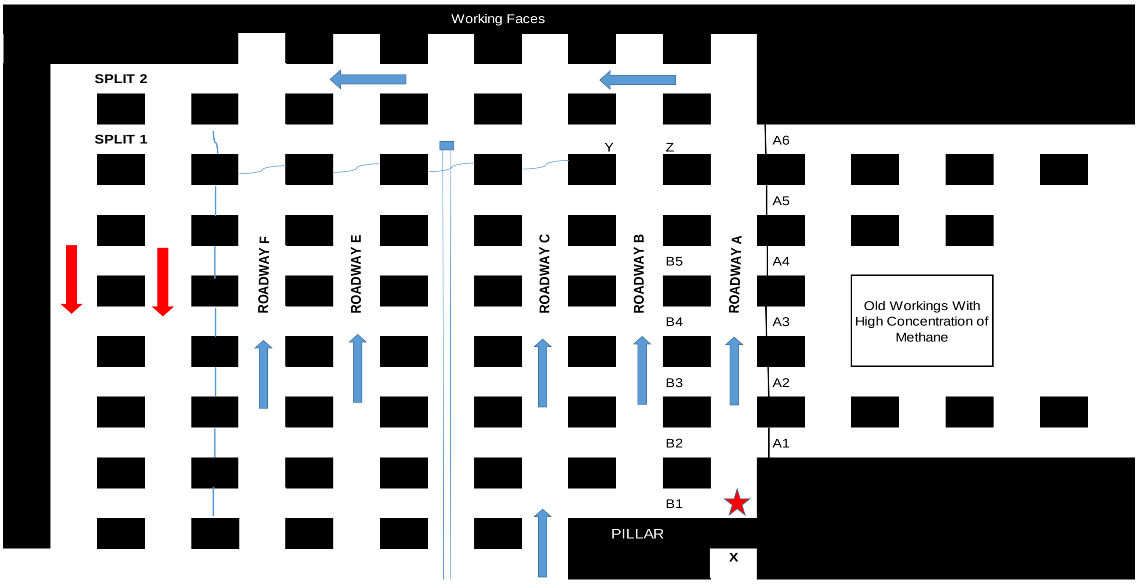

Figure 6.37 Simplified layout of Section 5, Ermelo Mines

By Saturday 7th November this holing was nearly complete and two labourers were assigned to drill the face in preparation for blasting. They were not authorised to charge the holes with explosives unless the miner was present but they did so, even wiring up the face and running the blasting cable to a place of safety. The miner arrived back from the face just before 8am and attached his exploder and blasted. There followed a far larger explosion than expected and in the face of large volumes of smoke the three men retreated in intake air and informed the control room. Within two hours a fresh air base had been established and the mine's Proto Team (personnel trained in mine rescue procedures) entered the section. A number of workers at the production faces had evacuated the immediate area and were receiving attention for varying degrees of burns and the Proto Team found and recovered 4 bodies and reported 3 fires burning. At this stage two workers were still missing and a second Proto Team entered the section, retrieved a further two bodies and extinguished the fires.

When the investigation began on Monday 9th, it was found that the shot holes had gone completely through the remaining portion of the pillar (at x in Figure 6.37).

The detonator wires were burnt but still visible on the Roadway A side of the pillar remnant. Walls A1-A6 had disappeared and the conveyor structure in Roadway D was covered with the remnants of bricks. The old workings on the right of the panel were now sealed off with brattices and work was due to start on building walls to replace the brattices. Diurnal pressure changes were causing variations in methane content in Roadway A and the panel was still very humid. Interviews with all three personnel involved in blasting the pillar indicated that all three thought the explosion happened at the instant the explosives were detonated but the two assistants could not remember the miner testing for gas, while the miner insisted he had tested on both sides of the pillar since he was expecting to hole through with this blast.

On a second visit to the section by the investigation team, a sharp-eyed mines inspector noticed a strip of mortar around the periphery of the splits at positions B1 to B5. This was difficult to see because of the soot and the wet conditions caused by the condensed water vapour from the explosion. Nevertheless, these traces were proof that walls had existed and it was the smashed remnants of these walls which came to rest on the conveyor structure. A reconnaissance by the mine Proto Team into the old workings confirmed that the walls at A1 to A6 had been blown about 4 pillar lengths into the old workings. In the planning for the section move the Environmental Department wrote a Memo to the Mine Overseer, the contents of which are reproduced below. A marked plan was also attached.

1) Air crossings to be built at position 1
2) A standard mandoor must be installed at position 2
3) Walls to be built at positions shown on plan
4) Remove walls as marked X
5) Holings to be effected at position 5

The outbye construction foreman was interviewed but denied there were walls at positions B1 to B5. However, after a few hours and with great emotion he told the investigation team that he had finished the appropriate construction work on the air crossings ahead of schedule and so moved on to his next task. After the explosion he realised that by building walls at B1 to B5 he had constructed a “shoe box”, one roadway wide and dipping from the pillar towards the working faces. Although examination of all shift reports showed expected ventilation conditions, the previous afternoon shift reported higher than normal methane levels at the right hand side of the working faces. We can surmise that the blasting activities so close to the walls A1 to A6 created slight damage leading to methane leakage. The miner in charge placed a brattice between points Y and Z. This cleared the faces but almost completed the closure of the “shoe box”.

It was fairly obvious the miner who blasted the explosives at position X had not taken the long walk around walls B1 to B5 and so had not tested for gas in the “shoe box”. Further investigation revealed that it was custom at the mine to leave some gas detection instruments (GDIs) in the lamp room on Saturdays for their weekly inspection and calibration. Lamp room records proved conclusively that this particular miner did not have his GDI with him and made no tests for methane before blasting.

6.7.3 Middelbult Coal Mine – 1993

On the 13th May 1993 Section 38 at Middelbult coal mine experienced a massive explosion. All 27 workers in the section died from trauma and/or burns, while 26 workers in the section on the other side of the primary development died from carbon monoxide poisoning or asphyxiation. The layout of the two panels is shown in Figure 6.38.

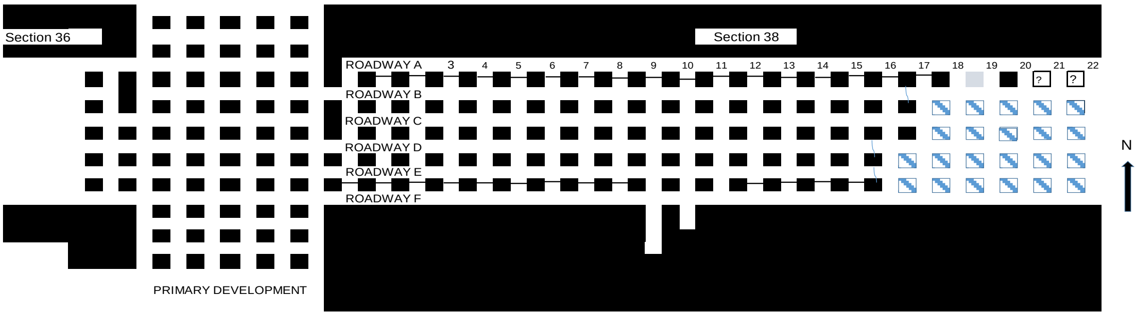

Figure 6.38 Simplified plan of Sections 36 and 38, Middelbult coal mine

Section 38 was a long panel that had been worked as a room and pillar operation until it reached the mine boundary (approximately 3 km from the primary development). Thereafter, the section retreated extracting the pillars and keeping the goaf line at an angle of 45º, as advised by the rock engineering department. Pillar extraction was scheduled to stop at split 16, since at that point the workings began to undermine a large slimes dam on surface. Approximately 83 m^3/s of air entered Section 38 along Roadways B, C, D and E and at the working faces was directed by brattices to ensure the fringes of the goaf remained free of methane. The return air split naturally in the last through road into single returns in Roadways A and F. No regulators were used to balance airflows. Furthermore, an attempt was made to have a bleeder road around the goaf and to that end pillars and walls adjacent to Roadway A should have been left intact. However, when the explosion happened one of the two Continuous Miners (CMs) was cutting a split in the pillar adjacent to Roadway A and between splits 18 and 19. This CM did not cause the explosion but provided evidence that the ventilation, as planned, was not carried out in practice.

The pillar extraction was conducted using two CMs and by the 13th May the machine on the right- hand side of the face had reached the prescribed limit of mining. A new panel was planned to the south of Section 38 and some new equipment had been delivered but planning was still in progress. Personnel in the section had raised concerns about a loss of production bonus and this may have contributed to the series of events that occurred. On the morning shift a miner first broke down the walls at splits 11, 10 and 9 between Roadways E and F. He installed a regulator in Roadway F between splits 6 and 7 and proceeded to mine in a southerly direction in splits 9 and 10. In split 10 the excavation was approximately 28m long and in split 9 the new excavation was approximately 47m long, with neither of these new excavations having any installed support or signs of having being ventilated. This method of mining contravened all mine standards and the ventilation change caused by removing these three walls was not sanctioned by the ventilation department.

