CHAIRMAN'S LETTER

6 September 1994

Dear Delegate,

Thank you for registering to become a participant in the Underground Roadway Development Workshop. This workshop is the third in a series conducted under the auspices of the Australian Coal Association Research Program (ACARP) to establish Underground R & D requirements.

The objective of this workshop is:

"To improve the efficiency and reduce the risk of underground coal mining by the development of roadway driveage methods achieved through improving the awareness of current and emerging technologies and to determine the future directions and priorities for industry R & D expenditure."

The workshop has been segmented into technology sessions chaired by a specialist in each field. Relevant papers have been compiled to examine a wide range of issues in each session. Enclosed is:

» a revised program with a listing of papers and presenters. **Note that Day One first session now starts at 8:30am.** A keynote speaker (Mark Hart, Cyprus-Amax US) has been included in Session 3, and a breakfast speaker (Allan Rossiter, Peabody Resources).

» copies of the session chairmen's summary papers which cover the issues to be explored in their sessions.

To participate in the question times conducted at the end of each session will require reading this material prior to attending the workshop. A full set of papers will be issued to you when registering on Day One.

A technical display has been arranged with the cooperation and support of industry suppliers and will be conducted concurrently with the workshop. This will focus on the latest commercial technology and equipment relevant to Roadway Development.

I commend you to take on board the ideas presented and encourage you to participate in the discussions and surveys conducted during the sessions. This will ensure the recommendations generated for future R & D will reflect the industries' needs.

Yours sincerely

J. Simpson
Chairman
Underground Roadway Development Workshop

Australian Coal Association

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The role of the Inspectorate in gateroad development systems. **Brian Lyne**

Analysis of a gateroad development system. . . . . **Bob Butcher**

Alternative organisational aspects of gateroad development systems. . . . . **Malcolm Roberts**

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The application of radio imaging method to outburst hazard minimisation. . . . **Scott Thomson & Mark Neil**

The use of microseismic methods for detecting the outburst phenomenon. . . . . **Lawrence Leung**

Outburst remote control mining at Tahmoor Colliery. . . . . **Peter Wynne & Bob Case**

SETTING THE SCENE

The purpose of Session One "Setting the Scene" is to show the current situation in Australia relating to underground roadway development. To achieve this, a number of areas have needed consideration.

The first presentation is an "Overview of Australian Gateroad Development Trends" by Mick Gaynor (South Blackwater Coal Ltd). This presentation examines the current state of underground roadway development through comparisons of what longwall mines are achieving and looking at their development performance. I draw your attention to a related paper which will not be presented, "Australian Longwall Mines Survey in 1993 Roadway Development Performance" by Bruce Robertson (Shell Coal Division).

The second presentation will jointly be given by Brian Lyne (Qld Chief Mines Inspector) and Terry Abbott (NSW Senior Mines Inspector). This presentation discusses the role of the Qld and NSW Inspectorates with respect to current and future gateroad development systems.

The third presentation will be presented by Bob Butcher (Power Coal - Angus Place). Bob will present a paper titled "Analysis of a Gateroad Development System". This presentation breaks down the demands needing consideration in formulating a successful development system for whatever mine environment is present.

The fourth presentation titled "Organisational Aspects of Gateroad Development Systems" will given by Malcolm Roberts. This presentation will discuss the integrated approach method for setting up a gateroad development system.

The fifth presentation will be given by Danny Gillespie from AMIRA. Danny will review the state of ACARP funding on gateroad development systems and will examine where funding has previously been directed and the current view of ACARP funding towards gateroad development systems.

The final presentation will be given by Rudolf Werner from The Fourth Wave Pty Ltd. Rudolf will discuss R & D taxation incentives for promoting R & D in mining. This area is appropriate to mine operators as it promotes tax effective R & D in gateroad development systems.

Through the six presentations given in Session One, workshop attendees will become familiar with the current performance of gateroad development in longwall mines, the inspectorate view of development systems, current methods of analysing gateroad development systems, the state of ACARP funding and how R & D in this area can be carried out with taxation incentives.

MICK GAYNOR
South Blackwater Coal Limited

Mick Gaynor

South Blackwater Coal Limited

OVERVIEW OF AUSTRALIAN GATEROAD DEVELOPMENT TRENDS

The demands for gateroad development appear to be quite simple when thought about in the "macro" sense. Mines with high longwall extraction tonnages will require rapid development of gateroads to facilitate their longwall panel requirements. It therefore stands to reason that current Australian longwall mines that are highly productive will have higher development tonnages and gateroad advance rates than mines that have less productive longwall panels.

This initial statement may appear to be reasonable but it is, in fact, not true when the 1993 production statistics for longwall mines are examined.

Table 1 shows the 1993 production statistics for Australian longwall mines. Figure 1 graphs the development, longwall and total tonnes for these mines.

Figure 1 shows with remarkable clarity just how similar the development tonnages were for longwall mines in 1993. The real variation between mines in production tonnes was in the actual production of longwall coal.

It is not intended in this workshop to examine the issues of longwall panel extraction, although I have no doubt that a fair bit of discussion on these matters will take place informally. The workshop is designed to focus on the issues affecting gateroad development, and as demonstrated by the 1993 development production statistics, these issues are relevant to all mines.

A 1993 SNAPSHOT OF GATEROAD DEVELOPMENT

Included with the papers presented for "Session One — Setting the Scene" is a paper by Bruce Robertson from Shell Coal Division titled "Australia Longwall Mines Survey — 1993 Roadway Development Performance".

This paper summarises the results of a survey carried out by Bruce into roadway development at Australian longwall mines and provides a wealth of information about the current state of roadway development.

The main points to be seen from Bruce's paper are as follows:

- Most mines develop over 200m/week of roadways at widths of between 4.5m to 5.5m and at a working height of between 2.4m to 3.2m.
- Most mines operate between 5 and 9 unit shifts on development per day.
- Most mines install rib support. Roof bolting is generally of an intensity of above 4 bolts/m of roadway with two mines installing up to 10 bolts/m of roadway.
- The most popular gateroad pillar length is between 100 to 124m.
- Panel extensions take between 1.5 and 5 shifts with the average being 3 shifts.
- About half the respondent mines develop an average more than 10m/shift, the other half develop less than 10m/shift on average.
- Most mines develop roadways with the conventional development system, this being the system using shuttle cars, a continuous miner and a conveyor belt.

GATEROAD DEVELOPMENT SYSTEMS

A general representation of a gateroad development system in Australia is shown on Table 2. The system has as its "base case" the traditional mining system that evolved many years ago, this being a continuous miner feeding into a shuttle car which off-loads to a conveyor belt. This system is tried, tested and true. For those mines lucky enough to be blessed with good ground conditions, this system is good enough and in some cases excellent.

For those mines with mining problems, the solution has been to modify the "base case" and/or add further tasks to the "base case" so that the problem areas are addressed. Nearly every modification or task addition results, however, in a reduction in the reliability and productivity of the "base case". As all operators know, the greater the number of links in a chain, the greater is the risk of having a weak link. As more and more modifications are made

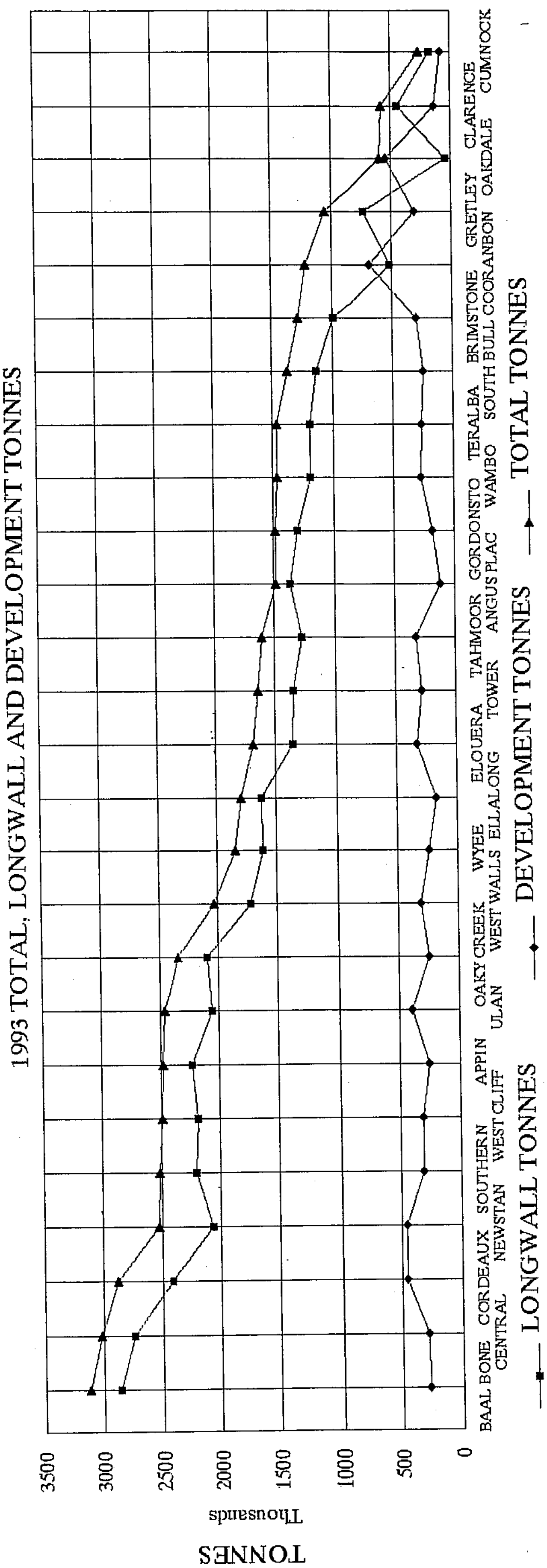


Figure 1. Comparison of Australian longwall mines.

Table 1. 1993 production at Australian longwall mines.
(TAKEN FROM MARCH 94 AUSTRALIAN MINING MONTHLY)

MINE	LONGWALL FACE WIDTH	LONGWALL PANEL LENGTH	SEAM THICKNESS	LONGWALL TONNES	DEVELOPMENT TONNES	TOTAL TONNES	RATIO OF LONGWALL TO DEVELOPMENT TONNES
BAAL BONE	200m	1530 to 2060m	2.1 to 2.4m	2854155	270970	3125125	10.53
CENTRAL	200 to 205m	1200 to 1600m	1.85m	2735194	282383	3017577	9.69
CORDEAUX	200m	1400 to 1920m	2.2m	2416344	458356	2874700	5.27
NEWSTAN	221m	2340m	2.7 to 3.3m	2072338	452992	2525330	4.57
SOUTHERN	200m	1920 to 2750m	1.7m to 2.8m	2205286	306865	2512151	7.19
WEST CLIFF	200m	1500 to 2900m	2.1 to 2.7m	2183554	310294	2493848	7.04
APPIN	200m	2000m +	2.4 to 2.7m	2229508	251909	2481417	8.85
ULAN	250m	1750 to 1795m	2.9m	2061827	395524	2457351	5.21
OAKY CREEK	198 to 202m	1710 to 1750m	3.2 to 3.5m	2092838	246290	2339128	8.50
WEST WALLSEND	185 to 200m	1870 to 2100m	2.3m	1718126	309868	2027994	5.54
WYEE	205 to 234m	820 to 1700m	3.6 to 4.2m	1608465	242028	1850493	6.65
ELLALONG	200 to 203m	1080 to 2000m	3 to 3.3m	1624290	171867	1796157	9.45
ELOUERA	149m	2760m	3.2 to 3.5m	1354068	332790	1686858	4.07
TOWER	150m	1000 to 1100m	2.1 to 3.5m	1349836	287290	1637126	4.70
TAHMOOR	225m	550 to 1110m	1.9 to 2.3m	1271887	331710	1603597	3.83
ANGUS PLACE	223m	1670m	Up to 5m	1363092	120000	1483092	11.36
GORDONSTONE	200m	1308 to 1467m	3m	1302263	180000	1482263	7.23
WAMBO	195 to 203m	1500 to 3300m	2.6 to 2.8m	1194046	270884	1464930	4.41
TERALBA	141 to 200m	1850 to 1950m	2.6 to 3.3m	1189669	270000	1459669	4.41
SOUTH BULLI	105m	845 to 1992m	1.9 to 2.7m	1132823	244564	1377387	4.63
BRIMSTONE	125 to 131m	1150 to 1825m	2.2m	985841	298211	1284052	3.31
COORANBONG	130m	526 to 800m	2 to 3.2m	526014	691585	1217599	0.76
GRETLEY	30 to 52m	350 to 870m	2.8 to 3m	740905	312264	1053169	2.37
OAKDALE	150m	1058m	2.1m	47352	552648	600000	0.09
CLARENCE	171m	1220m	3.8 to 4m	448307	135374	583681	3.31
CUMNOCK	129m	665m	3.3m	178988	82620	261608	2.17

Table 2. Gateroad development systems.

BASE CASE
1 X Continuous Miner
1 or 2 X Shuttle Cars
Panel Conveyor Belt
Hand Held or Miner Mounted Bolting Rigs

MINING PROBLEM	EFFECT OF MINING PROBLEM	BASE CASE MODIFICATION
High in-seam gas levels	Outbursts or unacceptable gas levels	Slow mining rate. Set up "outburst" panels with modified face equipment. Restrict entry of personnel to face. Pre-drain mining panels.
Seam water	Roadways are boggy or under water	Install face pumps and sumps. Carry out u/g road construction. Install extensive pumping systems.
Weak roof/rib	Roof and/or ribs deteriorate and/or fail	Install heavier densities of roof/rib support. Install secondary roof support systems. Reduce roadway width/height.
High ground stress	Roof and/or ribs deteriorate and/or fail	Install heavier densities of roof/rib support. Install secondary roof support systems. Reduce roadway width/height.

to the "base case" to compensate for the mining problems shown on Table 2, the lower is the productivity of the gateroad development system.

Changes to the conventional system have not been dramatic. As commented by Bruce Robertson in his survey:

> "Continuous mining equipment selections are diverse, but shuttle cars are the overwhelming transport choice. New generation cutter/bolter miners and fast cycle continuos miners have gained a foothold, particularly in mines with arduous conditions. Anecdotal comments returned from the survey attribute significant performance improvements to the introduction of these machines. Mobile boot end (MBE) devices have been enthusiastically received. although operators have initially preferred to confine their use of static applications. Several operators have been prepared to invest in 'double header' or 'superpanel' configurations to increase daily panel advance."

CONCLUSIONS

There are a variety of roadway development systems being used in Australian longwall mines and most of these systems operate the "base case" system of a continuous miner, shuttle cars and panel conveyor belt. It is really not that surprising, therefore, to see that the development tonnages achieved in 1993 for all longwall mines are remarkably similar.

The way in which development systems are being organised and the machinery being used is changing and advances in development rates are evident.

There are a number of components involved with any development system and a number of these have been surveyed by Bruce Robertson and presented in his accompanying paper.

It would be to the advantage of the underground coal mining fraternity if surveys such as this could be carried out on a regular basis.

BRIAN LYNE

Department of Minerals and Energy, Queensland

ROLE OF INSPECTORATE IN GATEROAD DEVELOPMENT SYSTEMS

The traditional approach by the Australian coal mining industry has been to seek 'approval' of the inspectorate (Government) to undertake activities outside of the normal.This has been evident in the introduction of new methods of GATEROAD DEVELOPMENT using new equipment or extending over longer distances. It is probably true to say that this has been fostered by the fact that employees have grown to mistrust the motives and competence of coal owners in appearing to rush new concepts and systems into place before all relevant matters, particularly those related to their safety, have been identified and addressed. Out of this, Governments have had thrust upon them an 'Honest John' mediator role to give various proposals some form of logic and address safety issues.

Such a concept has been in use in Australian coal mines for too long in fact some argue that inspectors now see it as their divine right to approve mining systems before activities commence.

Many mine operators seek an 'approval' from inspectors to help enforce the introduction of a new system of operating. One suspects that part of the motive is that if the process is proven to be wrong or unsafe, the responsibility is shared with the inspectorate. This is often used to diffuse employee arguments, particularly of an industrial nature.

It is a sad fact that even today employees rely upon the Government's involvement in order to have a say in the final outcomes. Consultation with employees is certainly improving however there is still considerable room for improvement.

Is the current role of government the best solution?

I would like to suggest NO, and in fact go so far as to say it is a second-rate solution.

To me it would be more logical for the Government's role to only set or help set legislative requirements and Australian Standards, for the base outcomes. These may include:

- Quality of atmospheric standard to be always available to employees;
- Quality standards for the machinery to be used in a particular type of hazardous area.
- Systems to ensure that an assessment of risks anticipated in gateroad is made such as:
  - gas make/ventilation
  - geological/geotechnical non-conformities
  - roadway support systems
  - coal transport
  - men/materials transport
  - fire
  - entrapment
  - communication
  - emergency support

followed by a properly documented system developed between the mine operators and employees, clearly defining the methods to be used to control or eliminate the identified risks.

In essence, what is being called for is a Quality System developed between the mine operator and employees:

- The risks are determined;
- Controls are put in place;
- The process is documented in a quality system methodology;
- Persons are trained in the system.
- The system is audited internally (by deputies, undermanager, managers);
- Faults are formally recorded and acted upon promptly (corrective action requests);
- Overall system audited (externally) by inspector.

The "system" identifies any action required to be taken and requires an organised "teamwork" approach.

This is in essence the direction that the new Queensland Coal Mining Bill has been developed.

CURRENT DEFICIENCIES

Contrary to what some people might say or think, inspectors are not "all seeing and all knowing". Governments might issue the leases but they do not know all the features (risks) they contain.

Mine operators have often been slow to:

- Evaluate geotechnical properties of the strata adequately before planning begins. Unfortunately it is often after development has commenced that geotechnical studies are attempted (if at all),
- Remove high gas make risks before mining (ie.pre drain the area),
- Introduce dust extraction systems on the continuous miner apparently preferring to apply a little extra stonedust in the mine, and
- Adopt lateral thinking in the emergency planning.

e.g. Why not introduce refuge chambers in long gateroad developments instead of requiring a limitation of numbers of people. In other words, managing the risk by a different process to that traditionally used.

Other countries have been using onboard exhaust fans on mining machines and underground refuge chambers for years.

SUMMARY

What is the role of the Government?

Is it to set the objectives (outcomes) to be achieved and fill in for the competency gaps of operators?

Competent operators with excellent safety and production records do much more than the mining requirements and wait for the Government to lead. They are prepared, they care about their employees,they involve their employees in decisions affecting their workplace, they achieve and need the least Government involvement.

Maybe the answer to the question is that the governments role is " to set the minimum standard of workmanship for the worst operator in the industry".

Let us bring forward the day when this is not the case.

BOB BUTCHER

Powercoal

ANALYSIS OF A GATE ROAD DEVELOPMENT SYSTEM

Most recent development strategies have failed to reproduce gains detailed in capital expenditure justifications.

Our mining industry is regarded as being very innovative on an international basis. AND, here we all are again, assembled in a room, having another go at getting it right!

Why is this the case?

Is the answer a complex array of high-tech integrated processes, or is it simply right under our noses?

WHERE TO START LOOKING

The traditional approach has been to jump in a car or on a plane in search of the 'holy' continuous miner. We reported back to our capital justifications the biggest and flashiest one, manipulated justification statistics to win board approval than proceeded to buy and introduce a system that did not realise the gains promised.

I will offer my view on where to start.

Two (2) criteria must be established, namely:-

(i) Key goals the development system must strive to achieve

(ii) Constraints the development system must integrate with

At this stage it is inappropriate to consider available technology.

ESTABLISHMENT OF KEY GOALS

These will normally come from a longer term corporate strategy. This strategy will find definition from:-

- Current business plan
- Next contract evaluation
- Corporate safety statement
- Corporate environmental statement
- Product quality requirements
- Reserves
- Budget tonnes schedule
- Revenue schedule
- Mining constraints
- Mine lease geometry
- Output per man year

Key goals should be firm and defined and consist of process expectations:-

- Number of development units
- Shifts per week
- Advance in metres per week
- Proportion of work face available for development
- Lost time injury frequency rate
- Total workforce

From this analysis the number of people,e, the number of machines, the amount of time, the rate and the safety at which development work must be performed is established.

It is still inappropriate to consider available technology.

ESTABLISHMENT OF CONSTRAINTS

The establishment of development system constraints is the next design area to investigated. The following global issues should be considered:-

- Mining conditions - geological
- Mining conditions - geotechnical
- Seam Environment (*susceptibility to spontaneous combustion and outburst, make of gas and ground water etc.*).

Further constraints will be determined by ascertaining the needs of site specific processes. These include:-

- Coal clearance system
- Size of longwall equipment
- Ventilation system
- Energy reticulation system

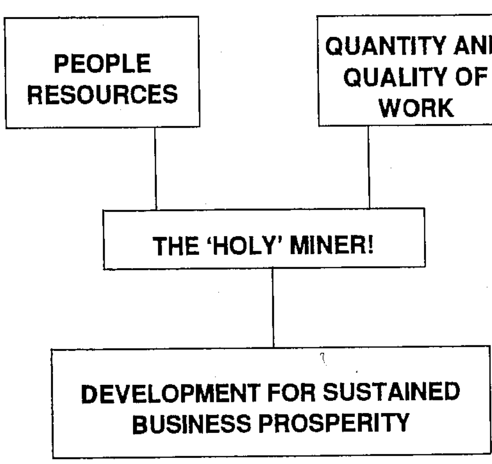
- Waste disposal (*pumping, dust etc.*)

Constraints will appear as:-

- Width of roadway
- Height of roadway
- Number of roadways
- Primary roof and rib support needs
- Secondary roof and rib support needs
- Cut out distances before primary support
- Weight and ground bearing pressure of machinery
- Size of machinery
- Materials handling needs
- Dust suppression needs
- Gas dilution needs
- Cut out distance before secondary support
- Development risks (*outbursts etc.*)
- Direction of mining

This analysis will yield the quantity, quality, timeliness and risks of doing development work.