During the investigation it was discovered that the controls of the CM were in the "on position", as were those of the shuttle car behind it. The bodies of the CM driver, shuttle car driver and the cable handler were found in Roadway F, indicating that some event occurred warning them of the impending explosion. The cutting head was at 75% of the seam height and at a band of iron pyrites in the seam. In addition, there were pick scratch marks in the sandstone roof all along this heading. Section 38 was examined meticulously by two teams, one from the Government Mining Engineer's Office and the one of which I was a member represented the mine's owner. All indications of equipment movement from the last known position were plotted and the violence of the explosion was illustrated when tyre tracks showed a loaded shuttle car had been pushed sideways more than 5m. The force diagram indicated that a major explosion of methane had occurred in the goaf and microscopic analysis of dust particles in the section indicated that coal dust involvement was minimal. Force directions were also clearly indicated by the destruction of ventilation walls. All the walls between Roadway B and Roadway A were blown violently into Roadway A, indicating that the major force passed down the centre of the panel. However, a different picture emerged in Roadway F. In splits 9, 10 and 11 there were obvious signs that the walls had been deliberately dismantled. In splits 12 to 14 and again in splits 4 to 8 the wall remnants were found in Roadway E, i.e. the force destroying them did not come from the centre of the panel but occurred in Roadway F.

It was possible also to draw a contour of the extent of flame, or at least heat, in the section since it was the mine's practice to hang a piece of chevron tape from roof bolts in every intersection. This tape was tested and melted at between 100º and 115º C. For the most part the chevron tapes were burnt throughout the section except in splits 1 to 4 where some were burnt and some not. Since it was impossible to enter the goaf to map the total perimeter of the area affected by heat, the burnt/not burnt chevron tape did not help in quantifying the volume of methane-air mixture involved. However, there was a good correlation between the accessible area of burnt tape and the area where characteristic "rain spots" on the floor indicated that the water vapour in the explosion products had condensed. Many surfaces were covered with soot while vertical surfaces facing away from the goaf were cleaner, again indicating the direction of the primary force.

Very simple computer simulations were run to indicate the likely ventilation pattern in the section and two months after the explosion a series of physical simulations of different ventilation scenarios were conducted using brattices to replace the walls. When the most likely situation, i.e. with walls in splits 9, 10 and 11 removed and the regulator between splits 6 and 7 in place, was simulated, the 82 m^3/s of intake air split into 53 m^3/s of return air in Roadway A and 29 m^3/s of return air measured at the regulator in Roadway F. Very interestingly 6.7 m^3/s of intake air passed through splits 9, 10 and 11 and travelled through the goaf before exiting the panel through Roadway A.

From all the evidence collected it was deduced that a frictional ignition occurred at the CM in the 47m excavation and that there was a methane layer in the roof which burnt. This slow burning layer gave the crew time to run to Roadway F leaving their machines running. This “fuse” of methane reached its explosive range in Roadway F and this blasted the remaining walls in splits 4 to 14 towards the centre of the panel. The flame then continued inbye into the goaf and somewhere in the goaf ignited a massive methane explosion. Research, using surface boreholes into a similar goaf at a nearby mine (see Section 6.4.2 and Cook, 1999), showed that gas movement in the goaf of a pillar extraction panel is by diffusion and is not significantly affected by panel ventilation. In the Witbank/Highveld coalfields of South Africa the methane content of seams is low and the build-up of methane in the goaf is slow. This eventually results in explosive pockets of methane-air mixtures in the goaf, which may contain hundreds, or indeed thousands, of cubic metres of mixture within the limits of explosibility.

6.7.4 Pike River– 2010

On the 19th November 2010 an explosion at the Pike River coal mine in New Zealand killed all 29 personnel underground at the time. Following the explosion a Royal Commission was appointed by the Governor-General of New Zealand to inquire into the circumstances that caused the explosion and to make recommendations to prevent a recurrence. The report of the Royal Commission (Pankhurst, Bell and Henry, 2012) was published in October 2012. The Commission sat through 51 days of hearing, 57 witnesses appeared in person and more than 200 written submissions were made. In fact, the transcript of the hearings runs to 5682 pages. The report of the Commission is massively impressive with volume 1 being a summary/overview and volume 2, which is in two parts being concerned firstly with what happened at the mine and then making proposals for reform, including recommendations concerning the regulators, the mining industry and its workforce and emergency management within New Zealand.

The Commission identified a myriad of failings at Pike River ranging from failure to adequately complete vitally important risk assessments, to failing to adequately train raw recruits to the mining industry, to allowing non-flameproof diesel equipment to operate within the restricted zone of the mine.

However, the first term of reference of the Royal Commission asked that it should inquire into the cause of the explosion. The Department of Labour also investigated the explosion and appointed an Expert Panel to assist with this task. Members of the Expert Panel gave evidence to the Royal Commission and, to clearly understand their conclusions, it is necessary to quote a few paragraphs of the Commission Report (Pankhurst, Bell and Henry, 2012) and also to remember that the mine has not been re-entered since the first explosion. The basic layout of the mine at the time of the explosion is shown in Figure 6.39.

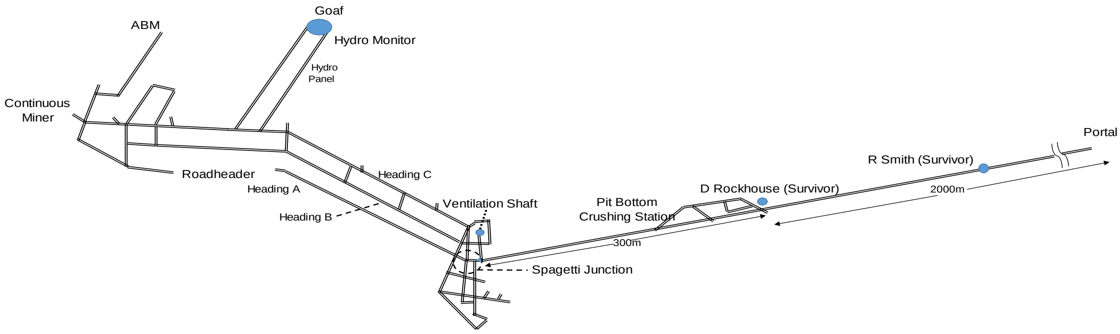

Figure 6.39 Plan view of Pike River coal mine

"25 To determine the source of the methane consumed in the explosion, the panel first assessed the likely volume of methane required to produce an explosion of the kind recorded on the portal closed circuit television (CCTV) footage. This footage was the starting point from which to work back and endeavour to extrapolate the initial methane volume.

26. The blast exited the portal for approximately 52 seconds, with most energy expended in the first 30 seconds. The cross-sectional area of the stone drift at the portal was approximately $22m^2$. To estimate the velocity of the blast, the speed at which debris passed through the 7.5m field of vision of the camera was calculated. The camera recorded four frames per second and debris cleared the field of vision in less than one frame. This indicated a blast velocity greater than 30m/s (metres per second) and within a range up to 70m/s

27. The expert panel's analysis of the blast enabled the volume of gas ejected at the portal to be estimated. Then followed the extrapolation process:

- Explosion products ejected at the portal 30–70,000m^3
- Double this for a similar volume of products ejected at the vent shaft 60–140,000m^3
- Divide by 5 (the assumed expansion factor of the explosion) to establish the volume of mine atmosphere which exploded 12–28,000m^3
- Reduce explosive mine atmosphere volume to 5% (the lower limit of the explosive range of methane) to establish the minimum volume of methane consumed in the explosion 600–1400m^3.

28. However, the expert panel concluded that the methane consumed in the explosion was more likely to be at least 1000m^3, and possibly a much higher amount. This reflected the high concentration of carbon monoxide in the post explosion gases, as confirmed by the two survivors' prolonged loss of consciousness and the analysis of early samples from the vent shaft. High

post-explosion concentrations of carbon monoxide indicate a fuel rich explosion."

In March 2012 a further communication from a member of the Expert Panel informed the Royal Commission that he favoured 2000m^3 pure methane (not the volume of methane-air mixture) as the upper end of the range of fuel quantities. This led to the conclusion that the almost certain source of the methane was the only goaf in the mine, i.e. in the hydro-mined panel. The Royal Commission then spent a great deal of time and energy investigating hydro-mining, how the methane could be expelled from the goaf and potential sites where it could have been ignited.

The flaw in the analysis by the Expert Panel lies at the very beginning of their extrapolation process (para 27 above). From the record of the CCTV at the portal they calculated the volume expelled,

i.e. cross-sectional area of drift x velocity x time = volume

They then regarded this as half the post-explosion explosion volume of water vapour, carbon dioxide and carbon monoxide and possibly a little oxygen. The exact composition is dependent on the composition of the methane-air mixture prior to the explosion. To consider that the post explosion mixture of water vapour and gases, which would be at some temperature between 1200 ºC and 2000 ºC divided into two "bubbles", one travelling out of the upcast shaft and the other exiting the portal without changing its temperature, and thereby its volume, is quite frankly ludicrous. To even suggest that the volume of air and other gases in the mine increased when the explosion occurred and so the increased volume was expelled is equally wrong. The products of the explosion, having expanded to between about 2 and 7 times the initial methane-air mixture, cools rapidly because it is in contact with the roof, floor, coal pillars and equipment all of which are at the ambient temperature of the mine, perhaps around 25 ºC, thereby creating a partial vacuum. This vacuum was due to the change in volume of the gases as they cooled and due to the massive change in volume due to the condensing of the water vapour. What then caused the CCTV to record an event?