Thus, to date, we have established the quantity of people resources and workload:-



CHANGE ANALYSIS

This analysis investigates the difference between the current development system practice and the practice necessary to achieve defined key goals.

It will highlight the critical process improvement areas where most gains can be achieved. It is still inappropriate to consider available technology at this stage.

Broad process requirements will be defined:-

- Simultaneous cutting/bolting
- Materials handling
- Primary support requirements
- Secondary support requirements
- Ventilation requirements
- Coal clearance requirements
- Service extension requirements
- Personnel workload and skills
- Roadway dimensions
- Cut out distances
- Roadway configuration (*Pillar Size*)

IN SEARCH OF THE 'HOLY' MINER

At this stage we can begin the quest for appropriate technology. It will be done in conjunction with two other key process parameters:-

- *People expectations*
- *Process engineering*

Various combinations of technology and process engineering (% of process steps performed in parallel) can be modelled.

Each combination should yield expectations for:-

- Return on Investment
- Safety
- Productivity

Several tools and skills can be utilised to assist in the selection of appropriate technology and process design:-

- Best practice and benchmarking
- Process re-engineering
- Cost benefit analysis
- Risk Analysis
- Equipment evaluation

REALISING POTENTIAL GAINS

This stage of implementation is the most difficult and most prone to failure. Many issues must be addressed and incorporated in the working system:-

- Work place reform and organisational development
- Work team involvement (*empowerment*)

- Skills and training needs
- Total productive maintenance
- Machine effectiveness
- Career Path Development (*work models*)
- Work place culture development (*attitude, satisfaction, involvement and organisational loyalty*)
- Risk Assessment
- Organisational commitment
- Organisational communication
- Work place change management
- Waste effectiveness
- Change our time management
- Continuous incremental improvement
- Process performance measurement
- Planning and scheduling

Each step in the process should have defined best practice goals. It is the gains achieved from individual steps that constitute greater overall process performance. Improvement strategies for each process step should be clearly defined and scheduled.

Monitoring should drive milestone objectives and seek out poor performing areas.

CONCLUSION

The design of a development system for a mine is complex. Considerations range from global issues to detailed individual process steps. Clearly defined strategies need to be defined for process and equipment selection, together with management systems to ensure predicted process gains are realised.

MALCOLM ROBERTS

Catalyst for Corporate Performance

ALTERNATIVE ORGANISATIONAL METHODS FOR GATEROAD DEVELOPMENT SYSTEMS

High performance and competitive advantage for lasting business success require:

- organisation structures and systems based on the business/work process
- the use of real knowledge instead of traditional management reporting systems
- committed, skilled people using real knowledge to kill waste and raise productivity
- an integrated approach.

INTRODUCTION

Australia's mining industry has a proud history of technical innovation. Our industry is a leader internationally in the application of technology.

Unfortunately while our industry has been very successful technically, performance in underground coal mining has lagged behind benchmarks set in North America and South Africa. Why is this? Why is our industry an international laggard - especially when we have many technically competent managers at all levels in underground coal mining?

High performance requires more than selecting the world's best equipment and shoving it down two holes in the ground.

The presentation will consider the effectiveness with which resources are used by managers at all levels. It will provide simple concepts that are essential for efficiently managing all business resources and ensuring our industry is internationally competitive and secure.

The presentation will address three levels of organisation - panel, mine and industry.

INTEGRATED APPROACH

This section will cover the need to optimise our overall mining process. It will lead into the main points of the presentation:

1. basing organisation structures on the work process,
2. ensuring true understanding of work processes with real knowledge
3. involving people
4. improving business performance by killing all forms of waste in the work process.

A SOLID FOUNDATION - THE WORK PROCESS

A quick discussion of leading international concepts in management will be followed by the key point that organisational systems need to be based on the work process.

Management's role will be discussed using the example of Wade Kathage's leadership as an Area Leader at Gordonstone mine to illustrate the importance of these simple yet powerful concepts. Wade led the improvement of Gordonstone's development performance to the point where in 1993 the mine's development rates were up to 100% better than the next best Australian performance under similar high density roof support.

REAL KNOWLEDGE AND PROPER UNDERSTANDING

Traditional management is limited by the fact that it does not use real knowledge. Instead decisions and plans are based only on data about the level of performance. ie., tonnes or average tonnes.

This is totally inadequate in the real world which is affected by variation in all areas. Real knowledge requires an understanding of both the level of performance AND the variation of performance.

Although time will not permit discussion of the concept of natural variation, a framework will be provided to show how variation can be an ally rather an enemy. This provides real understanding of work processes.

Real knowledge and understanding is vital for killing waste. The elimination of waste makes it possible to significantly improve business performance and minimise operating and capital costs.

Organisational considerations need to include systems to ensure people can understand their work processes.

PEOPLE USING REAL KNOWLEDGE TO KILL WASTE

Even organisations based on the work process and using real knowledge of their processes need a third ingredient - committed and skilled people.

To maintain competitiveness an organisation needs to "continually improve". This means more than simply repeating this tired phrase which is quickly becoming another piece of meaningless jargon. True continuous improvement means more than seeking gradual improvement. It means concepts that enable and support performance leaps - again based on real knowledge and understanding.

Organisation culture is another major determinant of development performance and will be discussed briefly.

KILLING WASTE LEADS DIRECTLY TO PRODUCTIVITY IMPROVEMENTS

When waste in all forms is removed, more can be achieved from the same fixed amount of resources. This is the heart of improving business performance.

Points will be explained drawing on experience and on recent extensive international and Australian research.

CONCLUSION

The presentation will conclude by stressing the importance of integrating all aspects to ensure high performance via alternative organisational methods for gateroad systems.

Malcolm Roberts

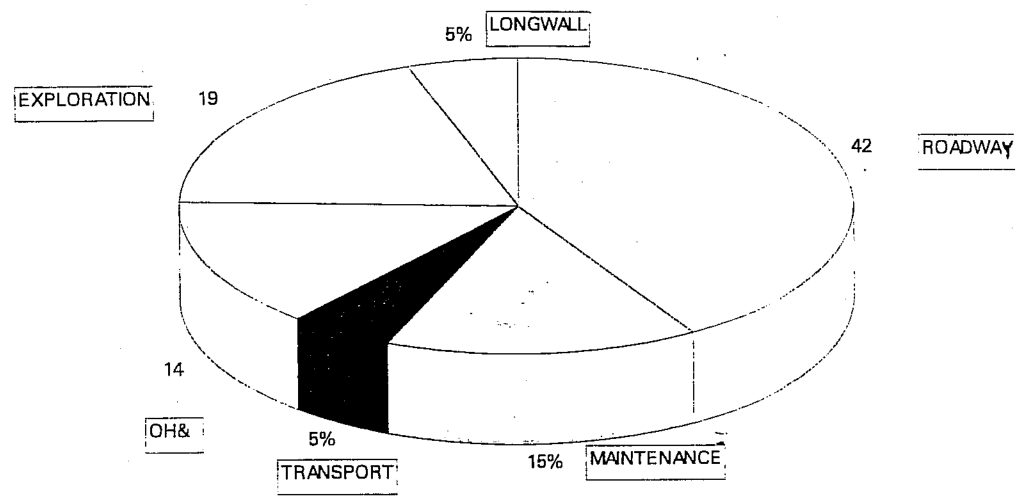


Figure 4. 1993 Underground allocation by priority.

Seminar was held in Sydney in October 1994 to examine what R&D was being conducted and where areas of research interest were.

Examination of NERDCC/ACARP records of project funding from 1984-1993, as shown in Figure 5, clearly shows that greatest research expenditure over this period has been put into Ground Support and Cutting Equipment.

Even with this research expenditure recently developed mines such as Gordonstone, North Goonyella and Springvale still confront problems in the area of ground support. We are still a long way from fully understanding and applying technology in this area. With underground mines facing the prospect of mining at greater depths, funding focus on ground support issues is a strategic necessity. Development production capacity requirements of 260t shift to maintain 3mtpa production capacity (Galvin 1994) are far less than the 341t shift being achieved in 1963 (Grant 1984). It is hard to justify the late 1980's focus on developing several cutting machines.

Changes in focus of finding for 1984-1993 are shown in Figure 6. This figure reveals that ground support issues have always been the highest priority and the importance of ground support funding relative to other issues is increasing. It now commands 45%of ACARP Roadway funding.

The priority of Cutting Equipment has dropped significantly in the last 3 years either as a belief that the 'ideal cutting machine' now exists or that this is the wrong issue to pursue. I suspect it may be a combination of both. The emerging priorities of the last three years are Emission Control and Systems.

There is a greater appreciation that improvements in individual technologies will not deliver the required roadway development rates. The technologies we need to achieve suitable development rates essentially exist. The challenge is to design systems which best package people and technology into a safer and more continuous mining process.

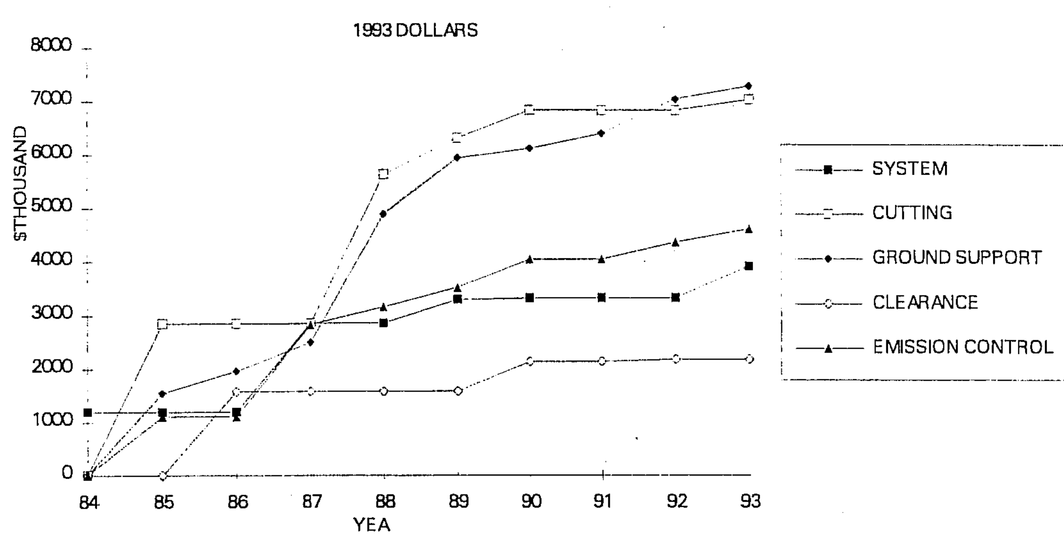


Figure 5. Roadway development cummulative expenditure.

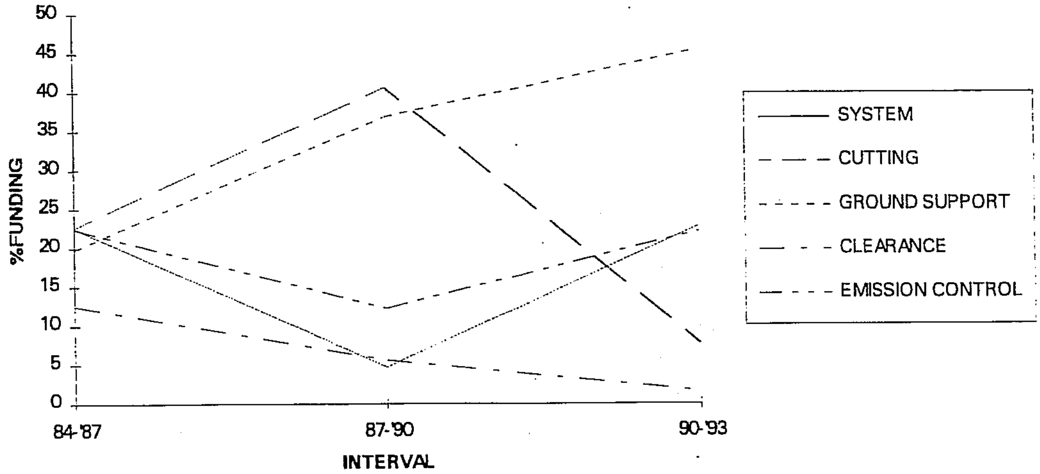


Figure 6. Roadway development funding priority.

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Megan Bartlett

The Fourth Wave (Australia) Pty Ltd

RESEARCH & DEVELOPMENT – TAXATION INCENTIVES FOR PROMOTING R&D IN MINING

The 150% Research and Development (R&D) Tax Incentive was introduced as part of the Federal Government's package to improve the climate for R&D in industry and to develop internationally competitive, export-oriented and innovative industries in Australia by:

- increasing companies' investment in R&D;
- encouraging better use of Australia's existing research structure;
- improving conditions by the commercialisation of new process and product technologies developed by Australian companies; and
- developing a capacity for adoption of foreign technology.

With the scheme having significant after tax benefits to companies, it is quite startling to find that there are only 2500 companies who annually register to take advantage of the tax concession. The 150% tax concession can have a dramatic effect on a company's bottom line after tax so why is it that R&D is often an ignored area of most companies' activities?

One of the major reasons we think that R&D has not been fully taken advantage of is the limited definition and understanding of what actually constitutes R&D. This is a good starting point for our discussion.

To illustrate and simplify what is a very stiff and concise definition contained in Section 73B of the Income Taxation Assessment Act, we use a generic industry example to explain:

"THE TROUBLE WITH HARRY"

(WE DON'T DO RESEARCH AND DEVELOPMENT???)

Harry has been in the metal industry for 35 years.

Harry is a senior production engineer.

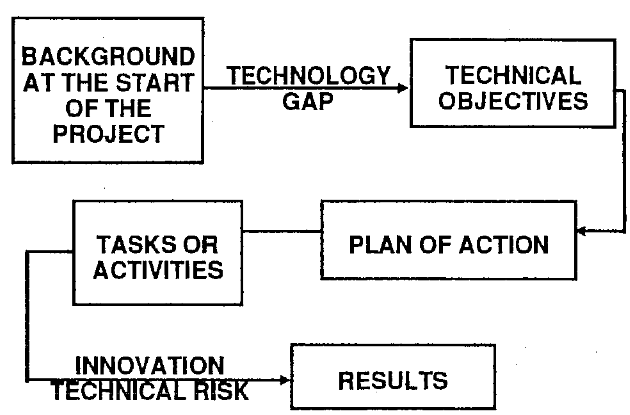
He has a vast body of knowledge on anything to do with metal and the making of saucepans. Recently a customer asked Harry to make a saucepan with a large top flange. Harry tried every way known to him to bend, roll and stretch the material to make this large flange. Unfortunately nothing that he tried was successful.

When I met Harry he was tearing his hair out in frustration. As we were discussing his work, he put it to me that he had this little project that had really stretched his mind but that he did not see it as Research & Development because it was **something that he has to do as part of his job**.

Quite clearly Harry did not know where to go. He had got to the point of making enquiries whether there was another way that he could achieve his goal. The truth is that Harry had a very clearly established technical objective. He was outside his technical comfort zone and in attempting to overcome the problems, he was being systematic in his tasks and activities.

As he had reached the stretch modulus of the material, as known to Harry, it was clear that there was plenty of technical risk in any further attempts he was making to achieve his result. I can still remember the pause and the big grin on Harry's face as he said "I guess I'm doing Research and Development after all."

It is clear to me that Harry's project would fit the following loop.



Many technical people are reluctant to admit that they have technical problems or tend to downplay the existence of a technical challenge on the pretext that the activities are what they usually do or that the problem will go away with a little more thought or experimentation. The idea that such processes are the actual doing phase of R&D is simply not entertained. We suggest that you spend some time analysing your activities and see whether they fit the above loop.

Formally, R&D activities are defined as being systematic, investigative or experimental activities involving innovation or technical risk that are carried on for the purpose of acquiring new knowledge or creating new or improved materials, products, devices, processes or services as well as directly related activities.

As well as activities which test a core theory you may claim associated activities. There are no rules which limit the associated activities. For example, in developing new software, one core activity would be the development of algorithms and the structure of the software programme. The associated activities could be the writing of a database to allow the algorithms to be tested. The writing of such a database could take a lot more time than the actual writing of the core algorithms.

An engineering example could be the build of a prototype product. This prototype would be a core activity. The testing of the prototype would be an associated activity. It may well be that the testing required is much greater than the building of the actual prototype.

To take full advantage of the benefits offered by the Federal Government's R&D programme a structured plan and system is required by your company. Recording for the R&D tax concession should be a part of daily operations just as are quality assurance, standards compliance or account reporting. The Fourth Wave has developed systems which allows you to plug R&D into your existing systems and practical training packages to assist you in putting together your own claim. Ring Gael Connop on 1800 63 4242 for information.

The 150% tax concession has a direct impact on a company's bottomline. The threshold is now only $20,000 per annum per company. Every R&D dollar not recorded costs you 16.5 cents. You are doing the work anyway. Are you willing to let this slip by for another year?

ARE YOU DOING R&D AND NOT REALISING IT?

Recently one of our mining clients asked us to look at a project in their washery. Their project was defined as a new piece of equipment which could improve efficiency in the washery. Costs had been incurred in thinking through the concept, drawing, drafting and manufacturing of a prototype. The manufacturer of the prototype was seen as the end of the project. In reality there was need to change some equipment in the washery to install the prototype. There was also need to pass product through the prototype to assess its viability and to test the theory as to its efficiency. The product passed through the prototype had to be monitored for quality.

In the end it was decided that five different products would need to pass through this prototype equipment. There were costs associated with mining these products, crushing, sorting, stacking, processing through the washery, monitoring and possibly resorting after processing. It was decided that a 24 hour run on each of these products would give sufficient information to prove the theory underlying the prototype. Obviously the original project had grown quite considerably in size from a mere $20 000 to a sum exceeding $400 000. At the same time the client was realistic in the length of testing required and the amount of product claimed. This example illustrates that by looking at the project broadly and seeing whether all aspects have been covered, this can increase quite significantly the level of your R&D claim.

Another more pertinent example to the underground mining context would be installing a new long wall system. If we took the installation of the long wall system in isolation we would have difficulties in actually constructing an appropriate R&D project. We would have to cross the initial hurdle of saying why this installation was going to represent a particular challenge. If, however, we took a look at the bigger picture our project probably started some time beforehand in planning, designing, developing long wall roads. Pillar design may include roof bolt support, some geotechnic investigation, looking at the subsidence effects and looking at the consequences of installing a long wall system in the particular geological conditions of the mine. If, for example, we are mining under particularly unstable roof conditions, the installation of the long wall system may be particularly fraught with difficulties. Instead of our project becoming the installation of a long wall system, our project may be the development of a new process of mining using a long wall mining technology in unstable roof conditions. In this way we can construct a legitimate and supportable R&D project.

If our project has been structured properly and thought about up front there are some distinct costing advantages for companies. For example, if our project was to test whether the long wall system was to mine at a certain rate and work effectively under the unstable roof conditions, then it may be that we can pick up some of the capital costs of the long wall mining system so long as we can demonstrate that it is being used in a trial capacity. There are a number of timing issues that then need to be considered. Documenting time becomes a critical factor as well.

There is certainly a lot of scope for looking at the R&D implications for any large capital projects that are being undertaken underground. Another interesting area where mining companies can take advantage of the tax concession is looking at the linkage between on-surface R&D projects and underground costs. For example, with our previous example on the washery, there is a cost of the coal produced. If we think of the project up front and can isolate that we need x tons of coal from a variety of sources then we are able to pick this cost up as part of our R&D project being conducted on surface. Although it may be routine for the underground people it forms part of the R&D project on top and therefore the costs can be eligible.

WHAT ARE THE BENEFITS OF THE TAX CONCESSION?

Given a corporate tax rate of 33 cents in the dollar, every $100 000 spent on R&D provides an after tax benefit of $16 500. Have you ever wondered what it takes to earn an after tax profit of $16 500 from sales revenue? Suppose company X develops a new roof bolting rig and process to secure roof bolts in unstable ground conditions and in doing so spends $100 000 in salaries, prototypes, labour, testing and design. This development is claimed as routine business expenditure. **If** company X recorded and submitted this $100 000 as expenditure then company X would have an extra tax deduction of $50 000. This results in an after tax benefit of $16 500 ($50 000 x 33% at corporate tax rate). This figure could equate to quite a significant number of tonnes of coal revenue.

This simple illustration makes it reasonably clear that companies can really take advantage of this Government scheme and claim a generous business deduction for **work that is already being carried out**. By relating R&D to profit margins and after tax savings a company's mindset can change from considering R&D recording as a chore to accepting R&D recording as the precursor to a genuine benefit.

Our experience suggests that the following activities need to be undertaken in each company to fully benefit from the tax concession:

- Educate and motivate the production, technical and financial personnel on what is R&D and the types of expenditure that can be claimed. This ensures that R&D is being captured in the traditional and non-traditional R&D areas.
- Install an appropriate and simple project identification and cost recording system that is compatible with the company's existing systems. Simple changes to the cost recording system should facilitate this.
- Factor R&D performance into the company's strategic plans and where possible credit the technical units with the R&D benefits. We find this is one way of ensuring maximum compliance when it is performance oriented.