The shock wave caused by the rapid rise in pressure during an explosion travels in all directions. In directions leading to a dead end it becomes a reflected wave but often it travels long distances until it exits the mine, causing destruction to ventilation control devices along the way. When the pressure wave reached the drift at "Spaghetti Junction" it was faced with a single path about 2.3 km long, i.e. similar to a gun barrel. It then compressed the air ahead of itself pushing it out of the mouth of the drift. The energy consumed by this process of compressing a long column of air slowed the pressure wave leading to a high initial velocity of ejection, which slowed down as the pressure wave got closer to the portal c.f. para 26 of the quote above.

What then was the average velocity of the pressure wave while in the drift?

Using the simple equation

$$velocity = \frac{distance}{time} = \frac{2300}{52} = 44.2\ \frac{m}{s}$$

This is well within the range deduced from studying the video recording and reported by the Expert Panel to be between 30 and 70 m/s (para 26 above).

Since the CCTV recording cannot be used to assess the volume of post-explosion products, the rest of the analysis in para 27 above is equally invalid and, since the quantity of methane involved is unknown, the Commission's focus on the goaf of the hydro-panel was misplaced. In fact the Royal Commission (Pankhurst, Bell and Henry, 2012) did sound a note of caution i.e. "*The commission considers that there are some contentious issues relating to the source of the methane*".

The only reliable source of information comes from statements made at the time by the two survivors, Daniel Rockhouse, whose courage in saving himself and rescuing one other was in the finest tradition of the mining industry, and Russell Smith. Rockhouse was not actually in the drift but in the diesel refueling bay, just off the drift and roughly 2000m in from the portal and about 300m outbye of "Spaghetti Junction". Despite this he saw a bright flash, as did Smith, who was on his loader travelling inbye and at a point 300m outbye of Rockhouse. Neither of them suffered trauma injuries, supporting the idea that the shock wave was of low intensity, was running out of energy and was slowing down. Rockhouse made a comment shortly after the explosion that "it felt like some big fella just went boof into your chest and I fell over". Neither of the two survivors suffered burns and described the atmosphere following the explosion as warm but not hot. These comments do not help in identifying where the methane-air mixture was ignited or the size and composition of the methane-air cloud. The same observations could result from a nearby weak explosion of a small quantity of methane-air mixture, with the composition well away from the most explosive mixture or a larger, more violent explosion somewhere deeper in the mine workings. The stronger pressure wave from such an explosion would dissipate as it passed each intersection on its way outbye. The only clue remaining is the fact that both men saw a bright light.

Light does not propagate well in underground coal mines. Although coal is notoriously heterogeneous, a good estimate of its average reflectance is 5% (Daly, 1990), while stone in the roof can have higher values if clean. The floor, on the other hand is usually mud from the fine coal mined or broken from a soft floor and has about the same reflectance as coal. At and near Spaghetti Junction six roadways enter the path made up of the drift and Heading A. All roadways except Heading A have at least one 90º bend when entering the drift / Heading A route. Even Heading A bends 25º when joining the short transition road and there is a further bending of 35º as this route enters the drift. This tends to suggest that, for Rockhouse 300m down the drift and inside the diesel bay at the Pit Bottom Crushing Station, to see a bright flash, the explosion must have been close to Spaghetti Junction. However, any further comment would be pure speculation.

The last piece of evidence from the two survivors is their exposure to the products of the explosion, often referred to as afterdamp. Their symptoms have been attributed to breathing carbon monoxide, the effects of which are well documented. It can reasonably be assumed that both Rockhouse and Smith, in the minutes following the explosion had respiration rates at, or higher than, those of a person working. However, Rockhouse's immediate collapse after removing his SCSR and Smith's collapse before using it, suggest it was initially the oxygen deficiency that affected them. Thereafter, their descriptions are of carbon monoxide poisoning. Without detracting anything from Rockhouse's bravery and quick thinking in opening compressed air valves, the evidence would suggest a lower level of carbon monoxide than discussed at the Royal Commission. Of course, the "breathing" of the drift after the pressure wave exited and caused by the vacuum at the explosion epicentre would have diluted the afterdamp and so no firm conclusions can be drawn.

6.7.5 Conclusions

The Ermelo explosion occurred in the days before risk assessment became an integral part of the management of change. The sequence of events clearly indicated the need for ventilation specialists to be fully involved in the planning and execution of all mining changes at a mine. Communication, in a memo, of a list of tasks necessary to effect a simple change, without indicating the order in which they should be performed, led to the forging of one link in the chain of events leading to the disaster.

The lesson from the St Helena gold mine is that once a mine is declared a fiery mine, the restrictions imposed must be adhered to at all costs. There is a worrying trend in South Africa that cost saving exercises have removed "guards" employed at shafts to check for contraband and the increasing number of accusations of bias when random searches are conducted. The explosion occurred in a small chamber, off an intake shaft carrying 150 m^3/s of fresh air. This is a situation where no one could anticipate the accumulation of a large volume of methane-air in its explosive range. It also showed that, when studying the forces involved in an explosion, the old saying that "the weight attached to any piece of evidence is directly proportional to its mass" was again proved true. The forces necessary to topple a dam wall were far more significant than those necessary to buckle sheet metal enclosures. The destruction of the switchgear enclosures was eventually attributed to the post-explosion vacuum.

The Middelbult explosion exposed the need for real-time monitoring of section airflows and that the results should be displayed in a surface control room and monitored by a supervisor with no financial interest in any production bonus. This person must be empowered, in the event of unsatisfactory ventilation, to stop section production using control of the section belt. Such control may not have been effective at Middelbult because of the in-section non-compliance with mine standards and outright wilful disregard for safety. The investigation at Middelbult was thorough and the correlation between the area of burnt chevron tape and “rain spots” showed how the hot, expanded products of the explosion cool rapidly to 100 ºC and contract due primarily to the condensation of the water vapour.

An analysis of the known facts regarding the Pike River explosion shows the information presented to the Royal Commission regarding the volume of methane involved was fatally flawed and the truth is that nobody knows whether it was a large explosion deep in the mine or a smaller, weaker explosion near the inbye end of the access drift.

In the first three case studies discussed above, significant changes were in progress. At Ermelo it was a ventilation change within the production panel, at St Helena the building of the dam walls restricted ventilation and also made inspection of the area behind the walls difficult, while at Middelbult unplanned ventilation changes combined with irregular mining practice to create the conditions conducive for an explosion. Even at Pike River, which was still in the development phase, change was an everyday occurrence. Similarly a study of the literature reveals that a disproportionate number of explosions have taken place at a time of change. This leads to the obvious conclusion that the management of change is crucial and of equal importance to the management of risk during normal production.

CHAPTER 7 CONCLUDING REMARKS

This thesis, which is a record of some of my research involvement over the past 45 years, has been prepared for examination for the degree of Doctor of Engineering. The criteria for success are set out in the Faculty of Engineering and the Built Environment's Rules and Syllabuses. There are eight exit level outcomes listed and the thesis should be used in its entirety to demonstrate whether they have been achieved. However, the Faculty's requirements go much further and the associated assessment criteria are that "the applicant demonstrates, through a coordinated compendium of original, published work submitted for examination, a distinguished contribution to the practice of engineering; the work must be a record of engineering development of major, technological, economic or social significance carried out under the technical direction of the learner". This final chapter of the thesis highlights two or three aspects from each of the five research chapters in an attempt to demonstrate such a contribution to mining engineering knowledge. While in the United Kingdom and Australia I was in a position to undertake hands-on research. However, at the University of the Witwatersrand, most of my research effort was devoted to the technical direction and supervision of postgraduate students.

Chapter 2 deals with my research into the basics principles of rock excavation using mechanical tools. In undertaking this research I was extremely fortunate in two ways. Firstly, the rock type of interest at that time in the United Kingdom was the Bunter Sandstone, a rock known for its homogeneous and isotropic nature. Secondly, while planning the experimental work I was taught the intricacies of partial factorial experiments and orthogonal Latin Squares by Professor Protodyakonov. His six week visit to the University of Newcastle upon Tyne in 1973 was one of the rare occasions when he was permitted to travel outside the Soviet Union and I believe this was one of the first applications of partial factorial planning of Rock Engineering experiments. These two factors combined to produced experimental data which confirmed and gave clarity to the results of previous researchers. The paper presented at the 3rd Congress of the International Society of Rock Mechanics contained clear, unambiguous relationships between the geometry of cutter picks and their performance when operating in a cutting array (Roxborough and Phillips, 1974).

During the early 1970s the Bunter Sandstone was a rock formation of great interest in the United Kingdom due to the number of tunnels being planned and the projected use of tunnel boring machines. These machines generally use free-rolling disc cutters and the same partial factorial planning of multivariate disc cutting experiments yielded excellent empirical results. However, the highlight of my research at the University of Newcastle upon Tyne was my theoretical analysis of the forces acting on the disc, which led to equations predicting thrust and rolling forces and resulted in a simple model explaining that the optimum spacing to penetration ratio for discs is the same ratio as that of compressive to shear strength of the rock being excavated. This was published in the prestigious, peer-reviewed International Journal of Rock Mechanics and Mining Science

(Roxborough and Phillips, 1975) and the theory was validated by other experimental data (Morrell et al, 1970) and (Phillips and Bilgin, 1977).

The two papers (Roxborough and Phillips, 1974) and (Roxborough and Phillips, 1975) were my most significant contribution to rock cutting mechanics. However, this material formed part of my Ph.D. thesis and cannot be re-examined here. On the other hand, the unambiguous relationships obtained proved invaluable in interpreting the sparse data obtained in cutting coal (Grant, Pieterse and Phillips, 1981) and (Phillips, Grant and Pieterse, 1983) and later in establishing the relationships set out in a chapter of a book (Bilgin and Phillips, 1995)

The building of a laboratory and the creation of a research group working on the problem of spontaneous combustion is dealt with in Chapter 3. The start of this research coincided with my appointment as Head of the Department of Mining Engineering in 1986 and the realisation that, in future, my contribution to research would be through the technical direction of the postgraduate students I supervised. Between 1987 and 2004 this group successfully completed 3 M.Sc. dissertations and 4 Ph.D. theses.

One of the earliest breakthroughs and certainly the most enduring aspect of this work was the realisation that crossing-point temperatures and the stage II slope of a coal thermogram are independent predictors of spontaneous heatings and that they could be combined to form a spontaneous combustion index. Initially a graphical method was used where the inverse of crossing-point temperature is plotted on the y-axis and the stage II slope on the x-axis. By joining the two points and calculating the triangular area under the graph the two parameters are taken into account (Gouws, 1987) and (Gouws and Phillips, 1993). This method was refined and tested and became the Wits-EHAC Index, which is still used by the South African coal mining industry. The laboratory developed in the 1980s is also still used on a regular basis to test approximately 25 coal samples a year.

While the development of the Wits-EHAC Index was the most significant single aspect of this work, the research undertaken for the Grootegeluk open-pit coal mine was the most exciting. This mine dumped over 30 million tons of waste material (inter-burden and coal preparation plant discards) every year. This material ranged from mildly reactive to highly reactive and the surface dumps created significant environmental problems due to the spontaneous heating of this material. Adamski, an experienced mining engineer and the pit-superintendent, researched the possibility of building dumps with the most reactive material at the core, with material of gradually increasing reactivity forming the outer layers, thereby consuming oxygen before it could reach the core (Adamski, 2003). Since he proposed backfilling the open-pit with this material (and thereby putting the mine at risk), his research needed to be meticulous and comprehensive. Conceptual thinking was followed by laboratory work, pilot scale studies and then the first three years of dumping 7 million tonnes a year of material in strict sequence. As a precaution this dump was continuously monitored by buried thermocouples. It is extremely rare for a Ph.D. study to embrace a project of this magnitude and it was a privilege to be involved.

During the 1970s and 1980s the Chamber of Mines Research Organisation was by far the largest, private research organisation in the Southern Hemisphere. Their main endeavour was undertaking research for the gold mining industry, including a sustained programme looking at heat and heat stress in workers as well as ways to cool mines. COMRO's successor organisation (CSIR Miningtek) continued working in this field from 1992 onwards. In 1998 the Deepmine Programme of research was launched and since the research was aimed at enabling gold mining to continue down to a depth of 5 km, refrigeration, cooling and ventilation were key components of the programme. While the bulk of the research funding went to Miningtek or to its previous staff, the Department of Mining Engineering at Wits was given a small project and asked to review the current status of the insulation of chilled water pipes. This was my only major excursion into gold mining research and came as an indirect consequence of the Kinross mine fire in 1987, when 177 men lost their lives due to toxic fumes generated by burning polyurethane foam. Until then chilled water for cooling deep mines was delivered by pipes insulated with a variety of unsafe materials. By stripping out the old insulation and, in many cases, replacing it with less efficient insulation the cost of cooling deep mines rose sharply.

Deepmine's concern was that the costs of providing mine cooling at depths of 5 km would render the industry financially unsustainable and the research undertaken in this area is reported in Chapter 4 of this thesis. A staff member in the School elected to investigate this problem as his M.Sc. study (Rawlins, 1998) and his Ph.D. study (Rawlins, 2002) and I supervised both the dissertation and his thesis. As the scope of the Deepmine project expanded, we were asked to look at mine cooling strategies as well as the insulation of the chilled water pipes, the use of backfill to prevent heat ingress in back areas of the stopes and finally, since other Deepmine projects were looking at mechanising mining operations, to assess the effects heat loads from machinery would have on ventilation and cooling strategies.

Over a five-year period, Rawlins and I published our results in four papers Probably the most interesting finding was that although backfill provided insulation, it introduced a heat load into the mine during its transport and placing. The cost of cooling this additional heat load plus the costs associated with backfilling would out-way the savings due to its insulation effects. However, since backfill would be placed in any case for rock engineering reasons, the savings in cooling requirements due to its insulation against heat ingress in back areas would be substantial (Rawlins, Phillips and Jones, 2002).

Chapter 5 deals with my research interest in the monitoring of respirable dust exposure in coal mining. This began in 1979 when I was asked to undertake work at one mine in New South Wales, using a newly acquired SIMSLIN instrument. Being very new to this type of monitoring and being unqualified in the discipline of occupational hygiene, I needed to study the literature and I published a review (Phillips, 1980). Since then I have been the sole author of two papers and been the co-author of 8 papers with Belle, the most recent being in 2013.

Although the research with Dr Belle explored a wide range of issues associated with respirable dust conditions in and around the headings of room-and-pillar mining in South Africa, the two most influential pieces of research I conducted in this area were those that helped the coal industries in both New South Wales and also in South Africa move to gravimetric sampling. My work in New South Wales was initiated because of my membership of the Joint Coal Board's Standing Committee on Research and Control and was clearly focused on producing a gravimetric standard equivalent or better than the existing standard, thereby protecting the workforce from pneumoconiosis. This involved a preliminary visit and a full shift of sampling underground at each of 29 mines across the State. The results were presented to the NSW coal industry in a journal paper (Phillips, 1984a) and opened to international scrutiny at the Third International Mine Ventilation Congress (Phillips, 1984b). It was gratifying that the results of this work were used to draft changes to the legislation in New South Wales, making gravimetric monitoring and gravimetric standards the prescribed method of undertaking respirable dust sampling from February 1984.

The influence of my research, while on sabbatical leave and working in South Africa at COMRO, was less direct. A research report was produced (Phillips, Pedroncelli and Ribbink, 1981) but permission to publish the data was withheld. The research mandate was to investigate the effectiveness of dust suppression techniques on continuous miners. These machines were very new to the South African industry and the research programme was intended to evaluate different locations of spray blocks and to gauge the effects of using different water pressures. An incidental product of this research was the ability to correlate the longstanding measure of respirable dust (P.E.R.) with gravimetric readings. This was done at only two mines and could not be compared with the programme undertaken in NSW. However, the most important aspect of the work was the first real use of gravimetric sampling in South African coal mines and the high gravimetric values obtained. While not definitive, the programme exposed dust levels which could be up to 15 times the statutory limits set in the United States. Although the report was for restricted circulation to all coal mining members of the Chamber of Mines of South Africa, it did result in significant interest and it was the industry that exerted pressure for gravimetric sampling to be introduced.

The longest chapter in this thesis is Chapter 6, which deals with methane release rates and coal mine explosions. This is no coincidence since this is where the bulk of my research effort has been directed since my arrival in South Africa. Between 1983 and 1993, 253 mineworkers died in 5 major explosions, while over 20 others died in smaller ignitions and explosions. This was a situation that could not continue and a sustained research effort was required. I believe that I have made some contribution in a number of ways.

My appointment to the Explosion Hazard Advisory Committee in 1985 was the beginning of a long involvement in committee and administration work. Appointment to the Kloppersbos Research Advisory Committee (1987-1992) followed and this was the start of a busy period as a member of SIMRAC's

Collieries Environmental Engineering Advisory Group and chair of its Special Interest Group on Explosion Hazards (SIGEH). The commissioning of the 200m gallery, undertaken by Cook but under my technical direction, was a major commitment of time but was rewarded by seeing changes in the stone dusting requirements incorporated in new legislation. The specification and costing of the 20m tunnel was undertaken in collaboration with the late Dr Jan Oberholzer but the preparation of a compelling motivation became my responsibility.

The Leon Commission of Inquiry into mine safety and health was a watershed in the approach to mine safety in South Africa. As Chair of SIGEH I was asked by the Chamber of Mines to prepare the industry's position paper on the explosion hazard. I did this with the assistance of Landman (Phillips and Landman, 1993). While the Leon Commission Report (Leon et al, 1995) was enthusiastically embraced by the industry, the recommendations were based on the simple premise that those who create risk should manage it. This worked well in the 1990s when the South African mining industry was dominated by six or seven mining houses, each with large technical teams at their respective head offices. These structures were ideally equipped to advise the individual mines on technical issues but with the passage of time they have largely disappeared. Perhaps it is time to question whether a two-tier system is now needed, with mines deemed to be capable of managing their risks being allowed to do so, while those operations without this expertise returning to prescriptive regulations.

On the research side there were two projects that made a significant contribution to changing industry thinking. The first was Dr Gys Landman's Ph.D. that determined explosibility of hybrid mixtures by mapping the contours involving methane concentration, explosible dust concentration and ignition energy, using the 40 litre chamber at Kloppersbos. This, combined with his underground measurements, alerted the industry to the fact that even in a low methane environment an explosive mixture is constantly present within the cutting drum of a continuous miner and it is the application of sufficient water in droplets that prevents an ignition. When published, (Phillips and Landman, 1992) and (Landman, 1992), this really was a wake-up call for local industry.