The 150% tax concession has a direct impact on a company's bottom line. Every R&D dollar not recorded costs you 16.5 cents. You are doing the work anyway. Are you willing to let the benefit slip by for another year?

(Written by Megan Bartlett. Megan Bartlett is a Consultant with the Government Incentive & Technology Consultancy, The Fourth Wave (Australia) Pty Ltd. This paper is being presented by Rudolf Werner, who is the principal consultant with The Fourth Wave).

Rudolf Werner.

Bruce Robertson

Shell Australia

AUSTRALIAN LONGWALL MINES SURVEY 1993 ROADWAY DEVELOPMENT PERFORMANCE

A survey was conducted of 1993 roadway development in Australian longwall mines to build up a picture of the range of operating conditions, system configurations and performance parameters in operation. All 27 mines responded by completing the survey form (see example, Figure 1). It is anticipated that the information gathered will be of value to researchers and industry as a reference base for the development of technology to improve driveage productivity.

SURVEY FORMAT

The format of the survey form was designed to capture the following types of information:

- general development performance over the year,
- prevailing mining conditions,
- operational configuration and intensity,
- analysis of performance and breakdowns.

Individual collieries have not been identified and results are presented in random order. There were variations in the quality of data received and some operators did not routinely collect some of the more detailed data requested. Where ranges of values were quoted, a mean value has been used in the database.

An analysis of the results of the survey follows.

INTERPRETATION OF RESULTS

General Performance

Weekly roadway driveage is shown in Figure 2, segmented into gateroad and other driveage (recorded as 100% gateroad driveage where total undifferentiated). A 50 week year has been assumed to normalise some operations that did not operate for the full year. Most mines are required to develop 150-250m/wk in gateroad and an additional 50% on average in non-gateroad headings.

It is estimated that a total of about 370km of headings were developed (67% in gateroads) during 1993. Development intensity to achieve this performance is illustrated in Figure 3.

Shift productivity is illustrated in Figure 4, calculated from total metres developed and productive shifts reported for 1993. Values are determined for gateroads and total mine where possible. About half of the mines are achieving 10m/shift or more and it appears that performance in gateroads is usually, marginally better than in other headings.

Although planned shifts vs productive shifts were surveyed, the response was too inconsistent for meaningful comparisons.

Mining Conditions

To simplify categorisation of roof/floor conditions, a broad classification was invited (weak, moderate, strong). As shown in Figure 5, most operators appear to consider the strengths of roof and floor "moderate" at least. Of course there are other factors impacting on mining conditions which affect performance (stress, gas, water etc.). Support requirements perhaps reflect a more definitive picture of mining conditions. Figure 6 combines roof and rib support intensity. It is clear that a wide variety of support density is in evidence with from 2-10 (average) roof bolts per metre and 0-6 rib bolts per metre. Straps are the most common form of ancillary bolting control although mesh is also in wide use.

Roadway dimensions are profiled in Figure 7. It is observed that, with one or two exceptions, roadway heights fall between 2.2 and 3.2m, with widths ranging from 4.5 to 5.5m.

Cable bolting is widespread, especially for installation, take off and cutthrough zones. Flexibolts are emerging as a versatile intermediate remedy.

System Configuration

Gateroad pillar length variations are indicated in Figure 8. Whilst most operators have adopted a 100m pillar length, variations have begun to emerge in line with new development technology (e.g. place changing and mobile boot ends). One or two mines have opted for single entry driveage methods, particularly in difficult conditions. Otherwise, twin-entry gateroads are the norm.

Continuous mining equipment selections are diverse, but shuttle cars are the overwhelming transport choice. New generation cutter/bolter miners and fast cycle continuous miners have gained a foothold, particularly in mines with arduous conditions. Anecdotal comments returned from the survey attribute significant performance improvements to the introduction of these machines. Mobile boot end (MBE) devices have been enthusiastically received, although operators have initially preferred to confine their use to static applications. Several operators have been prepared to invest in 'double header' or 'superpanel' configurations to increase daily panel advance, although the effect on overall productivity has not been quantified.

System Performance

In addition to annual performance, operators were requested to identify the configuration of shifts, as well as utilisation and productivity indices within shifts.

A wide variety of rosters are in use although only 10 of the responding mines regularly operated development units on weekends. Utilisation of shift time is illustrated in Figures 9-11 which indicates that:

- nonproductive shift time ranges 1 to 5 hrs, and averages 2.4 hrs/shift. (The accuracy of this data is conjectural.)
- inshift utilisation, which is defined as actual productive hours (cutting and bolting) divided by available face time, ranges from 30% to 75%, averaging 63%.
- real productivity, defined as average metres per cutting/bolting hour, is reported to range from 1 to 4 m/hr, averaging 2.4 m/hr.
- shift productivity in gateroads ranges 3.5-19 m/shift (avge 10.3) and in mains, 3.5-18 m/shift (avgs 9.6), for those responding.

  (Note: some inconsistencies were noted between calculated annual unit shift productivities and stated average productivity m/shift.)
- best unit shift performances in gateroads range 11-55 m/shift (avge 28) and the ratio of best/average in gateroads ranges 1.6 - 3.0 times (avge 2.7).
- panel extension durations in gateroads range 1.5 to 5 shifts, averaging 3, although the effect of simultaneous services advance was not indicated.

The relationship of advance rate and support intensity is shown in Figures 12 and 13. It is expected that the wider introduction of cutter/bolter machines will soften this relationship.

Availability Loss Factors

A detailed analysis of lost availability was outside the scope of the survey, but respondents provide a general indication of key loss factors. A broad classification has been made into the following lost-time categories.

- panel breakdowns (all equipment),
- mining conditions (roof, ribs, floor, gas),
- support infrastructure (belts, power),
- other causes.

The distribution between collieries is illustrated in Figure 14. A wide range of total and distributed downtimes are indicated which may reflect low order accuracy of the survey. The distribution, however qualified, will nonetheless be of some interest to industry members.

Other Comments

A variety of other comments were provided by some operators. It is apparent that most are prepared to introduce changes, and adopt new technology, in a steadily progressive fashion in pursuit of gateroad development improvements. Safety issues figure prominently as a change-driver.

CONCLUSIONS AND RECOMMENDATIONS

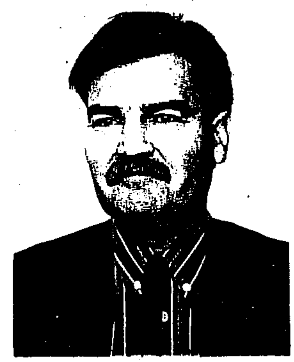
The survey of longwall mines in 1993 has served to present an overall description of the status of roadway development performance. Significant system parameters have been identified and compared which should be of interest to operators, researchers and equipment providers. The exercise has also provided another benchmark against which future performance can be assessed.

It is observed that several mines are achieving well in advance of 10m/shift in development roadways, although the potential for increasing longwall retreat rates and the pressure for lower cost operations will push these targets progressively higher. Despite the significant impact of cutter/bolter continuous miners on performance in difficult conditions, there are several mines reporting average development rates of much less than 10m/shift.

There is a wide divergence between reported utilisations, real productivities and availability loss factors. Whilst this may be attributable in part to differing definitions, interpretations or degrees of reporting accuracy, the potential upsides indicated between collieries provide some optimism for significant increases in system performances.

Given the observed interest in upgrading technology, it would appear useful to maintain an industry-wide performance review mechanism. Whether this is by annual surveys, or a more regular reporting process, as with longwall statistics (JCB), will depend on industry enthusiasm, in which case the less onerous the data provision process, the more likely the success.

Respondents to the survey are thanked for their efforts and cooperation.



Bruce Robertson

ACARP Roadway Development Workshop

CONFIDENTIAL

Survey of Australian Colliery Roadway Development Activities in Calendar Year 1993

EXAMPLE ONLY

LONGWALL COLLIERIES (1993)

Colliery Name	*Example Colliery No1*				
Total Performance in Cal Year 1993	**LW gates**	**Mains**	**Other**	**Total**	**Comments**
Total metres driven	12500	4000		16500	
Productive unitshifts operated*	1200	400		1600	
Planned production unitshifts*	1750	650		2400	
Avge active cont miner unitshifts deployed per day##	9	3		12	
(*count double header or super sections as one unit)					
(##count each contin miner separately)					
System Characteristics - Mining Conditions					
Total seam height (m)	4.1	4.1			
Roadway height (m)	2.8	3			
Roadway width (m)	4.8	5.5			
Immediate roof type	*coal*	*shale*			
Roof conditions (strong, moderate,weak)	*mod*	*strong*			
Immediate floor type	*sandst*	*sandst*			
Floor conditions (strong, moderate, weak)	*strong*	*strong*			
Max advance before roof support reqd (m)	3	3			
Pillar size (solid - m x m)	*95 x 35*	*40 x 40*			

System Characteristics - Section Configuration	**Contin Miners**	**Sh Cars**	**Other**	**Comments**
List major equipment type @ numbers in use	*abm20 @ 2*	*15sc @ 4*		*Double header panel*
(match haulage to CM)	*12cm20 @ 1*	*15sc @ 2*	*mbe*	
	12cm11 @ 1	*15sc @ 1*	*12t surge car*	

System Characteristics - Performance Details	**LW gates**	**Mains**	**Other**	**Total**	**Comments**
Typical roof support					
..roofbolts per metre advance	6	6			
..ribbolts per metre advance	2	2			
..straps/plates/mesh used	*straps*	*straps+mesh*			
..other support installed during development	*cables*				*9 m @ intersections*
..other support installed before extraction	*cables*				*11m @ 3m pattern*
System Performance					
Roster system (general type)	*4 @ 7hr @ 5day*				
Normal shift length (hrs)	7				
Available face time per shift (hrs)	5.5	5.5			
Avge cutting hrs per shift (hrs)	3.5	3.5			
Avge advance per cutting hr (m/hr)	3	2.8			
Avge advance per normal shift (m)	10.5	10			
Best advance per normal shift (m)	23	20			
Typical panel extension duration - net interruption(shifts)	4	2			

Loss Factors	**Factor**	**mins**	**Comments**
Main availability loss factors (Typical mins/unitshift lost)	*wet conditions*	30	
	belts	20	
	roof	20	
	eqpt breakdns	30	

Other Comments

abm20s lifted productivity from 6 to 11 mpus
super panel introduced mid year, yielded 30% panel advance rate increase
surge car introduced late in year, no noticeable increase in advance but safer cleaner system
mbe has improved panel safety, reduced labour costs, reduced belt trips
main concern is slippery wet conditions under cms and soft ribs
no problems with gateroad integrity for longwall working
currently looking at new bolt/mesh system to increase rib bolting productivity/safety

Contact for queries regarding this survey response: NAME: *J.Helpful* **PHONE:** *077-333333*
FAX: *077-444444*