The second major project with which I was involved was the bagged stone dust barrier. Jan du Plessis (now Professor du Plessis) had already completed much of this work and reported it to the committees where I was involved. However, it needed a final push to be accepted by an industry dominated by room-and-pillar workings. This was when Du Plessis and I negotiated the use of the Lake Lynn facility in the United States (Du Plessis and Phillips, 1998). The research completed in the room-and-pillar section of Lake Lynn finally persuaded the South African industry to adopt the bagged stone dust barrier as the standard protection against a propagating coal dust explosion (Du Plessis and Phillips, 2000). In 1998 Du Plessis registered as a Ph.D. student under my supervision and, as with Landman, it was a pleasure to work with him.

My involvement in the investigation of multi-fatality explosions gave me a unique insight into the causes and effects of explosions. "In South African mines, whenever there is a chance of saving lives, mines rescue teams are deployed based on a careful analysis of the risks and the experience of professional Mines Rescue Service personnel. Because of this approach the site of each of the South African explosions listed in Table 6.10 was made accessible to investigation teams shortly after the event and detailed analyses of each event were possible. Access to explosion sites, although at times a harrowing experience, was extremely useful in understanding the problem and commissioning of appropriate research and other measures to address the problem in practical ways. However, it is probable that investigations conducted in the 1980's and 1990's would not be conducted today due to different attitudes to safety and the risk based approach to mine safety" (Phillips, 2015).

Since 1993 there hasn't been a major, multi-fatality explosion in a South African coal mine. However, it would be totally wrong to claim this was solely due to the research undertaken. Many individuals in the industry have worked tirelessly to improve safety in general and, in particular, to prevent ignitions and explosions. Every mine has undertaken extensive risk assessments and produced Codes of Practice. If one single element in this multi-faceted approach to preventing explosions needs to be singled out it is the visits to Kloppersbos, where after a lecture on ventilation, methane and coal dust explosibility, groups of mineworkers observe a coal dust explosion in the 200m gallery. These training exercises have by now influenced virtually every worker in the industry.

In conclusion, the obvious and primary reason for preparing this thesis is to submit it for examination for the degree of Doctor of Engineering. A secondary but important reason has been my desire to collect together a body of my own publications (in Appendix B) and pertinent publications (in Appendix C). While many of these papers, reports and theses may now be considered dated, I have done this in the hope that, in the future, they may be of help to some student of mining engineering.

REFERENCES

Numbers in parenthesis, e.g. $[B1\ or\ C4]$ *indicate if the reference is available in Appendix B or C.*

Adamski, S. (2003) The prevention of spontaneous combustion in back-filled waste material at Grootegeluk coal mine, Ph.D. Thesis, Faculty of Engineering and the Built Environment, University of the Witwatersrand, p.1– 153 [C1]

Anon http://www.engineering-timelines.com/who/Brunel_IK/brunelIK3.asp

Belle, B.K. & Phillips, H.R. (2001) Analysis of dust concentration levels in a 30m heading development section with a 12m cutting sequence rule, Proceedings of the 7th International Mine Ventilation Congress, Cracow, June, p. 307-313 [B1]

Belle, B.K. & Phillips, H.R. (2002) Monitoring of low, medium and high respirable dust concentrations using South African cyclones, Journal of the Mine Ventilation Society of South Africa, Vol. 55, No. 2, April/June, p. 49-53 [B2]

Belle, B.K. & Phillips, H.R. (2003) Recent developments in the control of respirable dust concentrations following the 12m directive, Journal of the Mine Ventilation Society of South Africa, Vol. 56, No. 3, July/Sept, p. 85-90 [B3]

Belle, B.K. (2004) Development of a dust exposure level index (DELI) for South African underground coal mine workers, Ph.D. thesis, Faculty of Engineering and the Built Environment, University of the Witwatersrand, p.1-510 [C2]

Belle, B.K. & Phillips, H.R. (2006) Dust samples for exposure assessment in underground coal mines, Proceedings 11th North American Underground Mine Ventilation Conference, The Pennsylvania State University, June 5-7

Belle, B.K. & Phillips, H.R. (2007a) Use of peak exposure data for exposure and engineering dust control purposes, Journal of the Mine Ventilation Society of South Africa, Vol. 60, No. 1, Jan/March, p. 6-9 [B4]

Belle, B.K. & Phillips, H.R. (2007b) — Influence on section intake dust levels as an exposure assessment parameter in the dust exposure level index, Journal of the Mine Ventilation Society of South Africa, Vol. 60, No. 2, April/June, p. 42-47 [B5]

Belle, B.K. & Phillips, H.R. (2009) — Influence of section return dust levels as an exposure indicator and engineering control parameter in coal mines, Journal of the Mine Ventilation Society of South Africa, Vol. 62, No. 4, Oct/Dec, p. 6-9 [B6]

Belle, B.K. & Phillips, H.R. (2013) — Contamination of stone dust in the personal coal dust samples for exposure assessment in underground coal mines, Journal of the Mine Ventilation Society of South Africa, Vol. 66, No. 4, Oct/Dec, p. 12-14 [B7]

Bilgin,N. Phillips, H.R. & Yavuz, N. (1992) — The cuttability classification of coal seams and an example of mechanical plough application in Eli Darkale coal mine. Proceedings of the 8th Coal Congress of Turkey, Zonguldak, p. 31-52 [B8]

Bilgin, N. & Phillips, H.R. (1995) — Chapter 9 – Mechanical properties of coal COAL – Resources, Properties, Utilization and Pollution, Ed O.Kural ISBN 975-95701-1-4, p. 127-138 [B9]

Billenkamp, E.G. (1988) — Methane desorption characteristics of selected South African coal seams, M.Sc. dissertation, Faculty of Engineering, University of the Witwatersrand, p. 1-214 [C3]

Budavari, S. & Phillips, H.R. (1996) — Centennial reflections of the Department of Mining Engineering at the University of the Witwatersrand, J. of the SAIMM, Vol. 96, No 5, Sept., p. 185-195 [B10]

Chabedi, K.C. & Phillips, H.R. (2012) — The future contribution of coal to the energy requirements of South Africa, Cuprum, Vol 64, No 3, p. 15-25 [B11]

Chakravorty, R.N. & Kolada, R.J. (1988) — Prevention and control of spontaneous combustion in coal mines, Mining Engineering, Society of Mining Engineers, USA, October 1988

Churchyard, G.J. & Corbett, E.L. (2001) — Tuberculosis and associated diseases, Chapter 6, Handbook of Occupational Health Practice in the South African Mining Industry, Safety in Mines Research Advisory Committee, p. 153-192

Condran, B. (2009) — Recollections of mining through outburst conditions, Proceedings of Seminar, http://miningst.com/Outburst/presentations_publications/outburst_2009/OB%20Seminar%201109%20Condran.pdf) Accessed 3/08/2017 [C4]

Cook, N.G.W. (1968) — Analysis of hard rock cuttability for machines, Rock Excavation – Problems and Progress, Proceedings of Tunnel and Shaft Conference, Sept., p. 39

Cook, A.P. (1999) — Methane conditions in goafs, M.Sc. Dissertation, Wits University, Johannesburg, p. 1-166 [C5]

Cook, P.M. (1992) — The inhibition of coal-dust explosions with stone dust in a large-scale explosion gallery, M.Sc. Dissertation, University of the Witwatersrand, Johannesburg, p. 1-101 [C6]

Creedy, D.P. & Phillips, H.R. (1997) — Methane layering in bord and pillar workings, SIMRAC Research Report COL 409, p. 1-116 [B12]

Creedy, D.P., Burton, C., Sharman, A., Phillips, H.R. & Alexander, J.J. (2000) — Inertisation of coal mine panels using microbial organisms, SIMRAC Research Report COL 604, p. 1-22 [B13]

Daly, C.R. (1990) — The lighting of underground coal mines, Ph.D. thesis, University of New South Wales, p. 1-280

Department of Minerals and Energy, Republic of South Africa, 1994 — Guidelines for the gravimetric sampling of airborne particulates for risk assessment in terms of the Occupational Diseases in Mines and Works Act, No 78 of 1973

Du Plessis, J.J.L. & Phillips, H.R. (1998) — Effectiveness of dispersed stone dust barriers in bord-and-pillar mining. Phase 1-Evaluation of the Lake Lynn facility, SIMRAC Report COL 501, p. 1-45 [B14]

Du Plessis, J.J.L. & Phillips, H.R. (2000) — Development and implementation of the bagged stone dust barrier, Coal – The Future, 12th International Conference on Coal Research, SAIMM, Johannesburg, p. 107-111 [B15]

Du Plessis, J.J.L. (2000) — Development and evaluation of the baqgged stone dust barrier, Ph.D. Thesis, University of the Witwatersrand, Johannesburg, p. 1-454 [C7]

Du Plessis, J.J.L. (2001) — Evaluation protocol for active machine-mounted explosion suppression systems for continuous miners, CSIR Miningtek Report No. EC01-0149, p. 1-8 [C8]

Du Plessis, J.J.L., Phillips, H.R., Lebecki, K.& Belle, B.K. (2014) — Explosible dusts and mine explosion prevention, chapter in Ventilation and Occupational Environmental Engineering in Mines, ed. Du Plessis, MVSSA, p. 577-604 [B16]

Du Plessis, J.J.L., Saleh, J.K. & Phillips, H.R. (2017) — Coal mine explosions and the development of mitigating controls, 16th North American Ventilation Conference, Denver, [B17]

Eroglu, H.N., Gouws, M.J. & Phillips, H.R. (1991) — Spontaneous combustion liability contours, Proceedings of the 12th Scientific and Technical Mining Congress, Ankara, May 1991, p. 41-51

Eroglu, H.N. (1992) — Factors affecting the spontaneous combustion index, Ph.D. Thesis, University of the Witwatersrand, p. 1-269 [C9]

Eroglu, H.N. & Moolman, C. (2003) — Develop methods to prevent and control spontaneous combustion associated with mining and subsidence, CSIR Report to Coaltech 2020, Project 3.4.1, March 2003, p. 1–386 [C10]

Evans, I. (1962) — A theory of the basic mechanics of coal ploughing, Proceedings of International Symposium on Mining Research, London, Pergamon Press, 1962, Vol. 2.

Evans, I. & Pomeroy, C.D. (1966) — The strength, fracture and workability of coal, London, Pergamon Press, p. 1-277

Falcon, R.M.S. (1986)	The petrographic characteristics of South African coals in relation to spontaneous combustion, Proceedings of Symposium on the Spontaneous Heating of Coal, South African Coal Processing Society, Johannesburg, 1986
Falcon, L.M. & Falcon, R.M.S. (1987)	The petrographic composition of Southern African coals in relation to friability hardness and abrasive indices, Journal of South African Institute of Mining and Metallurgy, Vol. 87, No.10, p.323-336
Farquharson. D.C. (1988)	Grootegeluk spontaneous combustion – a practical experience: Air permeability, Comparison of heap behaviour, ISCOR Internal Report
Feng, K.K., Chakravorty, R.N. & Cochrane, T.S. (1973)	Spontaneous combustion – a coal mining hazard, C.I.M. Bulletin, Oct. 1973, p. 71-75
Fisher, Sir R.A. (1970)	Statistical methods for research workers, Oliver and Boyd, 1970, p. 1-332
Flint, J.D. (1990)	Mine gas and coal dust explosions and methane outbursts – their causes and prevention, M.Sc. Dissertation, University of the Witwatersrand, p. 1-807
Foreman, H.M. & Adams, T.H.F. (1953)	Problems in the operation of a cutter plough at Thrislington Colliery, Journal of King's College Mining Society, University of Durham(U.K.)
Genc, B. (2000)	Coal mine explosion suppression using active and passive barrier systems, M.Sc. dissertation, University of the Witwatersrand Johannesburg, p. 1-171
Genc, B. (2002)	The effectiveness of on-board suppression systems to prevent the propagation of methane ignitions under simulated coal mining conditions, Journal of the South African Institute of Mining and Metallurgy, Vol. 102, No. 1, Jan/Feb, p.49-58 [C11]
Genc, B. & Cook, A. (2013)	Spontaneous combustion liability of South African coals, Journal of the Mine Ventilation Society of South Africa, Vol. 66, No 4, Oct/Dec. p.15 – 19 [C12]

Genc, B. & Cook, A. (2015) — Spontaneous combustion risk in South African coalfields, Journal of the Southern African Institute of Mining and Metallurgy, Vol. 115, No. 7, July, p. 563 – 568 [C13]

Glasser, D. (1990) — Spontaneous combustion in beds of coal, Handbook of Heat and Mass Transfer,, Vol. 4, Advances in Reactor Design and Combustion Science, 26, Ed N. Chemisinoff, New Jersey, p. 1071 – 1148

Gooding, K. (2010) — Aberfan-inspired work, Fifth Column, Mining Journal, April 9, p.14-15 [C14]

Gouws, M.J. (1987) — Crossing point characteristics and thermal analysis of South African coals, M.Sc. Dissertation, University of the Witwatersrand, p. 1- 110 [C15]

Gouws, M.J., Gibbon, G.J., Wade, L. Phillips, H.R. (1991) — An adiabatic apparatus to establish the spontaneous combustion propensity of coal, Journal of Mining Science and Technology, Vol. 13, p. 417-422

Gouws, M.J. (1993) — A spontaneous combustion liability index based on ignition temperature tests and adiabatic calorimetry, Ph.D. Thesis, University of the Witwatersrand, p. 1-287 [C16]

Gouws, M.J. & Phillips, H.R. (1993) — The propensity of coal seams for spontaneous combustion, Proceedings of the 25th International Conference of Safety in Mines Institutes, September 1993, Vol. 3, p.139-147

Gouws, M.J. & Phillips, H.R. (1995) — Post explosion guidance systems, Proceedings of the 26th International Conference of Safety in Mines Research Institutes, Poland, Vol.2, p. 107-120 [B18]

Government of Queensland (2016) — Coal workers' pneumoconiosis, Department of Natural Resources and Mines, Government of Queensland, p. 1-2

Grant, I.L., Pieterse, J. & Phillips, H.R. (1981) — Coal cutting using chisel picks, Chamber of Mines of South Africa Research Organisation, Report No 11/82, December 1981, p.1-99 [B19]

Haldane, J.S., Martin, J.S. & Thomas, R.A. (1904)	Report to the Home Secretary on the Health of Cornish Miners, Hansard, House of Commons, U.K.
Hayes, P.J. (1983)	Some aspects of the gas emission characteristics of coal, M.Eng. dissertation, University of New South Wales, pps147 [C17]
Hardman, D.R., Creedy, D.P. & Phillips, H.R. (2001)	Strategies to monitor non-homogeneous atmospheres in sealed off panels in coal mines, SIMRAC Research Report COL 602, p. 1-31 [B20]
Jacobsen, M., Rae, S., Walton, W.H. & Rogan, J.M. (1971)	The relation between pneumoconiosis and dust-exposure in British coal mines, In Inhaled Particles III, Unwin Bros, London, Vol 2, p. 903-917
Jeppe, C.B. (1946)	Gold Mines on the Witwaterstand, Volume II, Transvaal Chamber of Mines.
Jones, M.Q.W. & Rawlins, C.A. (2001)	Thermal properties of backfill from a deep South African gold mine, Journal of the Mine Ventilation Society of South Africa, Volume 54, No 4, Oct/Dec 2001, p.100-105
Joubert, F.E. (1969)	Methane survey in Indumeni Colliery, Fuel Research Institute of South Africa, Technical Memorandum No. 44 of 1969.
Karmis, M., Hebblewhite, B., De Ruiter, H., Scoble, M., Cedron, M. & Phillips, H. (2015)	Global mining engineering education: past, and future, Extracting the Science: A Century of Mining Research, SMME, p. 490-500
Kenny, L., Baldwin, P.E.J., & Maynard, A.D. (1998)	Respirable dust sampling at very high concentrations, Report of the Health and Safety Laboratory, HSE, UK
Khan, F.J. & Mosenifar, Z. (2017)	Coal worker's pneumoconiosis, Medscape, http://emedicine.medscape.com/article/297887-overview, accessed 11 June 2017.
Kim. A.G. & Chaiken, R.F. (1993)	Fires in abandoned coal mines and waste banks, Information Circular 9352, U.S. Bureau of Mines, p. 1 – 64

Kininmonth, R.J. (1981)	Summary of investigations into Appin Colliery explosion, Proceedings of Symposium on Ignitions, Explosions and Fires, Aus. IMM, Wollongong, p. 9.1-9.16
Landman, G.vR.(1988)	Thin seam extraction at the Durban Navigation Collieries, M.Sc. Project Report, University of the Witwatersrand Johannesburg, p. 1-298
Landman, G.vR. (1992)	Ignition and initiation of coal mine explosions, Ph.D. Thesis, University of the Witwatersrand, p. 1-250 [C18]
Landman, G.vR. & Phillips, H.R, (1993)	Explosibility of methane/dust mixtures, Proceedings of the 25th International Conference of Safety in Mines Research Institutes, Pretoria, p. 49-58
Leigh, J. (1978)	Health effects on coal mining operatios, Proceedings of Workshop on Coal and Environmental Studies, Macquarie University, Sydney, p. 1-12.
Leon, R.N., Salamon, M.D.G., Davies, A.W. & Davies, J.C.A. (1995)	Report of the commission of inquiry into safety and health in the mining industry, Department of Mineral and Energy Affairs, Republic of South Africa [C19]
Lingard, P.S., Phillips, H.R., & Doig, I.D. (1982)	The permeability of some Australian coals, Proceedings of Symposium on Seam Gas Drainage, Aus.I.M.M., Wollongong, p. 70-80 [B21]
Lingard, P.S., Phillips, H.R. & Doig, I.D. (1983)	An automated five cell triaxial test bed for gas-mechanical measurements on coal and rock cores. Proceedings of Symposium on Alleviation of Coal and Gas Outbursts, UNSW, September, p. 13-18 [B22]
Lingard, P.S., Doig, I.D. & Phillips, H.R. (1984)	Laboratory studies on the sorption characteristics and permeability of triaxially stressed coal samples. Proceedings of the Third International Mine Ventilation Congress, Harrogate, U.K., June 1984, p 39-47 [B23]
Mahadevan, V. & Ramlu, M.A. (1985)	Fire risk rating of coal mines due to spontaneous heating, Journal of Mines, Metals and Fuels, Aug.1985, p.357-3