Please return this completed survey to Bruce Robertson Fax 07-8325751

FIG 1

FIG 2

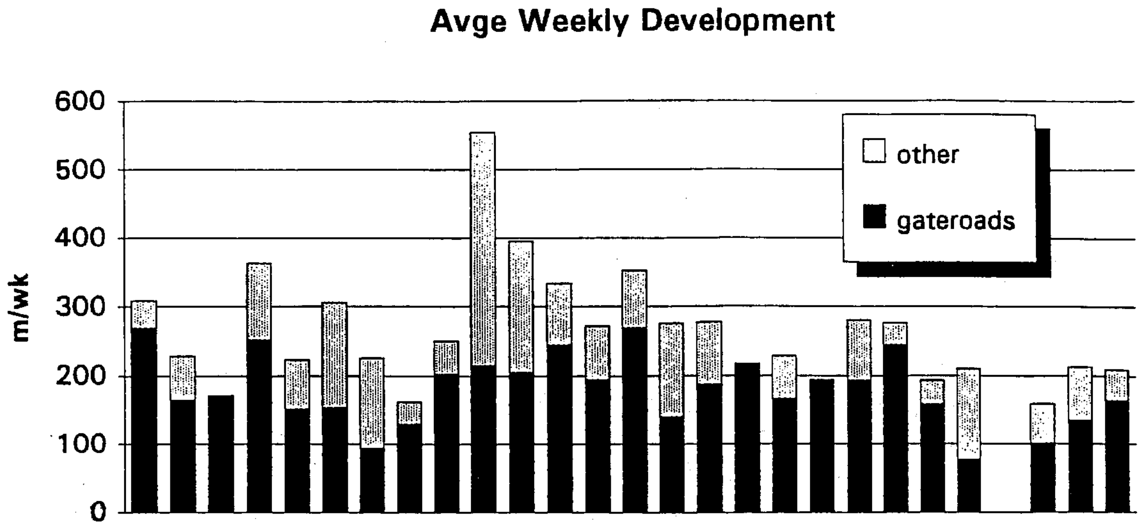


FIG 3

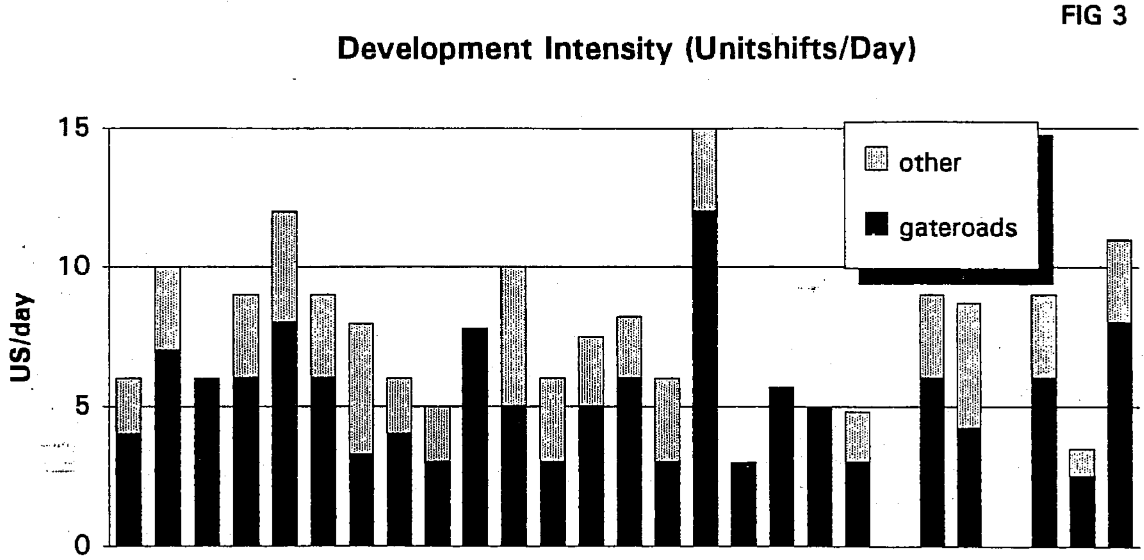


FIG 4

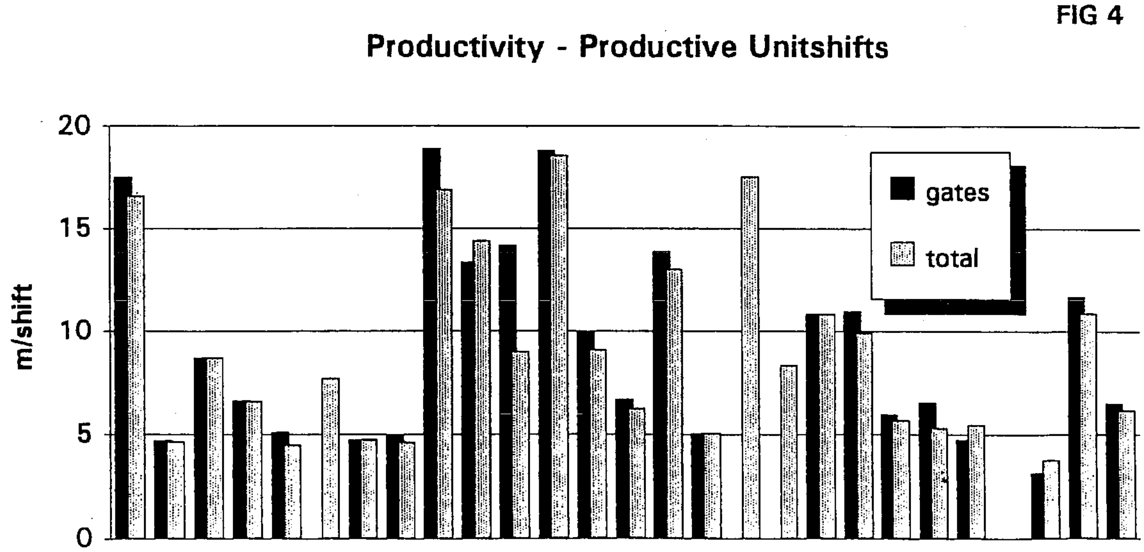


FIG 5

Roof/Floor Conditions

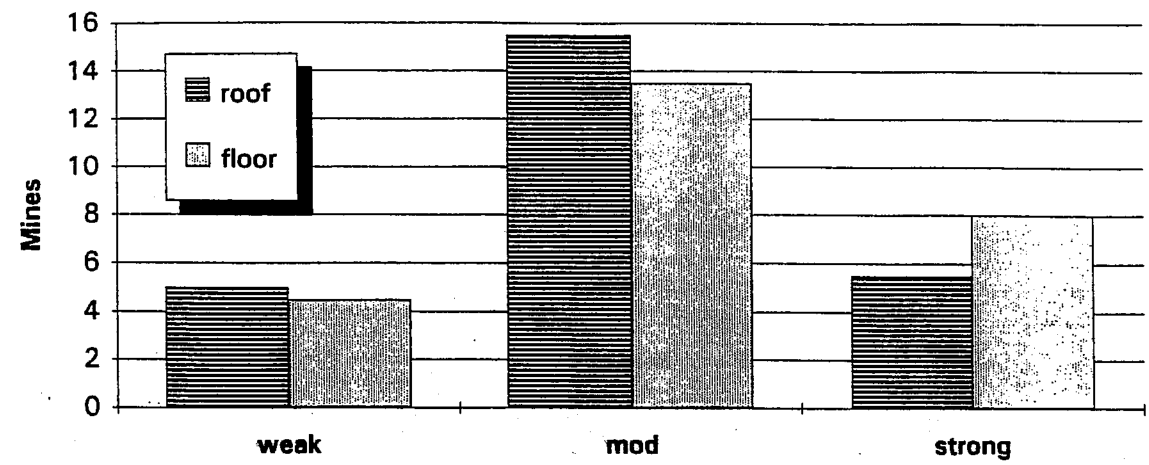


FIG 6

Bolting Intensity

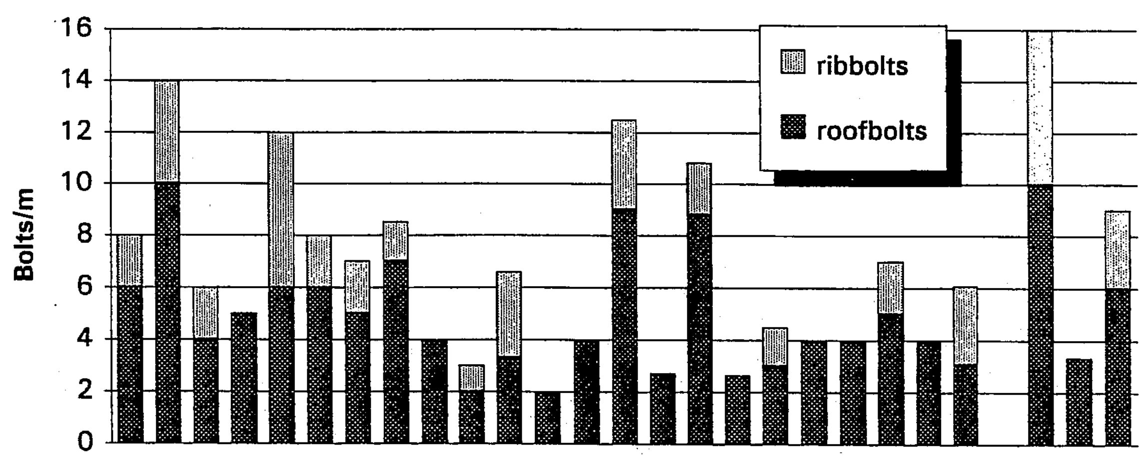


FIG 7

Roadway Dimensions

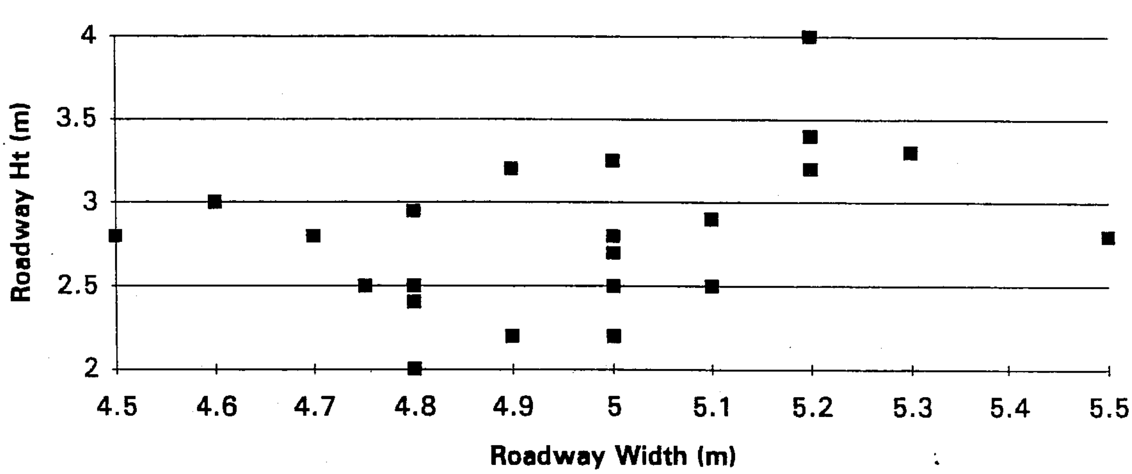


FIG 8

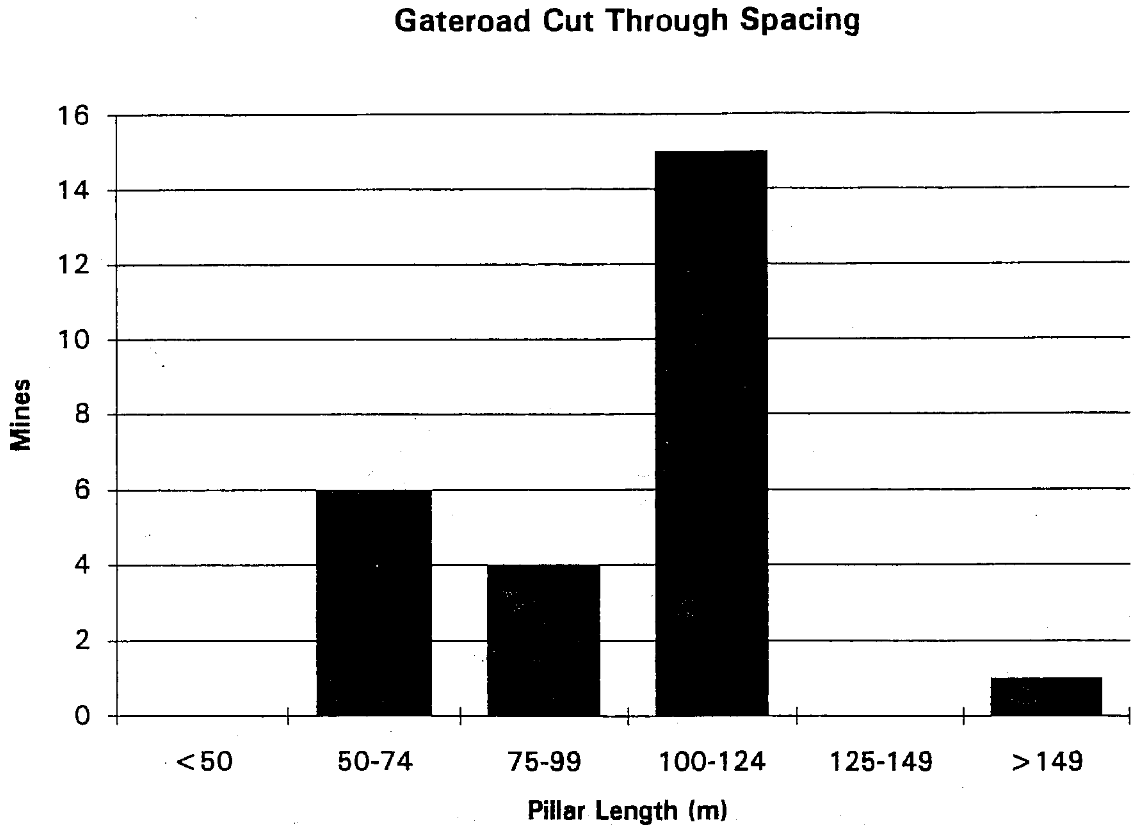

Gateroad Cut Through Spacing
16
14
12
10
8
6
4
2
0
Mines
<50
50-74
75-99
100-124
125-149
>149
Pillar Length (m)

FIG 9

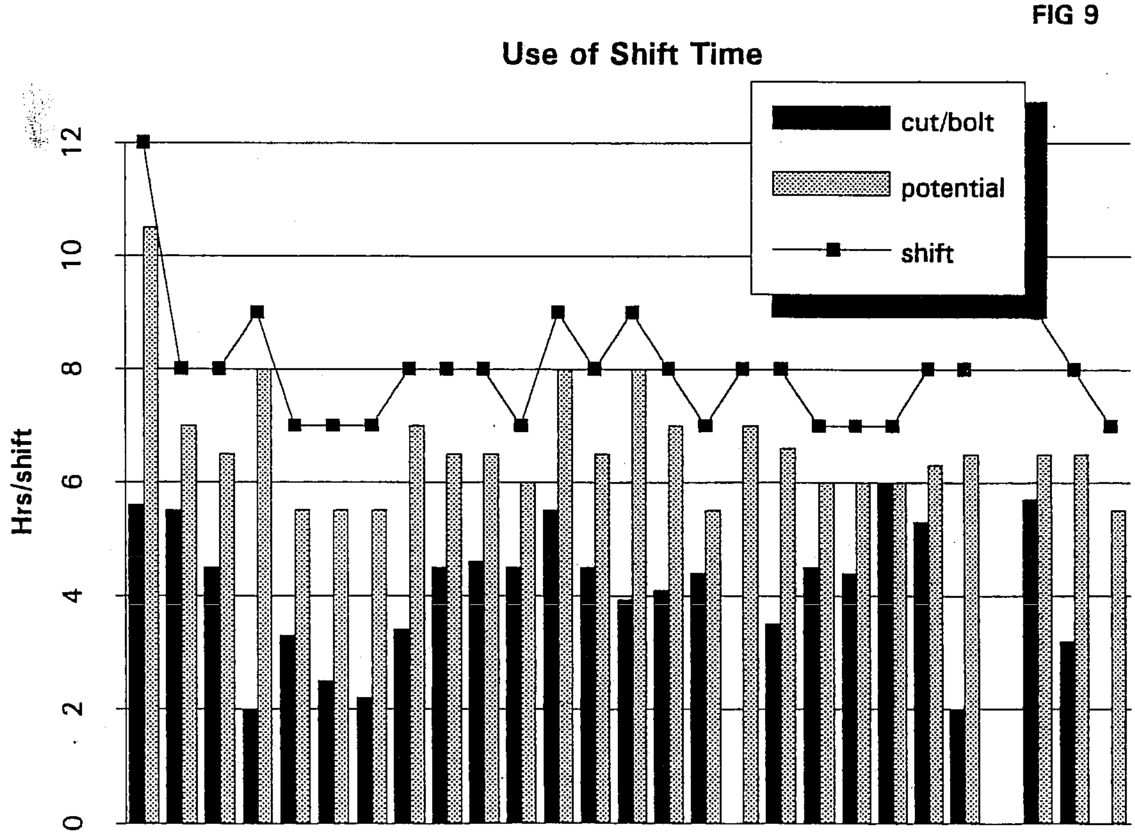

Use of Shift Time
cut/bolt
potential
shift
Hrs/shift
0
2
4
6
8
10
12

FIG 10

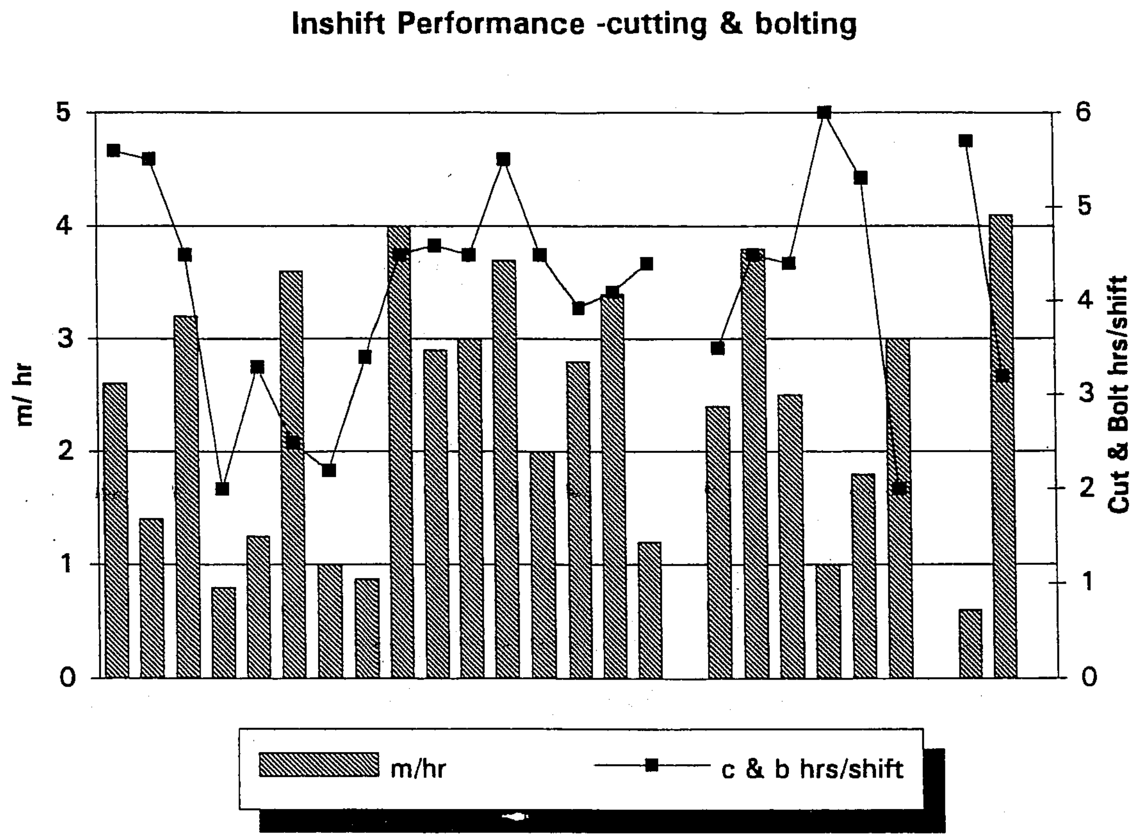

Inshift Performance -cutting & bolting
m/ hr
Cut & Bolt hrs/shift
0
1
2
3
4
5
6
m/hr
c & b hrs/shift

FIG 11

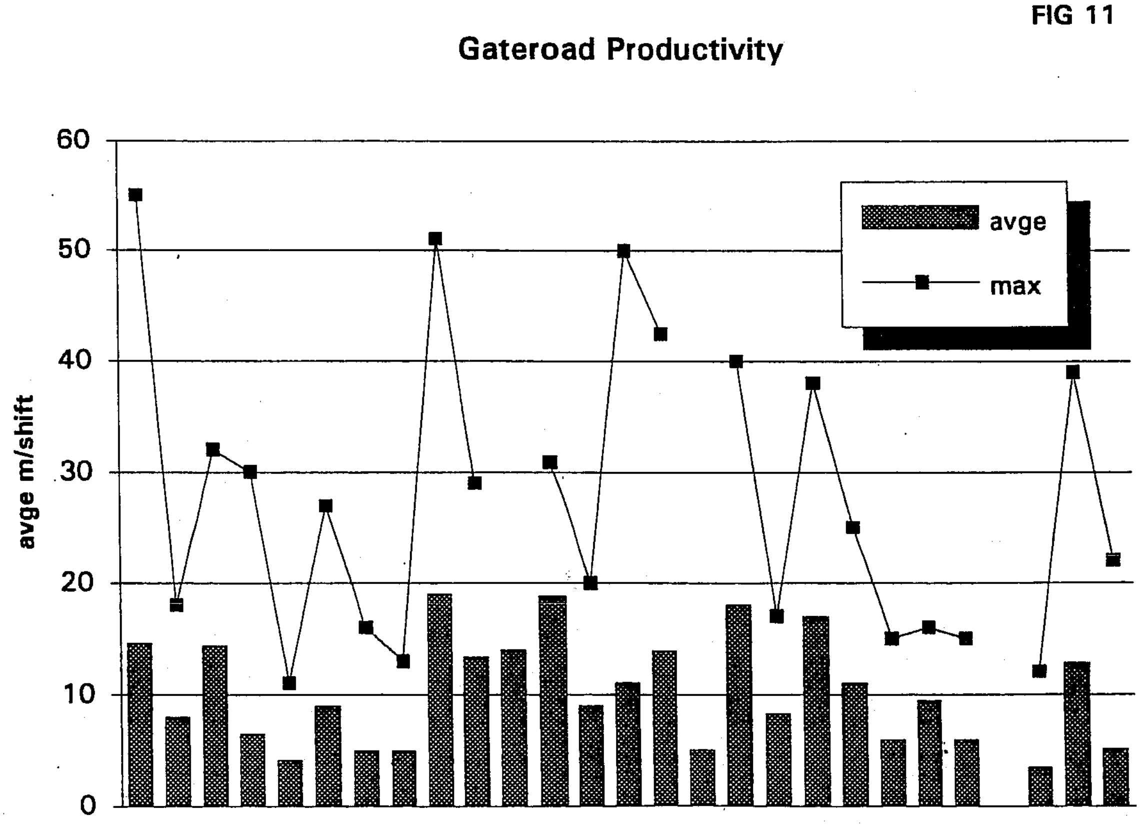

Gateroad Productivity
avge m/shift
0
10
20
30
40
50
60
avge
max

FIG 12

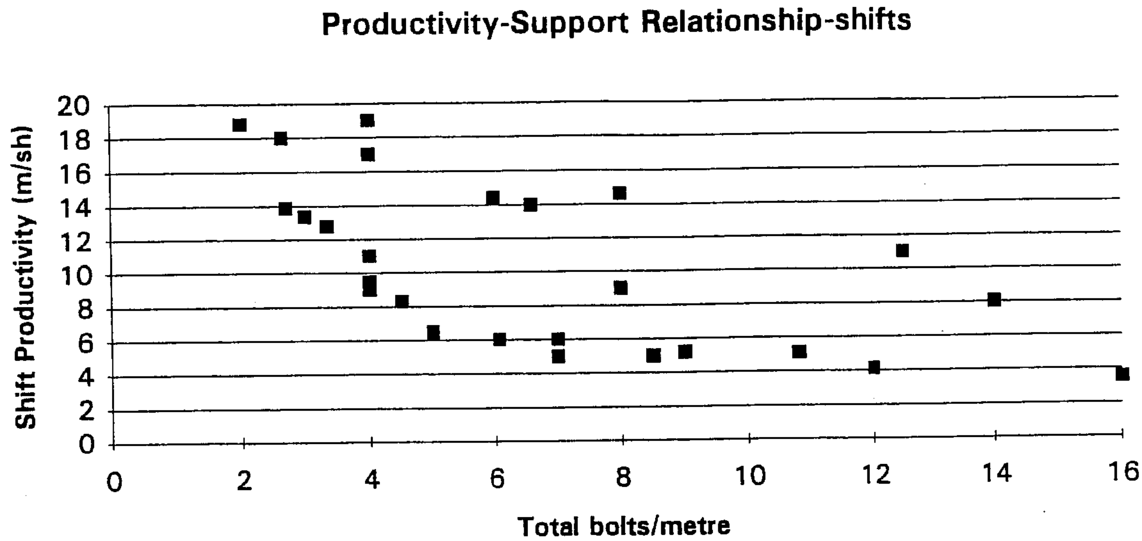


FIG 13

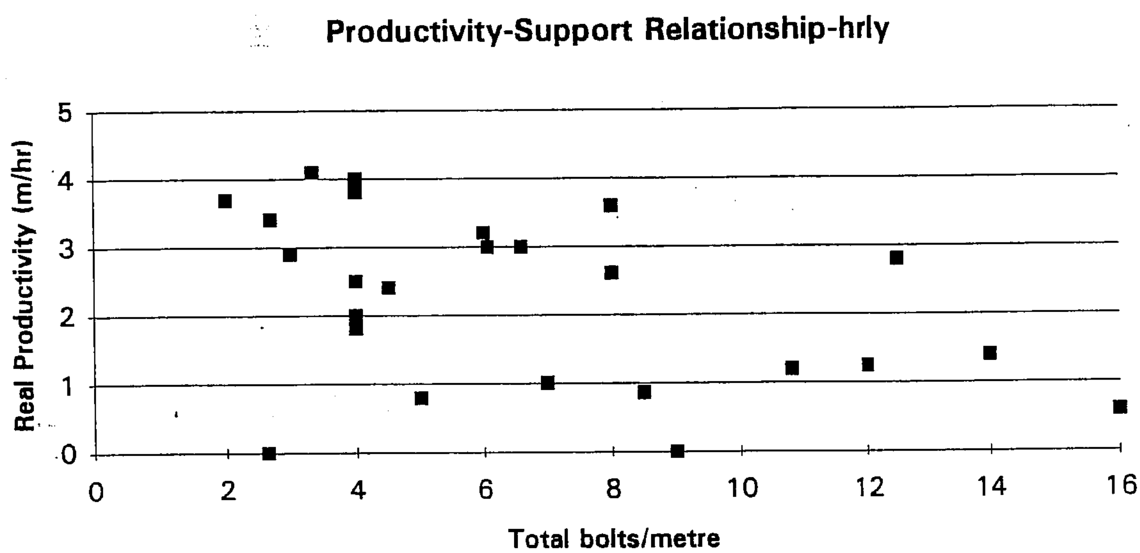
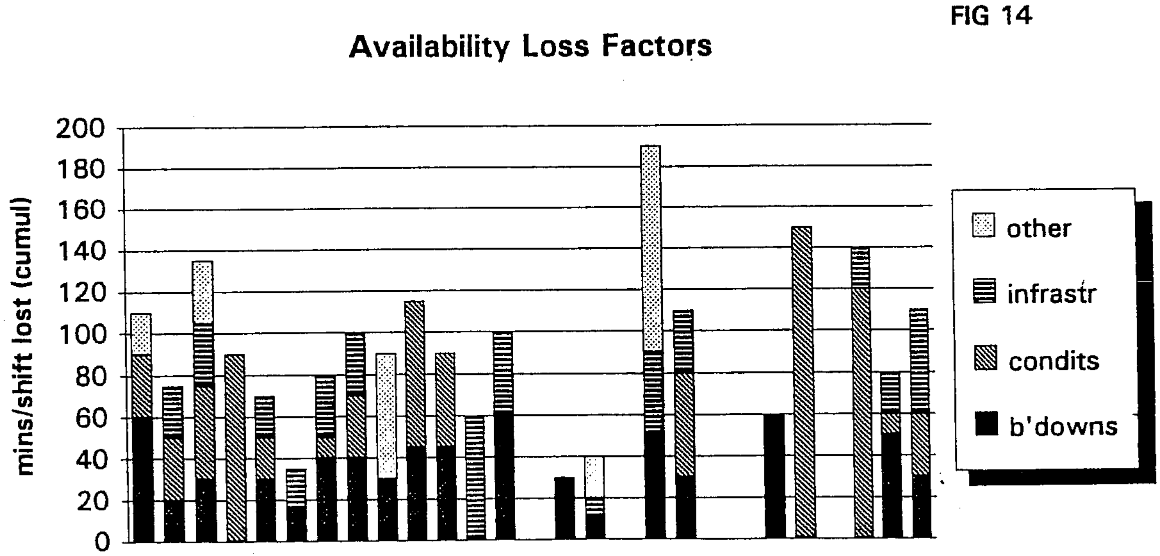


FIG 14



GEOTECHNICAL AND SUPPORT SYSTEMS

The task of driving roadways in coal mines can be viewed as a complex interaction of subsystems each of which can hinder the performance of the total system. This session of the workshop will deal with one of these subsystems. The topics will include a range of issues related to the existing ground conditions and the response of strata to excavation and reinforcement, during the development cycle.

As with all other aspects of mining, the key impact issues are productivity, cost, safety and risk. But are these issues adequately addressed in routine roadway support considerations and practice? Shouldn't we consider technology development in a framework which compares alternative systems using a consistent and disciplined methodology which encompasses the key impacts described above?

There have been a great many advances in the measurement, understanding and methodology applied to ground control in Australian collieries and, fortunately these progressive steps have been documented in the literature. In assembling the authors and papers for this session I have endeavoured to cover the field of current interest in the following areas:

» general design concepts
» roof reinforcement/control
» rib reinforcement/control
» floor management.

Time will not allow a full treatise of all issues, and the authors have been requested to focus on key aspects only.

Ross Seedsman will present an overview paper which calls for an engineering design approach to ground control. Ross believes that, with our hard won knowledge and understanding of the interaction of strata and reinforcement hardware, we are well placed to devise analytical techniques that will provide a quantum jump in roadway support strategy.

Winton Gale, Mike Fabjanczyk and Greg Tarrant have prepared a paper which summarises the SCT approach to design of reinforcement systems using computational techniques. Particular emphasis is placed on monitoring the behaviour of ground as inputs to validate simulation models which can then be used to predict behaviour under varying conditions.

Peter Fuller and Paul O'Grady describe a new WINDOWS-based analysis for designing roof reinforcement which is tuned using deformation measurements. Ian Clark's paper presents some example simulations to illustrate ways in which specific aspects of ground reaction, including local defects, can be accommodated. Ian illustrates the range of problems and treatment approaches that are now possible with digital processing power.

Ken McNabb and Leigh Wardle provide strong argument for wider and more intensive use of monitoring as an essential component of an integrated geotechnical program. A summary of methods and new developments is provided. Rod Thomas and Russell Frith have prepared a concise summary of the science of pretensioning roof bolts. They believe that there are substantial benefits that can be gained and develop their views using recent field experience.

Bill Lawrence presents an insight into the practicalities of managing a ground control program in parallel with mine operations at Gordonstone. In a paper based on uptake of new technology at Ellalong, Brian McCowan illustrates that despite the onset of very difficult ground conditions, new roof and rib reinforcement tools are contributing to a successful solution strategy. Likewise, Bob Butcher in his paper

dealing with the introduction of flexibolts at Angus Place Colliery, documents how a rational response led to the development of a safe and efficient roof control system.

Andrew Logan, Ross Seedsman and Geoff Cox have prepared a paper describing the outcomes of a recent ACARP-funded project on colliery road pavements. A manual has also been published.

Geotechnical aspects must be viewed as an integral component of the total roadway development system, not as an end in themselves nor as just a simple cost/time consumable. Perhaps the goals of geotechnical engineering in a roadway development context can be stated as "achieving a safe, reliable ground support for the mining conditions and development system used, with progressively faster and lower cost impacts on the total system".

BRUCE ROBERTSON
Shell Coal

GEOTECHNICAL SUPPORT SYSTEMS

Australian Coal Association

ROSS SEEDSMAN

Coffey Partners International Pty Ltd

AN ENGINEERING DESIGN APPROACH TO ROADWAY SUPPORT

In response to the demands for faster development rates, the Australian industry is being offered a sometime bewildering range of new products and procedures for ground improvement. This session will discuss some of these — flexibolts, AX bar, pretensioning, yielding rib bolts, instrumentation, FLAC analyses etc.

Furthermore, the regulations and guidelines under which the industry operates are changing with a move from government prescription towards industry self management. In this environment there is a need for a framework which compares alternative ground support strategies in the context of productivity, cost, safety, and risk. Such a framework should provide a consistent and disciplined methodology that can identify, and assess the confidence inherent in, the complex decisions that need to be made regarding ground support.

DESIGN IN ROCK MECHANICS

Ground improvement must be seen as an integral component of a total roadway development system. Geotechnical engineering issues, and especially those related to rock mechanics, need to be at the forefront when considering ground improvement, for it is the unknowns of the rock mass and stress field that present the greatest exposure to risk. It is proposed that a rock mechanics design methodology offers the framework required to compare alternative ground improvement strategies. In fact it can be argued that good risk management is simply good engineering design.

The last 10 years has seen major advances in our knowledge of the behaviour of bolted roof. The complex redistribution of stresses around a developing roadway and a retreating longwall panel have been measured and are now understood.

We can now measure roof and bolt deformations. It has been shown that AX bar, cables, cable bolts, and pretensioning can markedly improve some roof conditions. Both yielding and rigid rib support systems are being trialled. The application of civil road design and analysis concepts has allowed a major improvement in our ability to construct and maintain underground pavements.

Despite these advances in our knowledge, the 'development constraint' has remained — longwalls are still standing idle waiting for the next block to be developed — and even more productive (faster retreating?) longwall faces are being planned. The development constraint is now recognised to be the result of complex interactions between ground conditions, support hardware, mining machines, services, and especially man and system management.

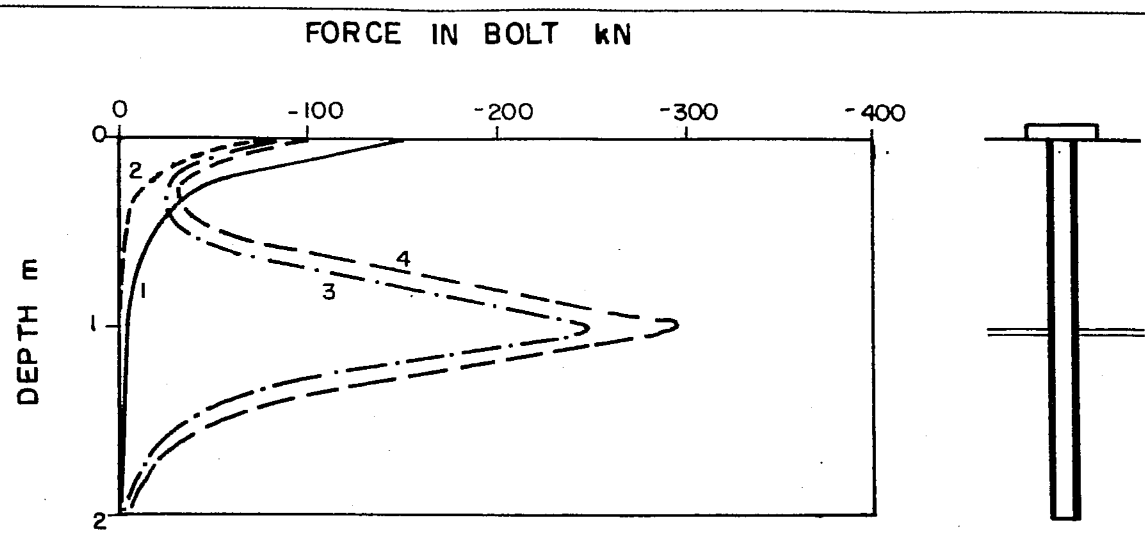
This paper proposes that what is needed from geotechnical engineers to assist in breaking the development constraint is the emergence of good practical *engineering design* especially based on the application of analytical design techniques.

In recent times, empirical and observational design methods have been used in Australia. These have allowed the industry to advance the state of the art but cannot, by their very nature, provide the quantum jumps that are need to break the development constraint. It is suggested that what is needed to make a significant advance in roof and rib support is the ability to explicitly predict the behaviour of individual bolts and how the rock behaviour can be modified.

THE BEHAVIOUR OF INDIVIDUAL BOLTS

Recent ACARP funded research has shown that bolt loads can be readily predicted from a knowledge of bedding shear or separation. Coffey has shown that well established and proven pile theory can be used to analyse bolt behaviour.

For example, Figure 1 shows the development of axial load in a bolt as a bedding surface opens. It can be inferred from this figure that almost any roof or rib movement will cause bolt yield. Figure 2 shows how the shear force applied by an untensioned bolt (dowel component) increases with rock modulus and shear movement. Information such as that shown in these figures can be used to assess which are the most appropriate avenues for improving bolt capacity.



RESIN f_s = 6 MPa

E_r = 2000 MPa

CURVE	DESCRIPTION
1	INITIAL TENSIONING TO 150kN AFTER RESIN HAS SET
2	CAP TIGHTENED
3,4	VERTICAL ROCK MOVEMENTS OF 2.5, 5mm RESPECTIVELY

Figure 1. Axial force distribution in response to bed separation.

AN APPROACH TO SUPPORT DESIGN

Bolts within a coal-mine ground support strategy may need to fulfil a number of perhaps conflicting roles:

- beam building in laminated roof,
- beam building in a sheared roof,
- suspension of detached blocks,
- mesh or strap installation,
- restraint of yielding ribs,
- reinforcement of a rib.

It is postulated that one job of roof bolts is to 'trick' a laminated roof into 'thinking' it is a thickly bedded unit which can readily span the excavation. From the design engineer's perspective, this translates to the requirement to prevent unacceptable amounts of shear between the laminations. An approach to specifying a support pattern would then follow the following steps:

- determine the horizontal and vertical stresses acting on the bedding surfaces,
- determine the magnitude of the shear stress in excess of the cohesive and frictional restraint offered by the surfaces,
- establish the magnitude of shear movement implicit in the adoption of an allowable centreline deflection,
- refer to bolt design charts to establish shear restraint offered by various bolt types and installation techniques,
- determine a bolting pattern that balances the excess shear with an appropriate factor of safety.

Figure 3 is representative of the bolt design charts that can be prepared. The figure compares a number of installation options for AX bar and compares active modes such as pretensioning, with passive modes such as the typical dowel mode and angled bolt installation. Monitoring of the roof would be based on checking that the design centreline deflection is not exceeded.

If the *in situ* horizontal stresses approach and exceed the compressive strength of bolted roof beam, shear failure of the beam is possible. In many ways the failed beam would appear like a large example of a failed rock specimen after an unconfined compressive strength test. It is considered that the resultant shortening of the roof beam would result in the transfer of the

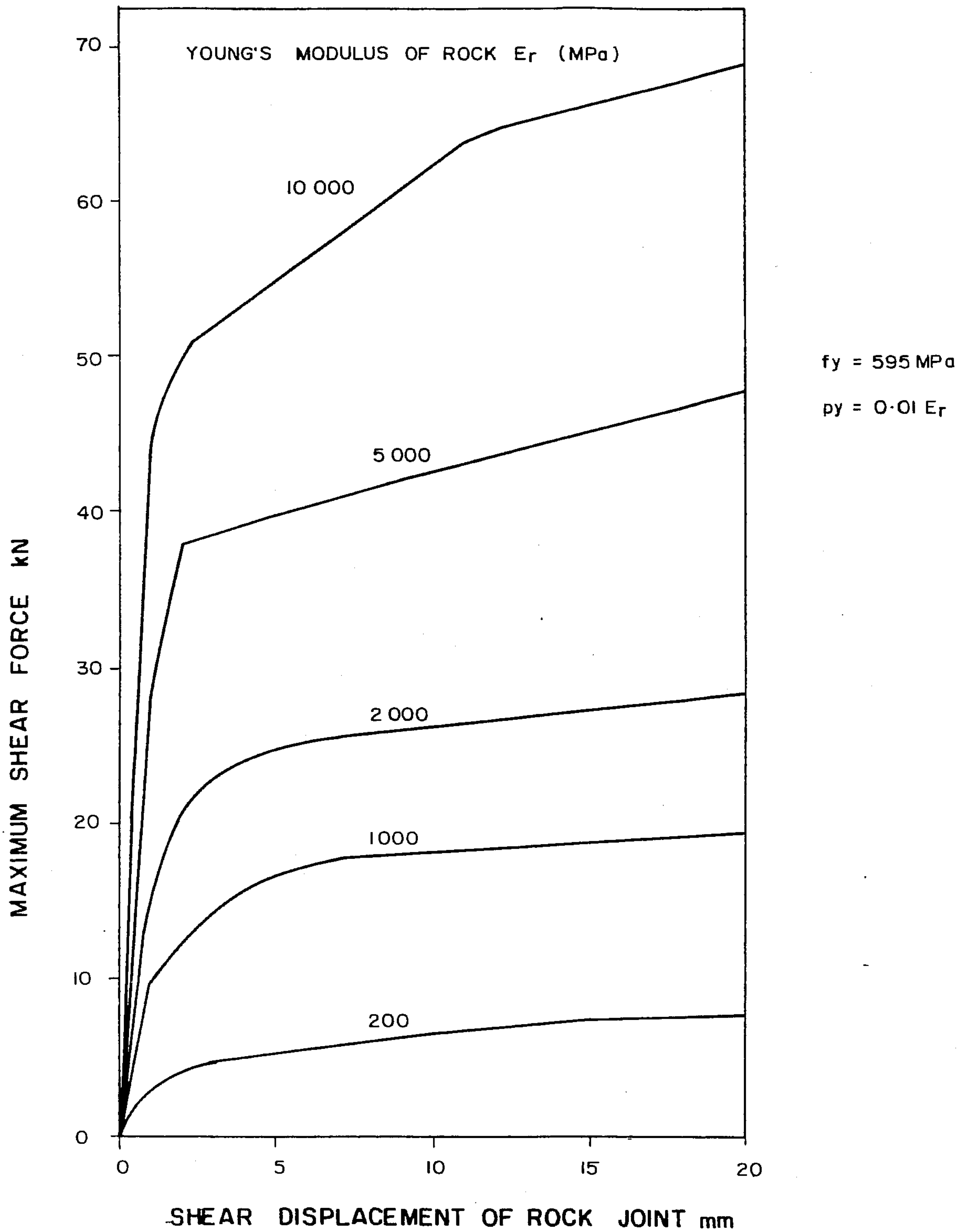


Figure 2. Effect of rock modulus on maximum shear force.

horizontal stresses into the overlying roof with the consequent development of a softened zone. It is postulated that the role of roof bolts in such an environment is to attempt to hold the broken wedges of rock together against a dead load from the softened zone (Figure 4). The statics of such a problem are readily solved and Figure 5 presents the relationship between the height of the softened zones and the required installed bolt capacity. It is noted that in such a model only axial bolt capacity is mobilised.

WHERE TO NOW?

If engineering design, and especially analytical techniques such as those outlined above, are the way forward, then the first step must be the calibration and validation of the various design

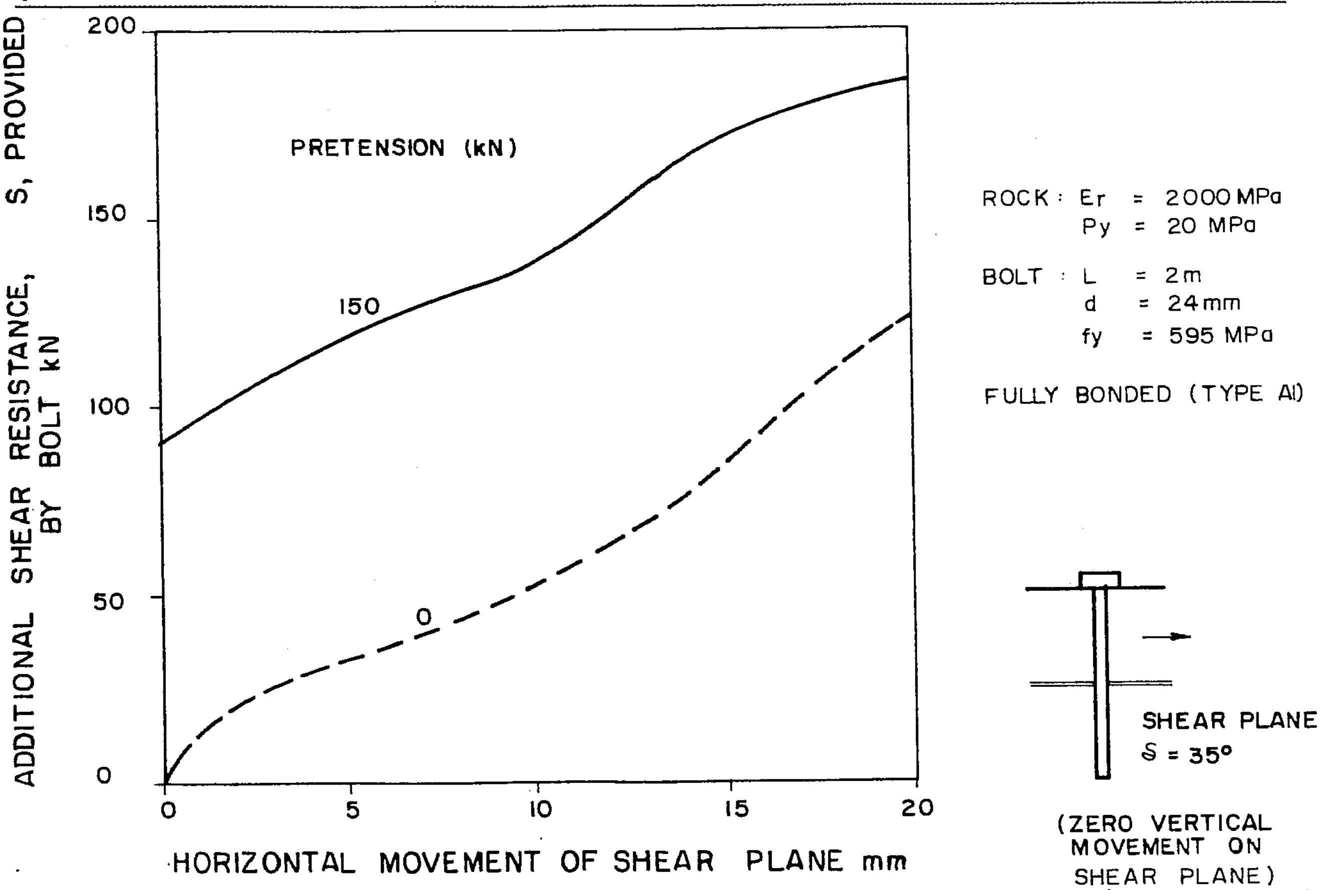


Figure 3. Representative design chart for fully bonded bolt.

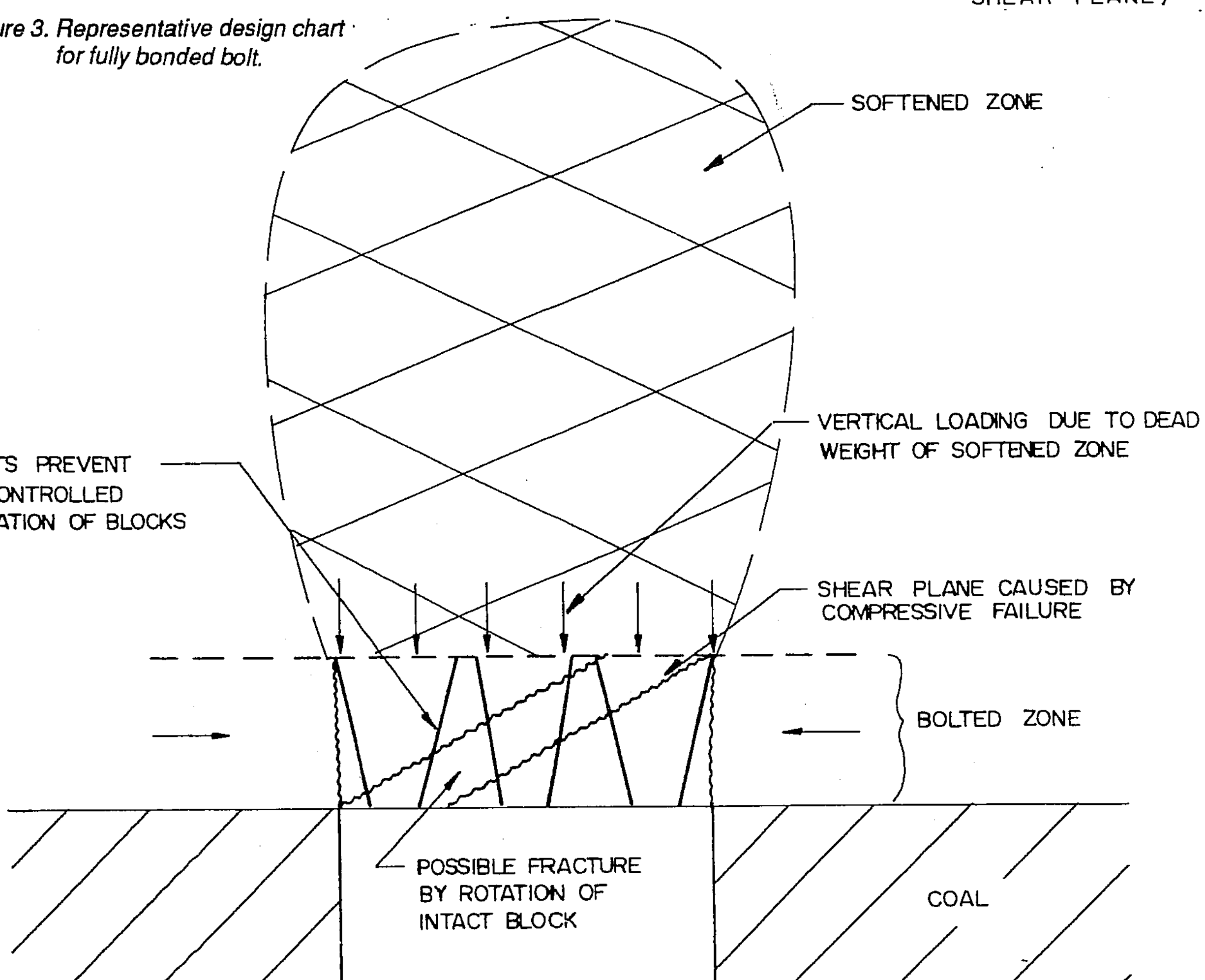


Figure 4. Support mechanism for roof undergoing compressive failure.