McPherson, M.J. (1993)	Subsurface ventilation and environmental engineering, Chapman and Hall, p. 1-905
Meyer, C.F., Doyle, B.A. & Phillips, H.R. (1994)	Evaluation of the suitability of using jet fans to ventilate continuous miner headings in collieries, SIMRAC Report COL 116, p. 1-53 [B24]
Miners' Phthisis Committee (1916 and 1919)	General Report, 1916 and Final Report 1919, Union Government of South Africa
Mining-technology.com (2013)	The top ten deepest mines in the world http://www.mining-technology.com/features/feature-top-ten-deepest-mines-world-south-africa/
Monash University (2016)	Review of respiratory component of the coal mine workers' health scheme for the Queensland Department of Natural Resources and Mines, Monash Centre for Occupational and Environmental Health, School of Public Health and Preventative Medicine, July, p. 1-116 [C20}
Morrell, R.J., Bruce, W,E. & Larson, D.A. (1970)	Tunnel boring technology – disc cutter experiments in sedimentary and metamorphic rocks, U.S. Bureau of Mines, R.I. 7410, p.32
Morris, R. (1986a)	An historical note of the research into the causes of spontaneous combustion prior to 1750, Journal of the Mine Ventilation Society of South Africa, December, p 168-170 [C21]
Morris, R. (1986b)	Spontaneous combustion in coal mines and the interpretation of the state of a mine fire behind the stoppings, Ph.D. Thesis, University of Nottingham, U.K.
Morris, R. & Badenhorst, G.P. (1987)	An analysis of underground fires in South African coal mines from 1970 to 1985, Journal of the Mine Ventilation Society of South Africa, Vol. 40, No. 10, October, p. 134-139 [C22]
Moss, K.N. (1927)	Gas, dust and heat in mines, Charles Griffin and Co, Ltd, London p 1- 233

Musingwini, C., Cruise, J.A. & Phillips, H.R. (2013)	A perspective on the supply and utilisation of mining graduates in the South African context, Journal of the Southern African Institute of Mining and Metallurgy, Vol. 113, No. 3, March, p 235-241 [B25]
MVSSA, 1987	Environmental engineering in South African mines, Handbook, Mine Ventilation Society of South Africa, p. 1-987
Oberholzer, J.W. (1995)	The establishment of testing facilities for active onboard suppression barrier systems for use in South African collieries, Proceedings of the 26th International Conference of Safety in Mines Research Institutes, Vol. 2, p. 161-172 [C23]
Oberholzer, J-W. (1998)	Develop information and training aids to assist all employees to recognise frictional ignition hazards, SIMRAC Report COL 431, p. 1-219 [C24]
O'Reilly, M.P., Roxborough, F.F. & Hignett, H.J. (1976)	Programme of laboratory, pilot and full-scale experiments in tunnel boring, Proceedings of International Symposium Tunnelling '76, IMM, London, March, p 287-299 [C25]
Orenstein, A.J. (ed.) (1959)	Proceedings International Pneumoconiosis Conference, Johannesburg, Churchill Press, London, p. 1-632
Pankhurst, G., Bell, S. & Henry, D. (2012)	Report of the Royal Commission on the Pike River coal mine tragedy, available at http://pikeriver.royalcommission.govt.nz/Final-Report
Phillips, H.R. (1968)	The development of electrical engineering in the mining industry, Undergraduate Report, B.Sc.(Eng.), University of Bristol, p.1-55 [B26]
Phillips, H.R. (1971)	Dynamic analysis of a coal ploughing system by digital simulation, M.Sc. dissertation, Faculty of Applied Science, University of Newcastle upon Tyne, September 1971, p. 1- 86 [B27]
Phillips, H.R. (1975a)	Rock cutting mechanics related to the design of primary excavation systems, Ph.D. thesis, Faculty of Applied Science, University of Newcastle upon Tyne, October 1975, p. 1-284 [B28]

Phillips, H.R. (1975b)	The mechanical cutting characteristics and properties of selected rock formations, Department of Mining Engineering, University of Newcastle upon Tyne, December 1975, p. 1- 304 [B29]
Phillips, H.R. & Fowell, R.J. (1976)	Chalk cutting research – a laboratory study, Proceedings of Symposium on Chalk Cutting, Transport and Road Research Laboratory, London, November 1976, p. 1- 36 [B30]
Phillips, H.R. & Bilgin, N. (1977)	Correlation of rock properties with the measured performance of disc cutters, Proceedings of the Conference on Rock Engineering, British Geotechnical Society, April 1977, p. 181-196 [B31]
Phillips, H.R., Bilgin, N. & Price, D.L. (1978)	The influence of tyre tip geometry on the design of disc cutter arrays, Proceedings of the 3rd Australian Tunnelling Conference, Sept. 1978, p. 48-52 [B32]
Phillips, H.R. & Roxborough, F.F (1979a)	The design and operation of cutter picks in mining, Proceedings of Fourth Tewksbury Symposium on Fracture Mechanics, Melbourne, Feb. 1979, p 6.1-6.18 [B33]
Phillips, H.R. & Roxborough, F.F. (1979b)	The contribution of R&D to the design of mechanised mining systems, Proceedings of the International Conference on mining machinery, Brisbane, p 512-518
Phillips, H.R. (1980)	Gravimetric methods of monitoring airborne respirable dust in coal mines, Australian Coal Miner, Vol. 2, No. 4, p. 26-29
Phillips, H.R. & Drummond, W.H. (1980)	Coal clearance on a longwall face, Australian Coal Miner, Vol. 3, No. 3, p 32-39
Phillips, H.R. & Roxborough, F.F. (1981)	The influence of tool material on the wear rate of cutter picks in mining, Proceedings of Conference “Metals in Mining”, Australian Institute of Metallurgy and Mining, Surfers Paradise
Phillips, H.R., Pedroncelli, E.J. Ribbink, E.I. & Jones, J.S. (1981)	An investigation of the effectiveness of dust suppression techniques for continuous miners, Chamber of Mines of South Africa Research Organisation, Report No. 31/82, p. 1-51 [B34]

Phillips, H.R., Grant, I.L. & Pieterse, J. (1983) — Laboratory coal cutting with corner cutting conical picks, Chamber of Mines of South Africa Research Organisation, Report 35/83, p. 1-139 [B35]

Phillips, H.R. (1984a) — The basis for the change to gravimetric dust standards in N.S.W., The Australian Journal of Coal Mining Technology, Vol. 6, May, p. 39-47

Phillips, H.R. (1984b) — New methods and standards for respirable dust monitoring in the New South Wales coal industry, Proceedings of the Third International Mine Ventilation Congress, Harrogate, U.K., p 203-208 [B36]

Phillips, H.R., Wait, D.B., Billenkamp, E. & Stripp, G. (1987) — The prediction of methane release rates for South African coal seams, Proceedings of the Symposium on Safety in Coal Mining, CSIR Pretoria, Paper 5.2, p. 1-16 [B37]

Phillips, H.R. (1988) — Who will manage the mining industry in the year 2000? Human Resources Management Journal, Vol. 4 No. 1, p. 9-10

Phillips, H.R. & Sullivan, P. (1991) — Fires in shallow abandoned coal mines, Proceedings of Symposium on Environmental Protection, Mine Ventilation Society of South Africa, p. 1-15 [B38]

Phillips, H.R. & Landman, G.vR. (1992) — The explosion potential of methane/dust mixtures at the coal face, Proceedings of the 5th International Mine Ventilation Congress, Johannesburg, p. 71-78 [B39]

Phillips, H.R. & Landman, G.vR. (1993) — Position paper on coal mine explosions, Mining Industry Submission to the Leon Commission of Inquiry, p. 1-24 [B40]

Phillips, H.R. (1994) — Mining Education in South Africa, Proceedings XVth Council of Mining and Metallurgical Institutes Congress, SAIMM, Johannesburg, Vol. 1, p 287-293 [B41]

Phillips, H.R. & Landman, G.vR. (1994) — The development of a discussion document on the safety related research needs of the coal industry, SIMRAC Report COL 034, p.1-46 [B42]

Phillips, H.R. & Gouws, M.J. (1995) Development of active guidance systems to overcome problems of disorientation and low visibility following severe explosions, SIMRAC Report COL 108, p.1-32 [B43]

Phillips, H.R. (1995) Coal mine explosions – the South African experience, Proceedings of the 26th International Conference of Safety in Mines Research Institutes, Vol. 2, p. 107-120 [B44]

Phillips, H.R. & Brandt, M.P. (1995) Coal mine explosions – risk and remedy, Proceedings of SIMRAC Symposium, SAIMM, Johannesburg, pps. 15 [B45]

Phillips, H.R. (1996a) Improvement of health and safety in South African mines, Proceedings of the Conference on Health and Safety in Mining and Metallurgy, IMM, London, p. 185-195 [B46]

Phillips, H.R. (1996b) Identify methods to reduce the risk of explosions and fires caused by frictional ignition hazards, SIMRAC Report COL 226, p. 1-50 [B47]

Phillips, H.R. (1997a) Frictional ignition hazards associated with the Use of Continuous Miners, Proceedings of the 27th Conference of Safety in Mines Research Institutes, New Delhi, pps. 16 [B48]

Phillips, H.R. (1997b) Explosion and fire risk in South African coal mines, Proceedings SIMRAC Symposium, SAIMM, pps. 9

Phillips, H.R. (1997c) To establish the current status of research, development and operational experience of wet head cutting drums for the prevention of frictional ignitions, SIMRAC Report COL 426, p. 1-74 [B49]