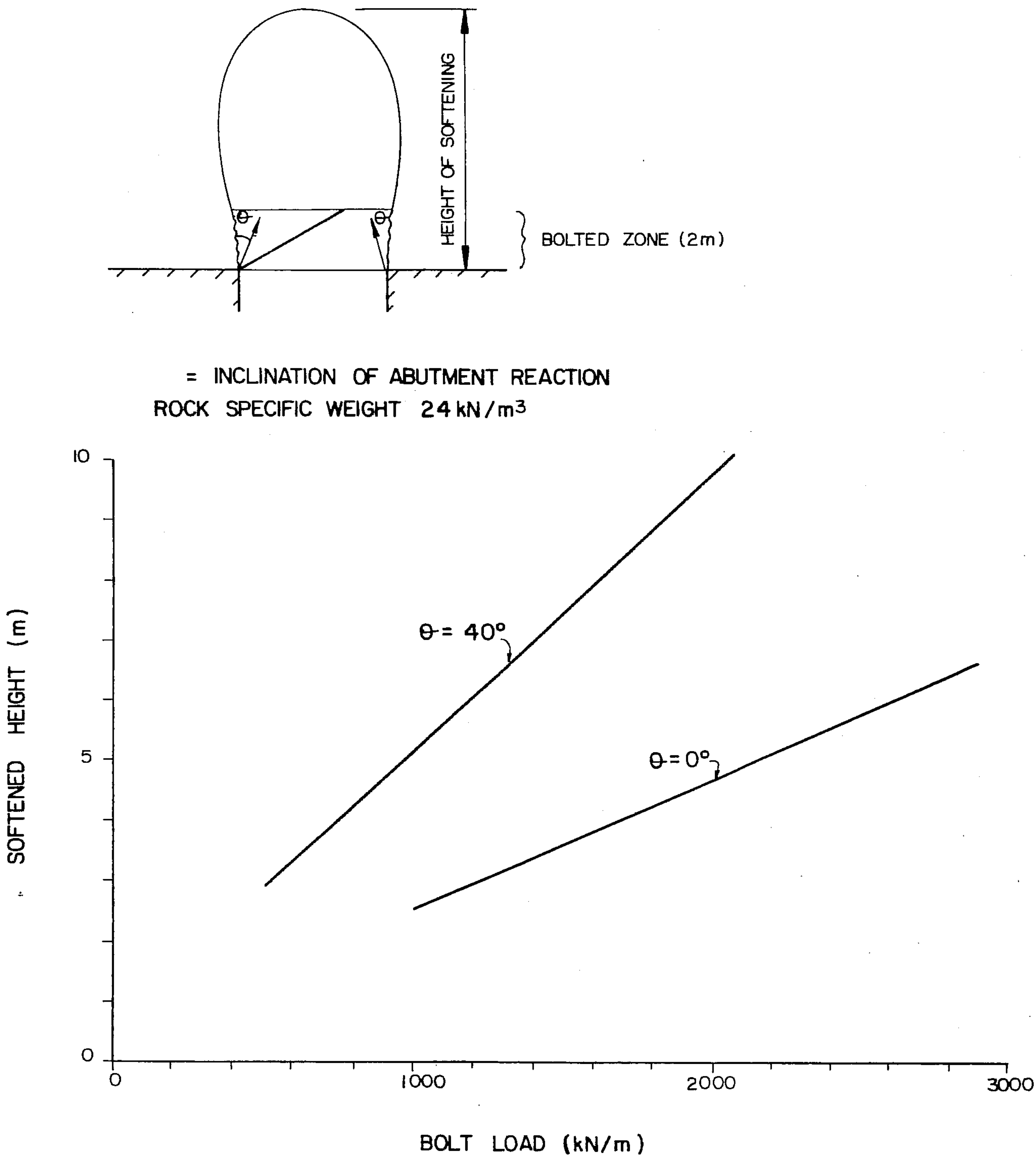


Figure 5. Bolt capacity requirements for support of softened zone.

techniques. Analytical techniques offer the opportunity to establish 'factors of safety' (typically defined as the quotient: - restraining stresses/imposed stress). However, it must be recognised that the numerical value of a factor of safety is an heuristic or 'rule of thumb'. Design values can and will change with the analytical techniques used to determine them and with consideration of the engineering venture. It is not a trivial exercise to relate factors of safety to probabilities of failure. It is stressed that the selection of the design factor of safety should be the responsibility of the design engineer and the project owner and not be prescribed by legislation.

Not only do the techniques described above need verification, but also the empirical techniques such as the RMR and Q system should be developed as they do show great promise especially if applied rigorously.

A similar model to that developed for compressive roof failure could perhaps be developed for the specification of rib support. It is postulated that only the self-weight of the coal rib may need be considered. Such a model would require that shear is the dominant mechanism in coal ribs. It is noted that this is perhaps the implicit assumption in metalliferous mining where split sets are used routinely for side support. Do split sets have a role in rib support, especially given their ease of installation?

As is always the case, the availability of engineering design techniques results in more emphasis on the quality of the input data. Whilst we know much about the distribution of vertical movements in bolted roofs, we must now recognise it is perhaps knowledge of the shear movements that may be more important. We must also learn more about how to identify the critical rock defects along which movement may occur and also develop techniques to rapidly assess rock properties. It is noted that instrumented rock bolts will require strain gauging at about 50mm centres to reliably detect shear movements.

From an operator's perspective, there is great potential in improving roof stability and reducing bolting cycle times in laminated roof strata with either pretensioned and/or angled bolt installations. However, it should be recognised that the final support pattern must recognise the possible need to support isolated blocks and the installation of straps and mesh. The final pattern must also accommodate the implications of the possible onset of compressive failure.

The Coffey work suggests that most if not all bolts do yield after very small strata movements. The reported success of fully encapsulated bolts may be related to their intrinsic post-failure or yielding capacity. There may be the opportunity to re-engineer point-anchored bolts so that they can yield better and gain significant reductions in bolt installation time.

WINTON GALE, MIKE FABJANCZYK & GREG TARRANT

Strata Control Technology Pty. Ltd.

DESIGN, CONFIGURATION AND CONTROL OF ROADWAY REINFORCEMENT SYSTEMS

The design of underground mine roadways requires a detailed understanding of the in situ stressfield, the rock failure modes, the performance characteristics of reinforcement or support members and assessment of the stress state about the roadway. Definition of these factors, together with monitoring of roadway behaviour during both development and extraction operations, allows rational design procedures to be undertaken.

The design approach utilises computational techniques to simulate the behaviour of the strata about the opening for a range of anticipated stressfields and various reinforcement options. This approach has been successfully applied at 11 Australian Collieries as well as Indonesia, Japan, U.S.A., U.K. and New Zealand over a period of 5 years.

The design procedure adopted combines:

1. a monitoring program to define the behaviour of the ground about the roadway and the performance of the reinforcement during development and extraction, together with
2. a design study utilising computational modelling to simulate the behaviour of the strata and reinforcement about the roadway for a range of stressfields which may be encountered and reinforcement options to be assessed.

Validation of the model simulations with field monitoring during the design stage and subsequent to the design is a key factor to provide confidence in the application of the design.

Significant variability in both lithology and stressfield is noted in many Australian mines. These variations include:

- Variations in lithology due to varying deposition,
- Affects of varying levels of geological structure,
- Variation in the *in situ* stressfield due to either the local or regional tectonic history.

Theses factors, which are responsible for change in conditions, are in general understood, however, the ability to accurately predict the location and intensity of the change is at present limited.

The computational modelling has been used to analyse the implication on roadway stability of the range of change likely to occur, but additional controls are required to identify the position and intensity of the change so that the appropriate change in reinforcement strategy can be adopted.

MODELLING PROCEDURE

The monitoring and computational modelling procedures are well described in Gale & others (1992) and Gale & Fabjanczyk (1993).

SCT has undertaken a program to develop rigourous computer simulation and field validation methods by which design of mine roadways may be undertaken with the assistance of computer methods. Computer simulations are typically used in association with field monitoring to provide validation prior to full implementation of the design.

The application of computer simulations requires a detailed definition of the strata and their strength properties within the intact and post failure states.

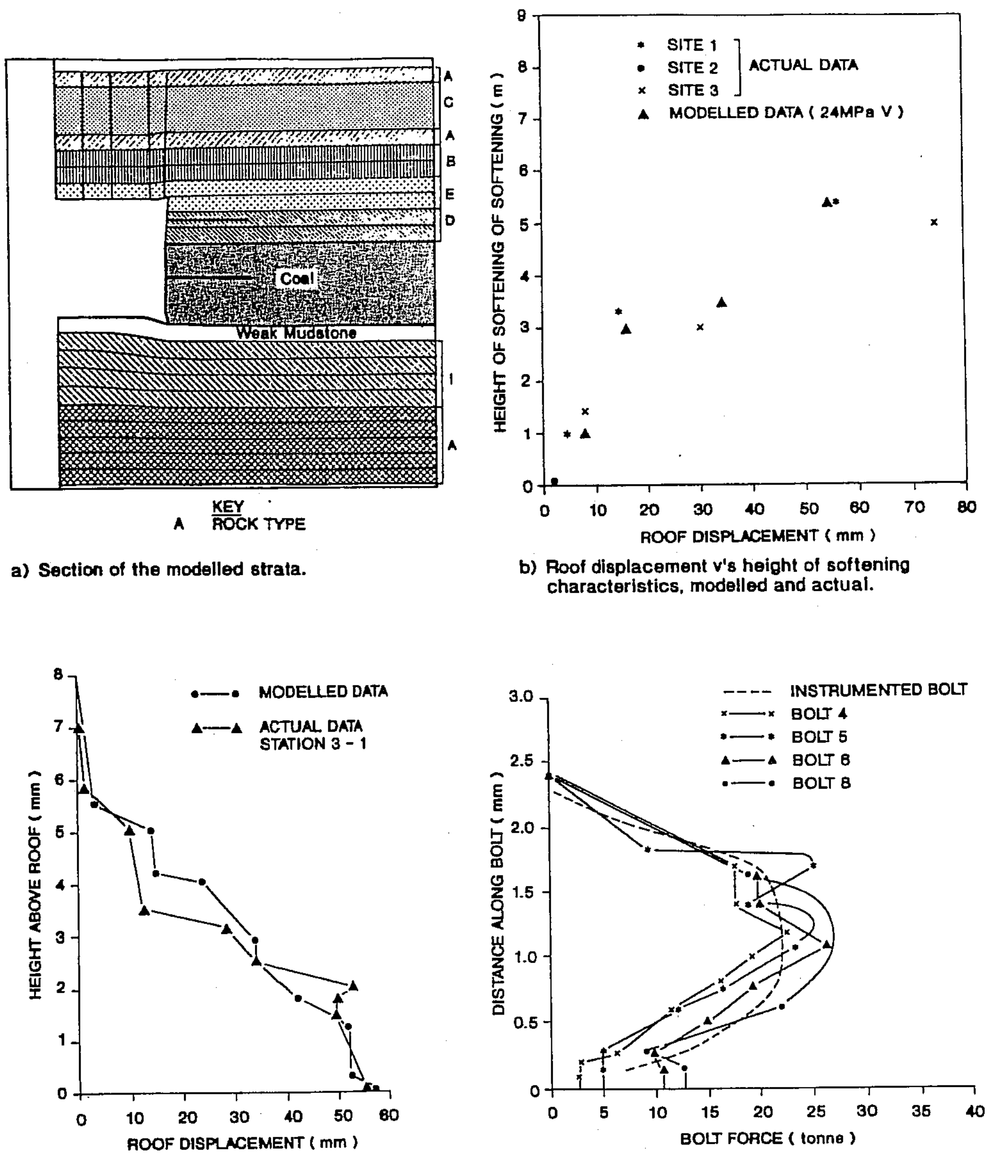
Similarly, a detailed understanding of the stressfields and the geological variability to be encountered, is necessary to select the most relevant situations to be modelled.

If the models are created with sufficient detail and rigour, then it has been found that they can provide an accurate simulation of:

1. roof and rib deformation
2. reinforcement axial force distribution, and
3. rock and coal strength characteristics about the roadway.

The simulation of these characteristics can allow the design of reinforcement patterns and an assessment of the stability of mine roadways using reinforcement options within a variety of stressfield, and strata conditions which have not been previously experienced.

Some results of the application of these methods are presented below which show the



a) Section of the modelled strata.

b) Roof displacement v's height of softening characteristics, modelled and actual.

c) Comparison of modelled and actual roof displacements.

d) Actual bolt load distribution compared to modelled (Station 3).

Figure 1. Roadway simulations with monitored behaviour.

correlation of a number of key criteria between the modelled and that actually monitored.

Figure 1 presents an example model with correlations of key factors. A correlation of the monitored roof deformation characteristics of roof displacement and height of deformation under a variety of stress conditions is presented in Figure 1a. The correlation is considered to be excellent and indicates that the model is simulating the general style of roof deformation.

A more detailed correlation of the roof deformation is provided in Figure 1c by a comparison of roof extensometer data from the mine and that predicted by the model.

A correlation of roof bolt forces developed within that deformation regime, together with those predicted by the model, are presented in Figure 1d.

The computer simulations when validated against actual monitored data can allow design studies to be undertaken within a range of as yet unexperienced conditions.

The application of such design studies is to provide a guide as to the anticipated roadway behaviour, deformation mechanisms and the design limits placed on various reinforcement systems under those conditions.

UTILISATION OF THE COMPUTATIONAL MODELLING SYSTEM

When the validated computational model is completed, the reinforcement design for the expected conditions can be prepared with a high level of confidence. The confidence provided by verification of the modelling through confirmatory monitoring and the

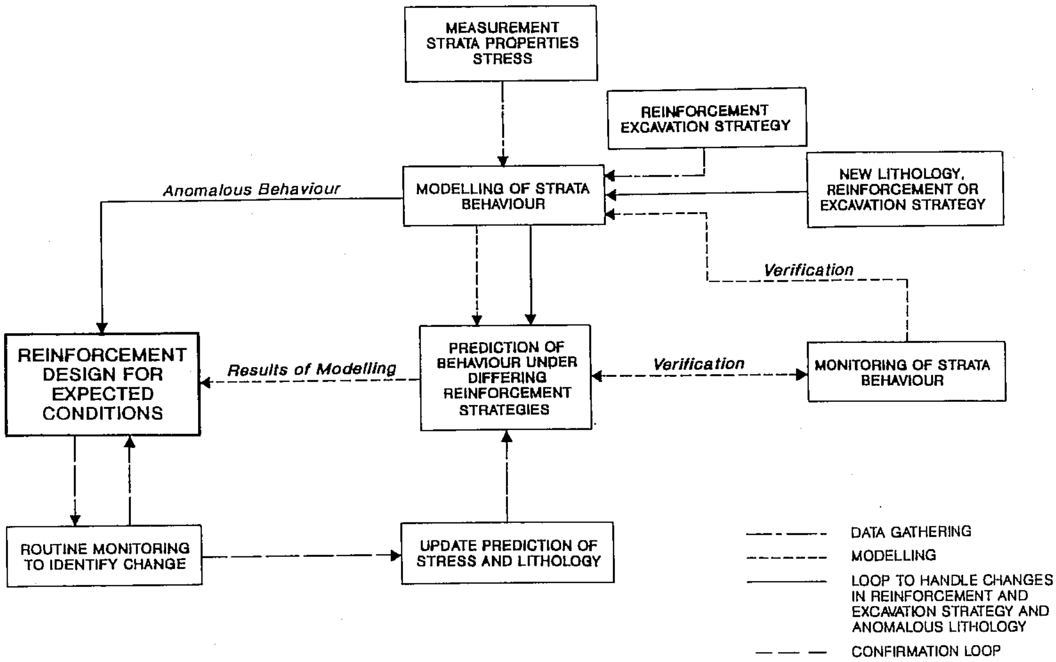


Figure 2. Integration of computational modelling and routine control monitoring.

understanding of the limits of applicability of the various reinforcement strategies under the likely variation in stress and lithological conditions.

Figure 2 illustrates how the output from the validated computational model can then be integrated with a systematic routine monitoring programme which, under pre-determined 'trigger points', can necessitate changes in the reinforcement strategy.

The 'trigger points' can be deformation levels or the commencement of movement in the strata above a certain height into the roof. These 'trigger points' can be determined from the computational modelling as styles or magnitudes of movement which are at, or close to, the limit of the original reinforcement strategy. Similarly, 'trigger points' can then be set at which time the reinforcement can be decreased back to previous levels.

Depending on the results obtained from the modelling, the alternative reinforcement strategy can consist of modification of the density or distribution of bolts at the face of the heading or else a systematic secondary reinforcement placement while the trigger points are exceeded.

ROUTINE MONITORING

The requirements for the routine monitoring can be determined on a site specific basis dependent on:

- The level of expected change,
- The variations in the style of behaviour expected, and
- The criticality or sensitivity of the reinforcement to change.

Where the initial reinforcement pattern has been shown to provide adequate levels of stability over a wide range of expected conditions, the routine monitoring can be limited to widely spaced displacement measurement systems. However, if the initial reinforcement pattern has been shown to be valid for only a limited range of the expected conditions, then the density and nature of the routine monitoring may need to be more comprehensive.

Methods that can be used for routine monitoring include:

- Conventional extensometry,
- Simple dual anchor extensometers or "Tell tales", and
- "Tell tales" capable of being connected to mine wide data acquisition systems.

Conventional Extensometry

If the variability of the expected behaviour is complex in terms of both the magnitude of deformation or height of softening into the roof, then multi-point extensometers can be used. This type of extensometry allows more confidence to be gained in the behavioural characteristics of the roof. This in-turn can allow the refinement of 'trigger points' so that simpler measurement systems can then be used.

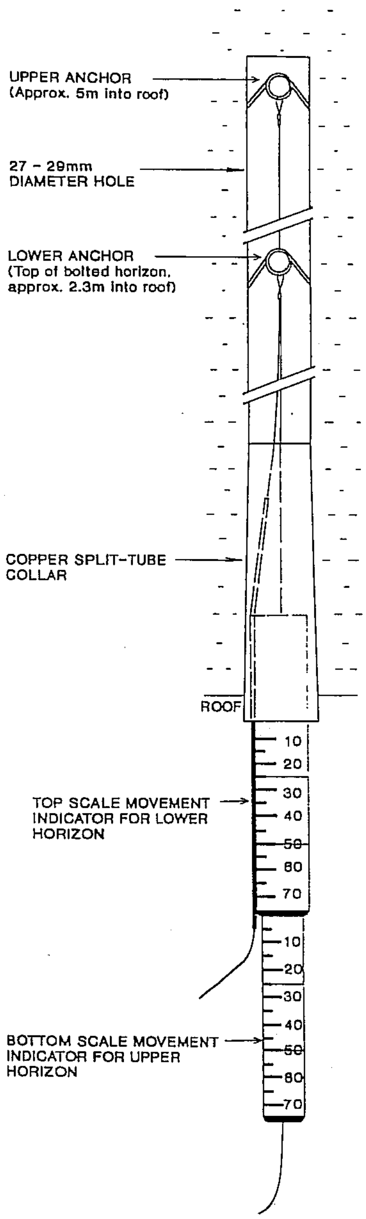


Figure 3. Dual height Tell-tale.

Tell Tales

The 'tell tale' system is based on a simple direct reading dual height extensometer shown in Figure 3. This system allows both total roof deformation and the amount of movement between selected heights in the roof to be determined. The visual display allows a rapid check on the level of movement occurring.

A similar system has in the past been used as an integral part of the reinforcement design in the UK (Siddall & Gale, 1992) and has now been used in a number of Australian mines.

The system is very quick to install and economic, which allows the tell tales to be used frequently (down to a 20m spacing) if justified by the variability in conditions.

The tell tale system can be automated and connected to standard mine wide data gathering systems.

SUMMARY

Computational modelling, when verified with targeted monitoring, has been used to significantly improve the level of confidence in reinforcement design. The computational techniques have been fully developed by SCT and are currently an integral part of the design approach. The modelling allows the assessment of the reinforcement over the range of conditions which are expected over the life of the roadway, as well as stress and lithology variations along its length.

This understanding allows the definition of the limitations of the reinforcement systems used as well as determining the most appropriate response when conditions change.

To fully utilise this knowledge, the results of the modelling can be integrated into a systematic routine monitoring programme which will indicate change in behaviour and allow the rational modification of the reinforcement system where and when it is required.

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Greg Tarrant

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FROM ROOF MOVEMENT TO BOLT LOAD

In the last two to three years the number of support elements (roof bolts and cable bolts) available to the underground coal industry has increased significantly. These have a wide variety of mechanical properties. Alongside the development of the steel hardware there has also been a parallel development of resins and cement grouts to bond these to the strata; again these have a variety of strengths and stiffnesses. A problem often faced by operators is knowing what load can be developed by the bolts when subjected to any given roof deformation pattern and how the roof deformation might that change if either the bolt or the resin is changed.

Measuring roof displacement is relatively simple and inexpensive and is now routine in several mines. Measuring bolt load is much less common and is relatively expensive whilst measuring cable bolt loads is difficult, extremely rare and very expensive. It is known that the way in which a support element accepts load is governed by a number of factors including; bolt stiffness and yield, bond behaviour and plate behaviour. Determining the load in a bolt in a given roof deformation environment requires that all these factors need to be taken into account and because some are interactive, this is a complex process.

To overcome the obstacle created by this complexity; an easy to use, WINDOWS based analysis has been developed which can allow mines to take known properties of bolts and bond behaviour, subject them to measured roof deformation regimes and determine the pattern of bolt loading.

This paper provides an outline of the analysis an example of the output that it generates and how the analysis can be used to predict the effect of a different bolt type on roof deformation.

INTRODUCTION

In Australia the support of roadways in underground coal mines is governed by the Manager's Support Rules. These frequently indicate a blanket pattern of support throughout a mine. However, as conditions vary throughout a mine it is probable that the level of support required should also vary. If conditions are such that more appropriate support elements can be used at lower installed densities to achieve at least the equivalent roof support, it is likely that development rates will increase. Therefore the economic gain is not simply the reduced cost of the roof support system but may also include the much greater gain of improved productivity. It should be emphasised that any variation in support levels must be validated with comprehensive monitoring of strata behaviour. The economic gains will be negated if the reduced support density allows conditions to deteriorate to the extent that either safety is compromised, production is impeded or a roof fall occurs. By monitoring roof displacements and analysing loads in the support elements, it is possible to determine whether there is potential to change the support element and achieve a more cost effective roof support and roadway development strategy.

CURRENT BOLT PERFORMANCE ASSESSMENT

Monitoring of strata and support behaviour normally consists of using an extensometer or other device to measure roof movement and determining bolt loads through the use of strain gauges. Strain gauged roof bolts are commercially available but are expensive and require experienced personnel to record, process and interpret the data. Problems can arise with the electrical connections and instrumented bolts tend to be used in areas where extreme difficulties are being experienced with roof stability. Routine monitoring of support performance with instrumented bolts does not usually occur.

Instrumentation of cable bolts is highly specialised, very expensive and is undertaken extremely rarely. Thus the loading condition of cable bolts is virtually unknown, and since cable bolted roadways can still fail it is evident that a greater knowledge of the loading behaviour of cables is required to improve cable bolting designs.

POTENTIAL TO ANALYSE BOLT PERFORMANCE

The measurement of roof displacement is relatively straightforward and can be (and is) performed routinely. Since support load is induced by strata displacement then the

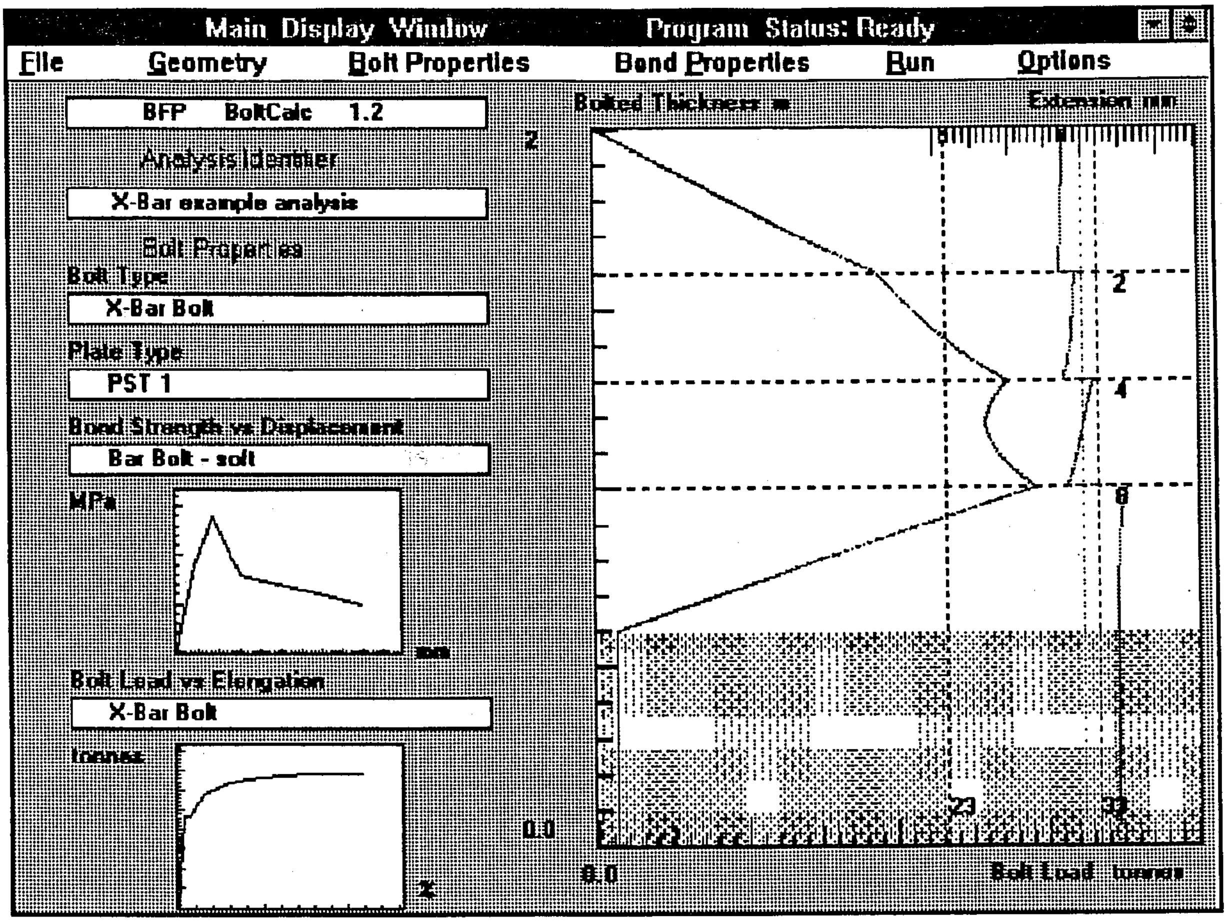

Figure 1. Example of BOLTCALC Output

calculation of bolt load directly from strata displacement is proposed as an appropriate means of assessing support performance. This can then provide the basis for an informed assessment of the need for additional support or the potential to reduce support density. If either the support density or the support type is changed then continued roof movement monitoring can be used to assess conditions and 'trigger' the need for any additional support outbye.

Mines may be able to gain a significant economic improvement by increasing development rates through reduced primary support densities and then using selective outbye bolting or cable bolting to achieve long term stability.

BOLT LOAD FROM ROOF DISPLACEMENT

The relationship between bolt load and vertical roof displacement is a function of bolt type, bolt length, bolt diameter, resin/grout type, resin/grout cure time, plate behaviour, encapsulation length, hole size and geology. These define the bond properties and the load-deformation characteristics of the bolt. Many of the above factors are interactive (i.e. when one factor varies it affects other parameters) which complicates the analysis process.

To overcome this and to be able to consider the calculation of bolt load on a routine basis (on PC based software) BOLTCALC has been developed to analyse the loads developed in both fully and partially bonded support elements in response to rock deformation.

The software has been set up to enable the user to select from a number of support types for which the load-deformation characteristic is known. Any support type such as roof bolts, Flexibolts, cable bolts or friction bolts for which load-deformation data can be determined from tests can be included in the program. The bond characteristics in the current version have been determined from experimental pullout test results. Site specific bond properties should be included when these are available. The program is based on the roof displacement being axial to the bolt and the displacement occurring at discrete positions, ie: separation occurs on a number of planes. The approximate location of each separation plane and the displacement assigned to it can be determined directly from roof displacement measurements recorded from, for instance, a sonic probe extensometer. The program will accept either direct entry of displacement on each separation plane from extensometer data or data can be directly imported from

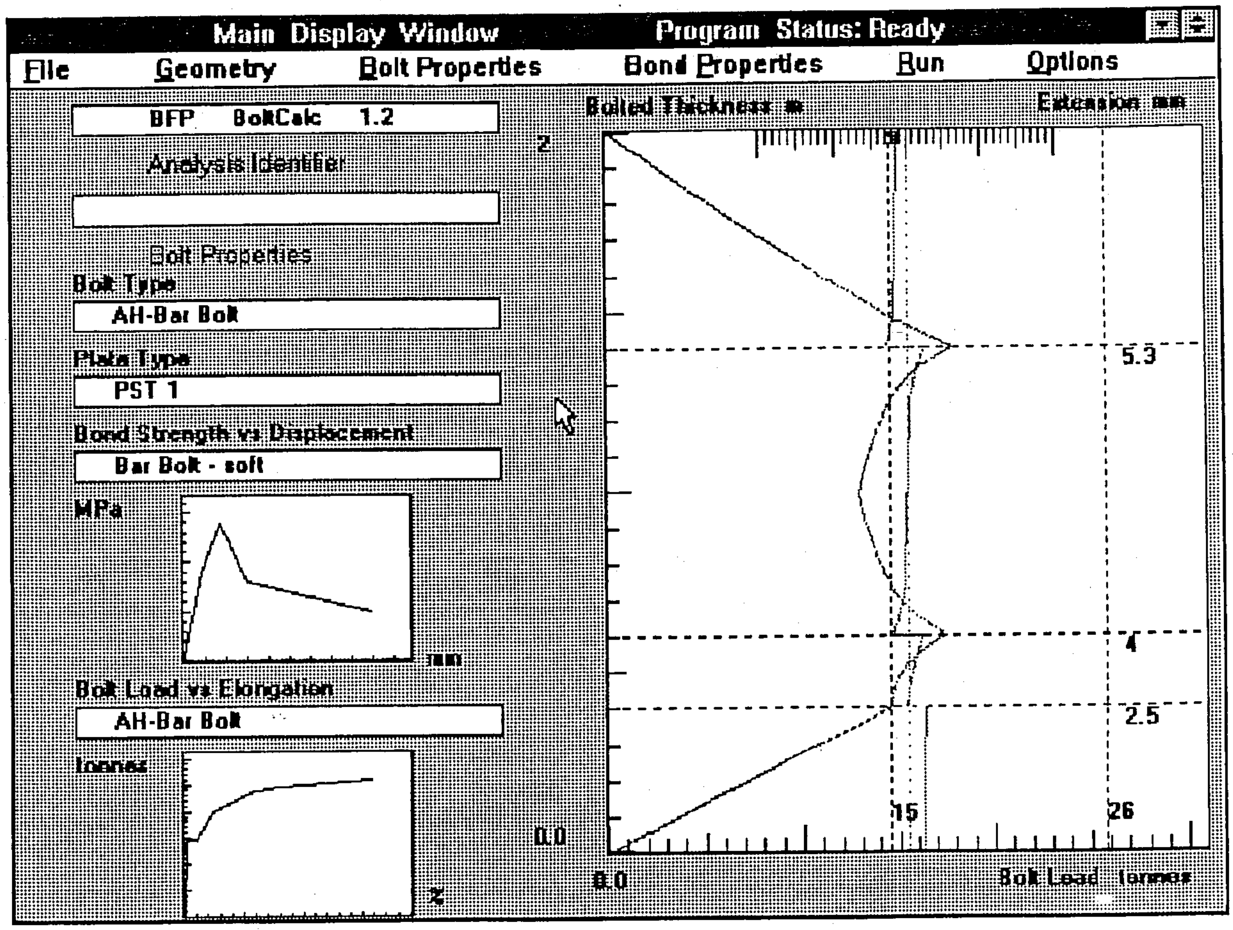

Figure 2. Calculated Bolt from Measured Roof Displacement when 2.1m long fully encapsulated High Strength Bolts were used.

extensometer data files. BOLTCALC then calculates the bolt load and the relative displacement between the rock and the bolt along the bolt length. BOLTCALC accounts for pullout at the top of the bolt and the load and deformation developed at the plate.

In order to be simple to use BOLTCALC has been written to run under Microsoft WINDOWS. It uses pull down menus and windows to create an easy to use environment to set up the bolt properties and geometry and to modify these and re-analyse as required.

The program requires:

•the bond shear strength versus bolt displacement for the relevant ground conditions,

•the load-deformation curve for the bolt,

•the load-deformation curve for the plate, and

•the roof displacement profile along the length of the bolt.

An example of the BOLTCALC output window is illustrated in Figure 1.

It shows the BOLT type being analysed; in this case an X-Bar type bolt. It illustrates the bond strength vs displacement characteristic and the bolt load vs elongation characteristic of the X-Bar bolt. The result output window on the right side is a plot of bolt load along the bolt length and shows the planes (as horizontal dashed lines) on which roof deformation has been assigned. The numbers on the horizontal lines are the deformation in mm from the extensometer measurements for each plane. The shaded area in the lower portion of the window illustrates the portion of the bolt which has been simulated as not being encapsulated. This section of the bolt acts as a free length and is able to elongate accordingly.

The bold vertical dashed lines show the typical yield and failure loads of the X-Bar bolt i.e. 23 and 33 tonnes. The Y axis shows the bolted thickness; in this instance a 2.1m bolt with 100mm of thread protruding from the collar to give a 2m bolted thickness. The X axis shows the load developed on the bolt in tonnes. The continuous line illustrates the calculated load along the bolt for the roof deformation environment shown; i.e. three discreet planes with a total of 14mm of roof displacement within the bolted horizon. The scale in the top right hand corner marked 'Extension mm' illustrates the relative movement (slip) of the bolt with respect to the roof strata. A zero slip line is shown as a faint vertical line just to the left of the ultimate load line for the bolt. Offsets to the left of the zero slip line indicate a downward relative slip of the bolt in the strata whereas right offsets show those portions of

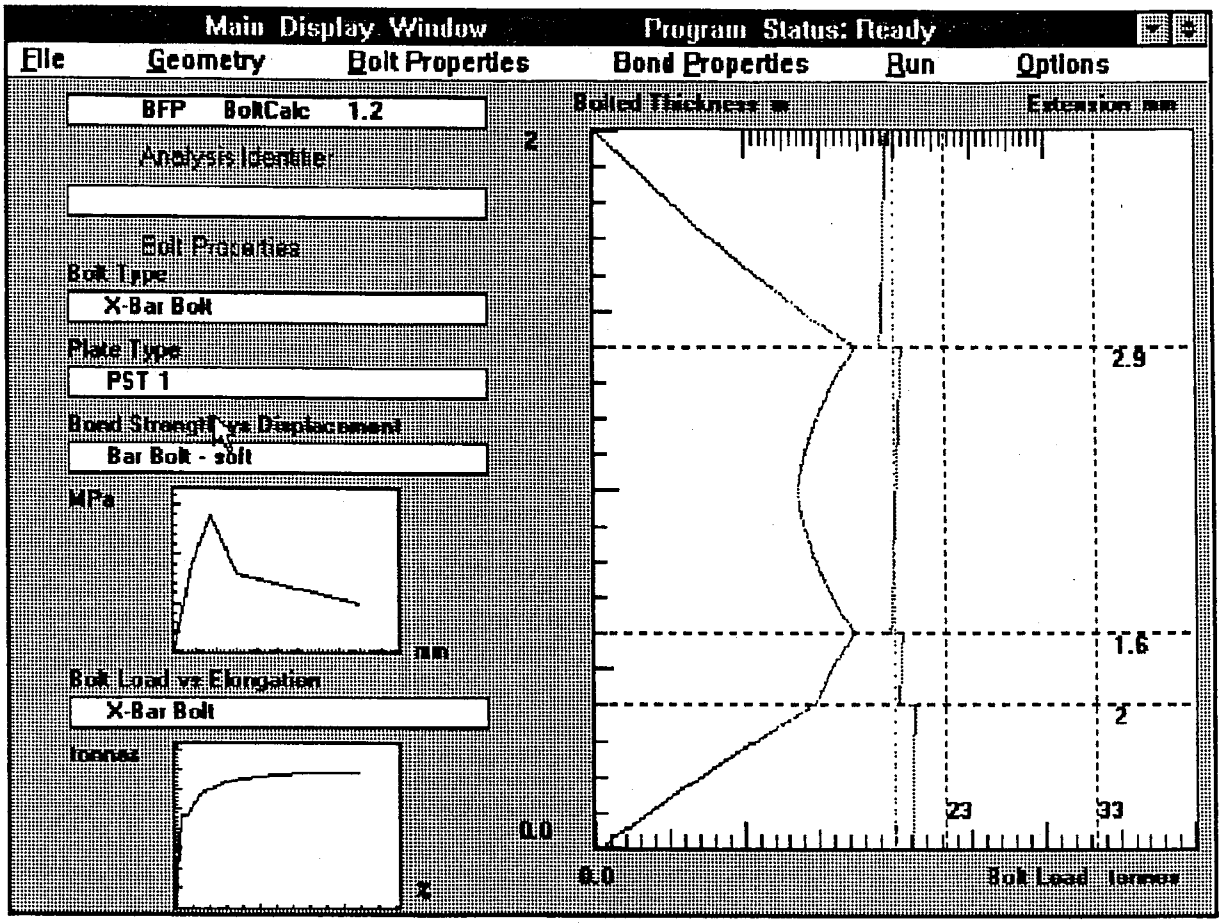

Figure 3. Roof Displacement required to generate Equivalent load in 2.1m long fully encapsulated X-Bar Bolts.

the bolt which have been pulled upwards as a result of the strata movement. This pattern gives some indication of the complexity of the bolt-strata interaction that occurs when strata separation occurs on multiple planes.

The bolt loading line indicates that in this example, the bolt is above yield from approximately 0.9m to about 1.4m into the roof and that a small plate load (about 1.2 tonnes) has developed. The software can be used in a number of ways. It can simply give an indication of the bolt loading pattern resulting from the measured roof deformation profile. Thus as the roof progressively deforms, the user can watch the progressive build up of bolt load and have an indication of whether the bolts are in yield and what length of the bolt is affected. An obvious extension to this is to test the bolts capacity to withstand any increase in roof deformation before reaching a failure condition.

Another use involves changing either the bolt type, bolt length or the encapsulation length and gradually incrementing the roof deformation on each layer to generate a similar bolt load distribution to that calculated for the original bolting system. In this way the program would show the benefits (perhaps as lower roof deformation and tighter roof conditions) that should arise from such changes in bolt parameters.

As an example of this, consider the case shown in Figure 2 in which roof deformation measurements showed a total downward movement of 11.8 mm within the bolted horizon when 2.1m long fully encapsulated high strength bolts were used (in this case type AH). From the pattern of the extensometer output, this could be apportioned between three layers located as shown in Figure 2. It can be seen that the bolt had reached yield at two zones each about 0.2m long. Note also that the upper bonded length has slipped and the upper end has pulled out 1.5mm.

With this analysed result for bolt load, it is relatively straightforward to determine the roof displacement that would result if higher strength X-Bar bolts were used to replace the high strength bolts. This involves selecting the new bolt type from the program menu and simply iterating the displacement values at the same three locations until the same bolt loading pattern is achieved.

The result is shown in Figure 3 where it can be seen that 2, 1.6 and 2.9mm would be needed on the three planes to develop the same bolt load. This implies that the roof would stabilise with a total displacement within the bolted horizon of 6.5mm or only 55% of that which occurred with the high strength bolts.

By increasing the displacements on each plane in Figure 3, it is very easy to check how much further roof displacement would be necessary before the X-Bar bolts would start to yield or reach ready their tensile limit.

Thus, with this approach it is possible to quantify the effect of changing to higher strength bolts and to assess the benefits of reducing the roof deformation which will effectively stiffen the immediate roof. Since each X-Bar bolt is capable of developing more load than shown in Figure 3, another option may be to reduce the number of X-Bar bolts and allow each to take a higher load. However, it must be noted that this will mean that the roof movement within the bolted horizon will probably increase above the 6.5mm value determined in Figure 3.

BOLTCALC has the facility to allow a number of analyses to be performed very quickly and then selected analyses can be examined in finer detail. Alternatively for complex analyses, the bulk of these can be initially carried out in a simplistic manner and then the analysis process an be reset to run in a more detailed manner once the bulk of the calculation has been completed.

A number of future developments are planned to allow the software to handle a greater range of support scenarios.

These include:

- allowing bond properties to vary over the length of the bolt,
- incorporating prestressing on bolts or cable bolts,
- incorporating shear at separation planes in the direction normal to the bolt, and
- calculating progressive bolt load development based on a series of extensometer results taken at various times.

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MONITORING SYSTEMS FOR ROADWAY SUPPORT

A lot of **time and effort** is presently directed to **correcting roadway support** designs that **perform poorly** and **fail to cope with changing geology and loading conditions**.

Production delays affecting development rates could often be avoided if roadway and pillar monitoring results were **routinely available** to help verify and optimise support design.

Why Monitor?

The 'art' of geomechanical engineering is often described as the ability to make rational decisions in the face of imperfect knowledge. Geologic uncertainties are endemic to geotechnical engineering. Inevitably, some degree of residual uncertainty remains related to either having an incomplete knowledge of geologic conditions and material properties through to an imperfect understanding of the processes and mechanics involved of the particular problem.

Roadway support and associated pillar design methodology and processes have been discussed in other papers in these proceedings. A general treatment of the subject has been undertaken by a number of authors (Bieniawski, 1992; Brady & Brown, 1992; Wardle & McNabb, 1991).

The typical design methods can be described as empirical, observational and analytical (ie numerical modelling). The observational method relies on monitoring changes in stress and displacement and is the only approach that can evaluate the results and predictions of the other methods.

Monitoring strata behaviour around roadways and their associated pillar systems is critical in determining the effectiveness of the installed support and for subsequent design optimisation. Parameters such as bolt type and length, pattern and density, timing and installation variables including hole size can be quantified.

The influence of geological structures such as bedding planes and faults on pillar performance can also be assessed. It is evident that geological variation in rock and discontinuity properties can significantly modify the performance of roadway and their associated pillars (McNabb & Wardle, 1986).

In summary, it is important to monitor and determine how the roadway performs from initial roadway formation through to final panel extraction. Monitoring can help 'get the geology right'.

INSTRUMENTATION

The response of roadways to panel extraction loading is directly related to the width and ultimate performance of associated pillars. It is therefore important to monitor not only roadway behaviour, but also pillar stresses and deformation.

The following discussion has been restricted to a few of the more popular monitoring instruments used by the Australian coal industry. It is not an exhaustive review but is simply meant to provide brief details on a few instruments. Typical results produced by these instruments are included in a number of the other papers.

STRESS CHANGE MEASUREMENT

Stress measurement techniques mostly involve inclusion devices that relate the measured borehole deformation to stress via a relevant formula which is usually derived from laboratory calibration.

A large range of instruments have been developed and used with varying degrees of success. It is important to note that all of the stressmeters have inherent advantages and disadvantages. These range in some cases from difficult and precise installation requirements through to unavailable or expensive monitoring systems.

Some devices also suffer from operational problems which require stresses to be inferred at the desired location or simply will not provide data at high(er) stress readings.

The most commonly used stress monitoring device over the last 20 years has been the uniaxial vibrating wire stressmeters produced by companies such as IRAD Gage (now Roctest), Geokon and GSA (Figure 1).