Phillips, H.R., Kielbloch, A.J. & Rose, H.J.M. (1997) An integrated approach to post incident survival strategy, Proceedings of the 11th International Conference on Coal Research, Calgary, p. 167-178 [B50]

Phillips, H.R. (1999) The Coalbrook story: Disaster and forty years of remedial research, Proceedings of the 8th New Zealand Coal Conference, Wellington, p 77-89 [B51]

Phillips, H.R. & Belle, B.K. (2003) — Quantification of inherent respirable dust generation potential (IRDGP) of South African coals, SIMRAC, Research Report SIM 02-06-04, p. 1-60 [B52]

Phillips, H.R., Uludag, S. & Chabedi, C.K. (2010) — Prevention and control of spontaneous combustion – Best practice guidelines for surface coal mines, Coaltech Research Association, p. 1-62 [B53]

Phillips, H.R. (2015) — Lessons learnt from mine explosions, Australian Mine Ventilation Conference, AusIMM, p. 19-27 [B54]

Pomeroy, C.D. & Brown, J.H. (1968) — Laboratory investigation of cutting processes applied to coal winning machines, J. Strain Analysis, Vol. 3, No.2

Pomeroy, C.D. (1968) — Mining applications of the deep cut principle, The Mining Engineer, June, 1968

Potts, E.L.J. & Roxborough, F.F. (1961) — Further studies on the ploughability of coal seams, The Mining Engineer, January

Potts, E.L.J., Roxborough, F.F. & Whittaker, B.N. (1967) — Experiments with the automatic variable geometry coal plough, The Mining Engineer No. 80, May, p 539-564 [C26]

Protodyakonov, M.M. (Jr) & Teder, R.I. (1971) — Rational methods of planning experiments, Dynamic Rock Mechanics, Proceedings of 12th Symposium of Rock Mechanics, Society of Mining Engineers, New York, p 129

Rad, P.F. Olson, R.C. (1974) — Tunnelling machine research – interaction between disc cutter grooves in rock, U.S. Bureau of Mines, RI 7881, p. 1-21

Rad, P.F. (1975) — Bluntness and wear of rolling disc cutters, International Journal of Rock Mechanics and Mining Science, Vol. 12, p. 93-99

Rawlins, C.A. (1998) — Insulation of chilled water reticulation systems in underground mines, M.Sc. Dissertation, University of the Witwatersrand, October 1998, p. 1-210 [C27]

Rawlins, C.A. & Phillips, H.R. (2001a)	Mine cooling strategies and insulation of chilled water pipes, Proceedings of the 7th International Mine Ventilation Congress, Cracow, p. 371-379 [B55]
Rawlins, C.A. & Phillips, H.R. (2001b)	Reduction of mine heat loads, Proceedings of the 7th International Mine Ventilation Congress, Cracow, p.381-389 [B56]
Rawlins, C.A., Ramsden, R., Butterworth, M. & Hatting, R. (2001)	Chilled water pipe insulation specifications, Deepmine Project 6.5.5, CSIR-Minintek, Johannesburg
Rawlins, C.A. (2002)	Reduction of heat flow into mine excavations with special reference to haulage insulation and backfill, Ph.D. Thesis, University of the Witwatersrand, p. 1-511 [C28]
Rawlins, C.A., Phillips, H.R. & Jones, M.Q.A. (2002)	The use of backfill to control heat ingress in deep level mining, Mine Ventilation, Proceedings of the 9th US Mine Ventilation Symposium, Kingston, Balkema Publishers, p. 355-361 [B57]
Rawlins, C.A. & Phillips, H.R. (2005)	Underground mine ventilation planning and design with regards to heat loads and cooling mechanisms, Proceedings of the 8th International Mine Ventilation Congress, Brisbane, pps. 12 [B58]
Ren, T.X., Denby, B. & Herbert, M.J. (1996)	Report to ISCOR on study of Grootegeluk spontaneous combustion problems, The University of Nottingham, U.K., June 1996
Roxborough, F.F. & Rispin, A. (1972)	The mechanical cutting characteristics of the Lower Chalk, Department of Mining Engineering, University of Newcastle upon Tyne, p. 1-196
Roxborough, F.F. & Phillips, H.R. (1974)	Experimental studies on the excavation of rock using picks, Proceedings of the Third Congress of the International Society of Rock Engineers, Sept. 1974, Vol. IIB, p. 1407-1412 [B59]
Roxborough, F.F., & Phillips H.R. (1975a)	The mechanical properties and cutting characteristics of the Bunter Sandstone, Department of Mining Engineering, University of Newcastle upon Tyne, March 1975, p. 1-284 [B60]

Roxborough, F.F. & Phillips, H.R. (1975b) — Rock excavation by disc cutter, International Journal of Rock Mechanics and Mining Science, Vol. 12, p. 361-366 [B61]

Roxborough, F.F. King, P. & Pedroncelli, E.J, (1981) — Tests on the cutting performance of a continuous miner, Journal of the South African Institute of Mining and Metallurgy, Vol 81, No 1, p. 9-25 [C29]

Roxborough, F.F. & Phillips, H.R. (1981a) — The change in performance of disc cutters when stalled, Fourth Australian Tunnelling Conference, Melbourne, p. 361-368 [B62]

Roxborough, F.F. & Phillips, H.R. (1981b) — Applied rock and coal cutting, Workshop Course 156/81, Australian Minerals Foundation, p 5.15

Searra, J.A. (1990) — Gas related problems in the Witwatersrand Supergroup, M.Sc. Dissertation, University of the Witwatersrand, Johannesburg, p.1-149 [C30]

Singh, R.N., Demirbilek, S. & Turney, M. (1994) — Application of spontaneous combustion risk index to mine planning, safe storage and shipment of coal, Journal of Mines, Metals and Fuels,

Spaeth, A., Belle, B.K. & Phillips, H.R. (2017) — Introducing a new age of highly effective automatic explosion suppression barriers, 4th Australian Mine Ventilation Conference, Brisbane, p. 197-201 [B63]

Stripp, G.P. (1989) — Methane emission characteristics of South African coal seam strata, Ph.D Thesis, University of the Witwatersrand, Johannesburg, p. 1-530 [C31]

Task Group on Coal Mine Explosions (1997) — Guideline for the compilation of a mandatory code of practice for the prevention of coal dust explosions in underground mines, Department of Minerals and Energy, RSA, p. 1-60 [B64]

Takaoka, S., Hayamizu, H. & Misawa, S. (1973) — Studies on the cutting of rock by disc cutters, Tunnels and Tunnelling, Vol. 5, No. 2, p. 181

Uludag, S. (2001) — Assessing spontaneous combustion risk in South African coal mines using a GIS tool, M.Sc. Dissertation, University of the Witwatersrand, p. 1-154 [C32]

Uludag, S., Phillips, H.R. & Eroglu, N. (2001)	Assessing spontaneous risk in South African coal mines using GIS tools, 17th Mining Congress of Turkey, p. 243-249 [B65]
United States Government Accountability Office (2012)	Mine Safety: reports and key studies support the scientific conclusions underlying the proposed exposure limit for respirable coal mine dust, Report to Congress GAO-12-832R, p.1-24 [C33]
Van Dijk, R.L.S. (2000)	Onboard active suppression on subcut heading machines for South African conditions, M.Sc. dissertation, University of the Witwatersrand, p. 1-120 [C34]
Von Glehn, F.H. (1987)	A study of heat exchange in advancing stopes, M.Sc. Dissertation, University of the Witwatersrand, p. 1-200 [C35]
Wade, L., Phillips, H.R. & Gouws, M.J. (1987a)	Spontaneous combustion in South African coals, Underground Mining Methods and Technology, Elsevier Science Publishers, Amsterdam, p. 65-73 [B66]
Wade, L., Gouws, M.J. & Phillips, H.R. (1987b)	An apparatus to establish the spontaneous combustion propensity of coal, Proceedings of the Symposium on Safety in Coal Mining, CSIR Pretoria, Paper 7.1, p. 1- 16 [B65]
Wade, L. (1988)	The propensity of South African coals to spontaneously combust, Ph.D. Thesis, University of the Witwatersrand, June 1988, p. 1-282 [C36]
Wagner, H.	The challenge of deep-level mining in South Africa, Presidential Address, Journal of the South African Institute of Mining and Metallurgy, Vol. 86, No.9, p377-392
Whillier, A. & Ramsden, R. (1975)	Sources of heat in deep mines and the use of mine service water for cooling, Proceedings of the 1st International Mine Ventilation Congress, Johannesburg, p. 339-346
White, N. (2001)	Occupational lung disease, Chapter 5, Handbook of Occupational Health Practice in the South African Mining Industry, Safety in Mines Research Advisory Committee, p. 119-151

Whyte, R.J. & Grant, I.L. (1979)	A description of the penetrometer instrument and procedures for on- site collection of data related to coal seam cuttability, C.O.M. Research Report No. 51/79
Wiles, A.N. & Fairclough, F. (1978)	The prevention and prevalence of pneumoconiosis in New South Wales coal mining, Fifth International Conference on Pneumoconiosis, Caracas
Windridge, F.W., Parkin, R.J., Neilson, P.J., Roxborough, F.F. & Ellicott, C.W. (1995)	Report on an accident at Moura No 2 underground mine on Sunday, 7 August 1994, Warden's Inquiry, Government of Queensland, December 1995, p. 1-102 [C37]