These stressmeters use the vibrating wire principle to measure the distortion of a steel proving ring wedged into an EX (38 mm)

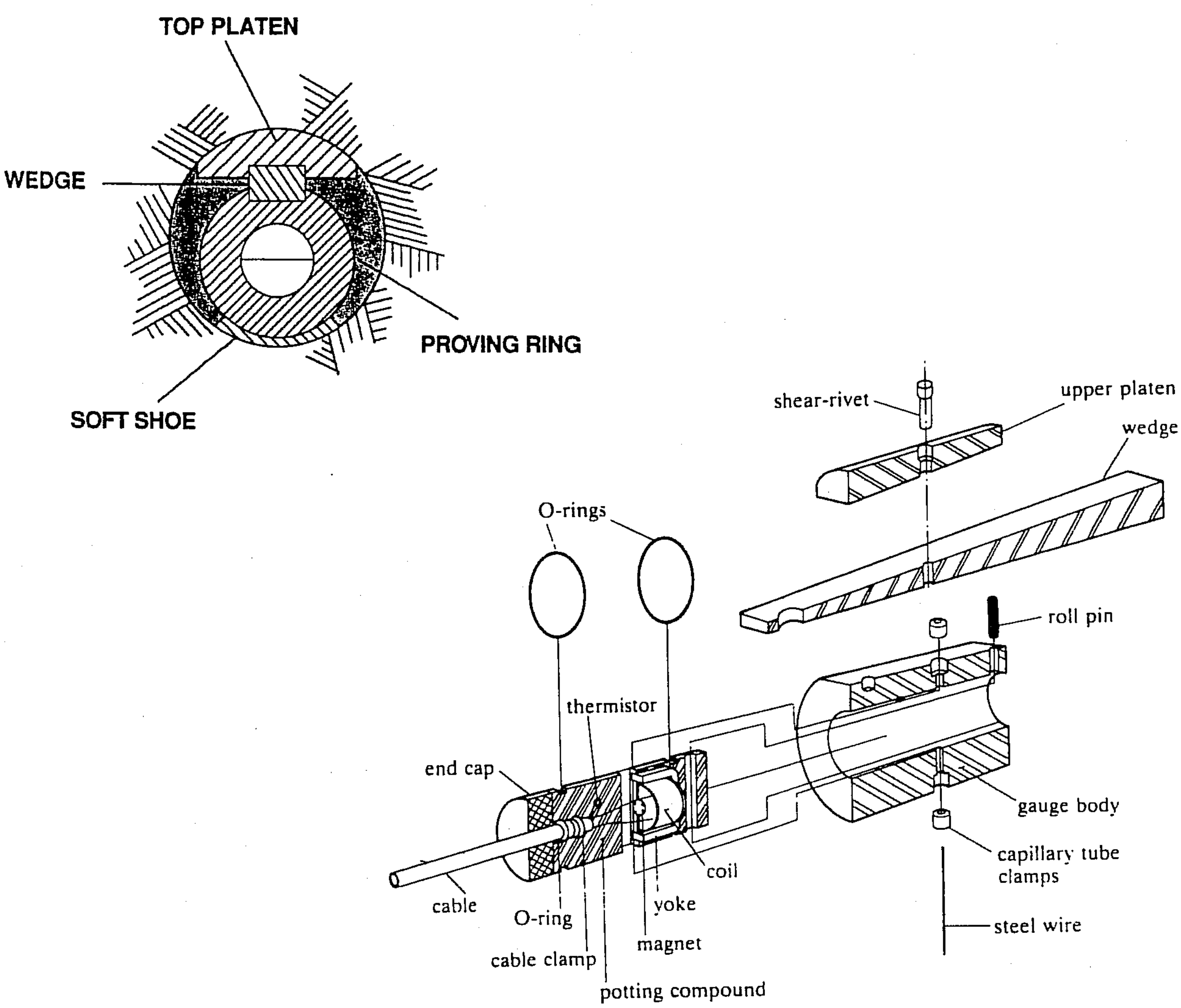

Figure 1. Exploded view of a vibrating wire stressmeter.

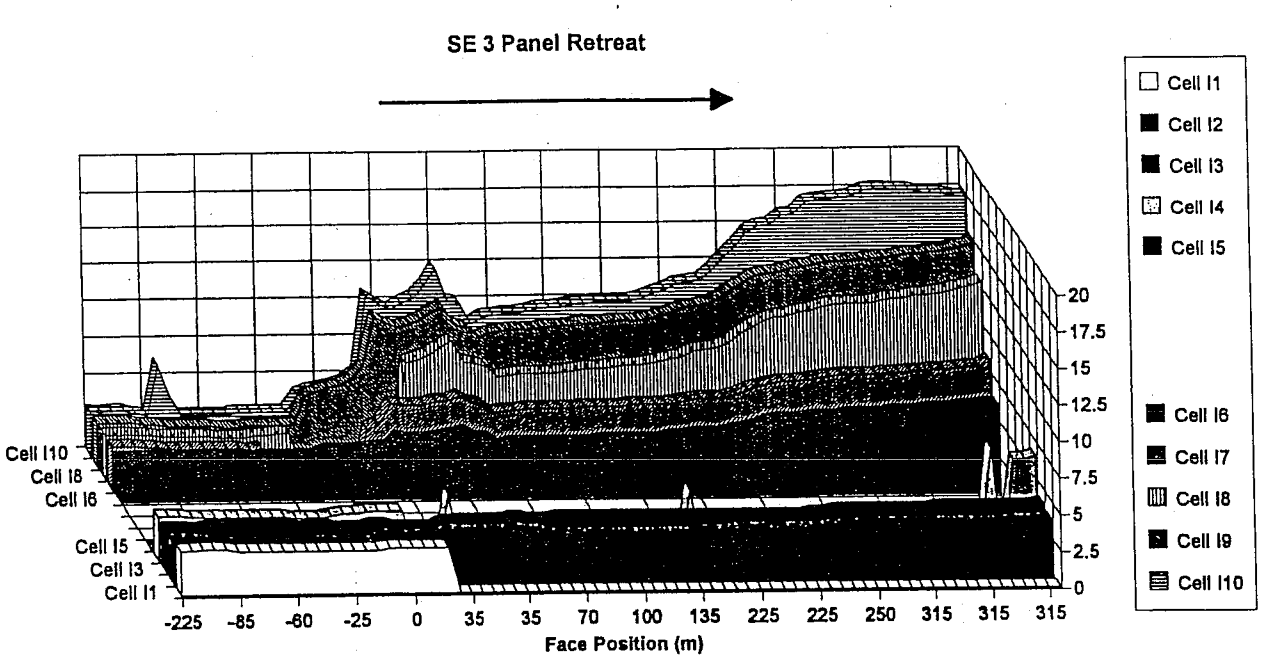

Figure 2. Vertical stress changes relative to SE 3 Panel face position (quasi 3-D presentation looking towards SE 3 Panel).

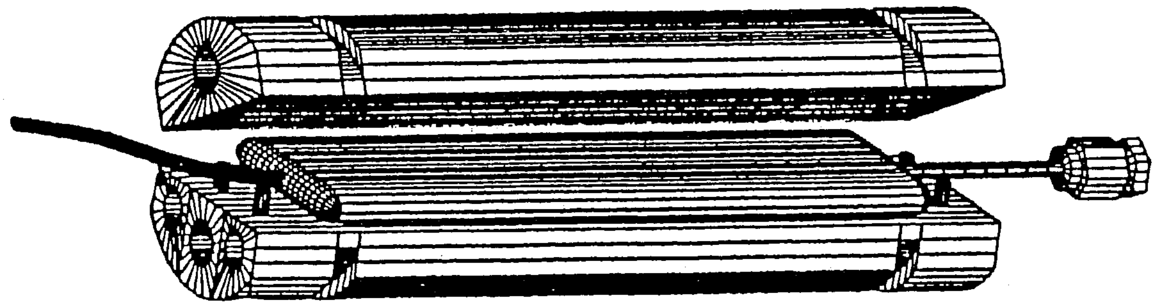

Figure 3. Schematic of hydraulic pressure cell showing flatjack and platens.

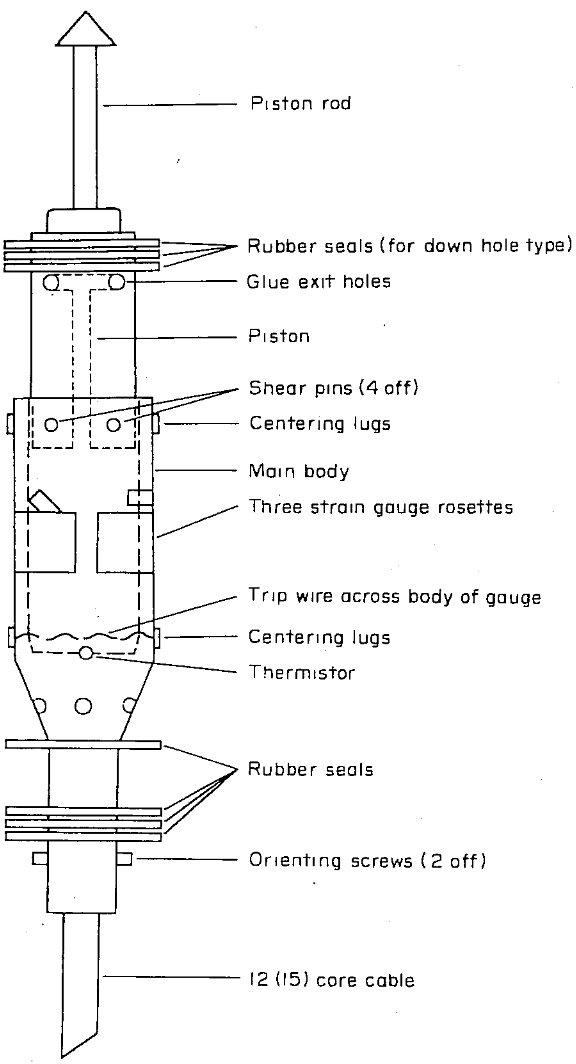

Figure 4. Schematic layout of CSIRO HI stress cell.

borehole (Hawkes & Bailey, 1973). They are monitored with a specially developed intrinsically safe readout unit or logger.

Figure 2 shows an example of the use of vibrating wire stressmeters at the Wallarah Colliery in NSW (Edwards & others, 1993). The figure shows the stress change versus face position for stressmeters installed in two narrow chain pillars. The results show the effect of abutment loading, goaf falls and a stress increase produced by overlying panel extraction.

Hydraulic pressure cells have been used on and off for the last 20 years with varying degrees of success. In the last 2 years they have been reintroduced through a new design developed under an AMIRA managed research project and also by ACIRL with an alternative design developed by ACIRL (Riley & Edwards, 1989).

Figure 3 is a schematic of the hydraulic pressure cell being developed under the AMIRA Project 396. These cells measure stress changes directly by relating external rock and internal flatjack pressures (Heasley, 1989). This design consists of a stainless steel flatjack and two steel platens that are installed in a 52mm borehole. Hydraulic tubing extends out of the borehole to an appropriate pressure gauge or transducer. Local readings are obtained with a Bourdon gauge and remote monitoring is available via intrinsically safe pressure transducers and loggers.

This device is being developed to specifically overcome a number of limitations of the alternative stressmeters. The stressmeter is easy to install and monitor and does not require any specialist installation equipment or training. Prototypes have been successfully installed and monitored at a number of collieries.

The CSIRO developed Hollow Inclusion triaxial stress measurement cell (marketed by Mindata) is typically used to monitor stress changes above pillars in competent strata in the immediate roof (Worotnicki & Walton, 1976). Use of CSIRO HI cells for monitoring involves bonding the hollow plastic tube into an EX borehole with epoxy cement and usually requires installation in competent strata above pillars. Figure 4 shows the schematic layout of a CSIRO HI stress cell.

A number of relatively new techniques are currently being introduced such as the

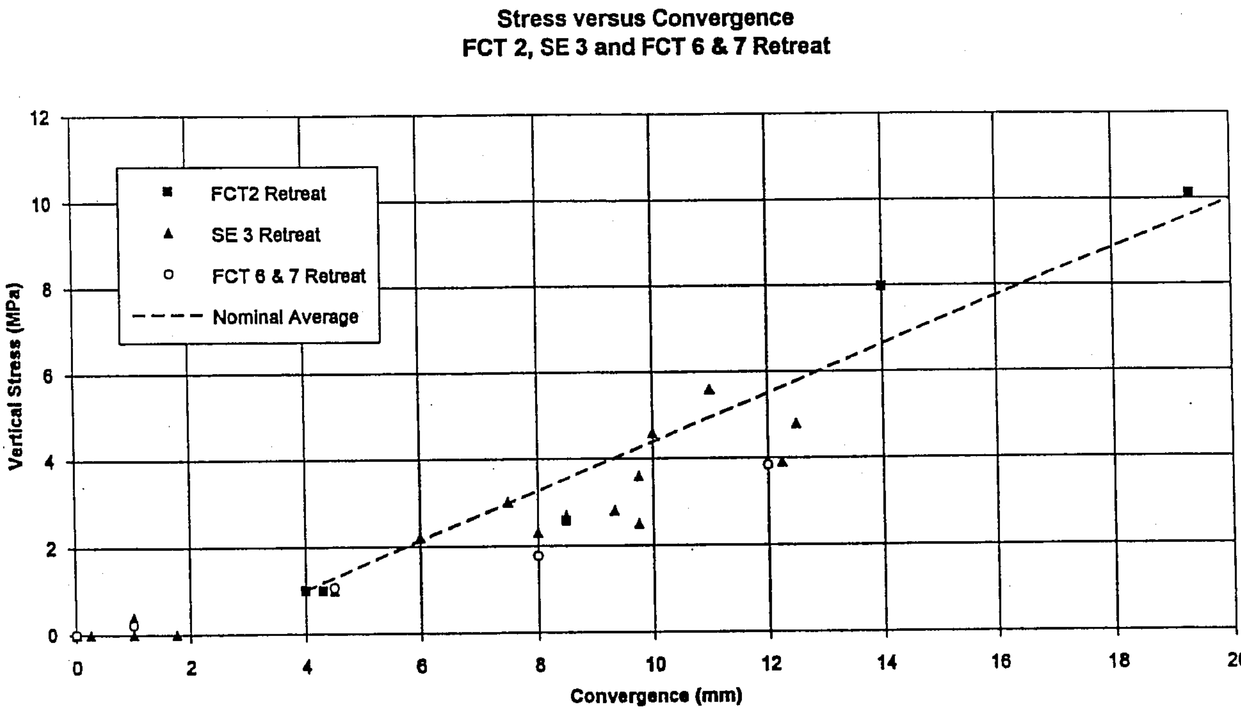

Figure 5. Monitored pillar vertical stress versus convergence for one seam.

hydraulic fracturing approach as used by the CSIRO developed Minifrac system. The CSIRO developed Yoke gauge (marketed by Mindata) has also been used successfully in coal mines.

In summary, stress changes around roadways and within pillars can be monitored via a variety of instruments. The current trend is towards simplifying and making the technology more available for routine measurement.

DISPLACEMENT CHANGE MEASUREMENT

Strata deformation around roadways is monitored using borehole extensometers.

Various multi-point borehole extensometers have been developed to monitor roof sag and bed separation between anchorages. Monitoring provides specific data on the height, location and degree of rock failure or softening. Similar designs are also used to monitor pillar dilation by installing extensometers at midseam.

Typical systems use wires running over linear potentiometers to measure relative movements between borehole anchors and the instrument head. The wires are kept in constant wire tension by using hanging weights.

Alternative roof monitoring devices include the Irad Multi-Point Sonic Probe system. The Sonic Probe basically measures the distance between magnets anchored along a borehole.

Roadway (and pillar) closure can be measured using various rod-type extensometers installed between convergence stations. Typically bolts are installed in the roof and floor.

Convergence meters usually consist of spring-loaded telescoping tubes containing an enclosed potentiometric transducer and are inserted between roof and floor anchor points. Figure 5 shows a combination plot of monitored pillar stress versus closure which can be used to determine the *in situ* pillar stiffness.

BOLT LOAD MEASUREMENT

The performance of rock bolts can be assessed by using strain gauges fixed along their length and monitored using a readout unit. Typically, 9 gauges are placed on either side to measure axial and bending strain. In addition to axial load development, the extra shear strength provided by bolts can also be measured. Cable bolts can also be strain gauged using a number of different approaches. (Results from rock and cable bolt monitoring are covered in other papers).

It is also possible to directly measure changes in roof bolt collar loads over time with specially designed load cells. This allows the success of full column grouting to be determined by seeing if collar load increases as ongoing ground deformation occurs.

CONCLUSIONS

Monitoring should be seen as an essential component of an integrated geotechnical program as it provides input data and tests the validity and effectiveness of reinforcement designs.

Monitoring is a diagnostic tool that helps explain why a support design may not be working.

Monitoring can assess roadway performance and reinforcement design by quantifying initial geologic site characteristics, stress field changes and strata deformation and residual rock mass properties.

It should be noted that monitoring programs need to be properly designed so as to provide appropriate data for the design task at hand.

The integrated use of a number of monitoring methods can produce better results and allow both global and local mechanisms to be identified.

Experience has shown that monitoring should not be used in isolation. Numerical models can be used help analyse results and allow new designs to be developed. The models can be used to determine various parameters such as optimum support mix and location.

The current trend is towards simplifying and making the technology more available for routine measurement. This involves developing cheap, reliable and easy to use instruments such as the hydraulic stressmeter.

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ROOFBOLT PRE-TENSIONING AND ITS EFFECT ON ROOF STABILITY

Roofbolt pre-tensioning is not a new concept. It has been discussed in many texts around the world and is routinely used in tunnelling as an important facet of rock stabilisation. Indeed, roof bolts installed within the Australian coal industry have always been pre-stressed to some degree (albeit at a relatively low level) according to the bolting machine used (hand-held or miner-mounted), with the almost universal use of two speed resins (fast and slow) implying that bolt pre-loading along as much of the bolt length as possible is a pre-requisite for an effective bolting system.

The aim of ACIRL's recent work in this area has been two-fold:

(i) to develop a safe method of applying higher bolt pre-loads **without** impacting significantly upon the operation of the C.M.

(ii) to conduct a series of controlled field trials to assess whether the application of higher pre-loads changes roof behaviour (visibly and through extensometry) and if so, whether the changes are beneficial or not.

If it can be demonstrated that each roofbolt installed 'can be made more effective' through pre-tensioning, the impetus for pursuing this line of investigation relies wholly on the potential benefits available to the coal operators. Though these benefits are highly variable and site specific, they can be broadly divided into two key areas:

(i) less roofbolts being used to gain a similar (if not better) level of roof stability — this clearly impacts strongly on roadway development rates whereby bolting time is the second major delay after machine availability.

(ii) far better roof stability being achieved with the same number of roofbolts, which should minimise the need for secondary support or in worst case situations allow the C.M. to operate independently of any secondary support installation.

This extended abstract details quite concisely the results and findings of ACIRL's recent work. The funding and support of ACARP and various collieries is gratefully acknowledged.

PHYSICS OF PRE-LOAD APPLICATION

Various methods of applying high bolt pre-loads have been used in the past. The two obvious ones are:

(i) torque multipliers.

(ii) hydraulic tensioners.

Both methods are cumbersome, while the former has safety and reliability problems and the latter is subject to 'frictional bleed-off', also making it very unreliable.

It was therefore apparent that neither method could be readily incorporated into a high production C.M. unit and consequently, another technique was sought.

The result of ACIRL's research was the incorporation of a thrust bearing between the drive nut and dome washer of the bolt head assembly (see **Figure 1**). The bearing minimises the friction between the nut and the washer, thereby enabling the torque from the drill rig to be more efficiently converted into increased bolt load. This easily covers the first requirement of minimal impact upon the operation of the C.M.. However, in order to put into context the mode of operation of the thrust bearing, as compared to say teflon friction washers, a more detailed description is warranted.

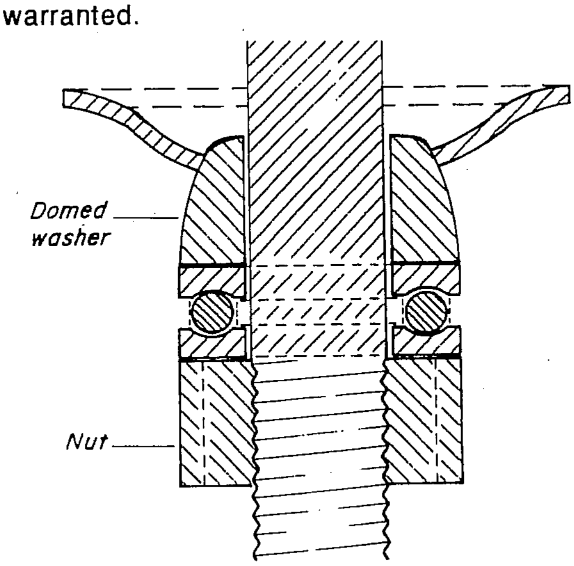


Figure 1. Schematic of Bolt Head Assembly and Thrust Bearing.

The use of the thrust bearing cannot simply be viewed as a more efficient friction washer as the complete rock and bolt system is of direct relevance to the increased efficiency of the bearing.

The application of the bearing can be compared to the action of a lubricant in any system where moving surfaces slide against each other. Consequently, the performance of the moving system is directly controlled by the coefficient of friction of the bearing. This is in turn controlled by:

(i) the contact pressures involved

(ii) the speed of relative movement

(iii) the distances being moved

The associated generation of heat is believed to have a key effect on the coefficient of friction and in extreme cases can cause rotating surfaces to fuse together (i.e. seize up). Under this scenario, the benefits of the bearing become clear and situations whereby it has offered no 'apparent' benefit are explainable.

Looking at the conditions relevant to the rotating nut/dome washer interface, it is clear that the contact pressures involved are very high (17 000 psi at 5t loading) and the speeds are relatively fast (at 150rpm). These conditions will clearly put a large demand on any friction washer, especially when one considers the number of nut turns required to generate a given pre-load.

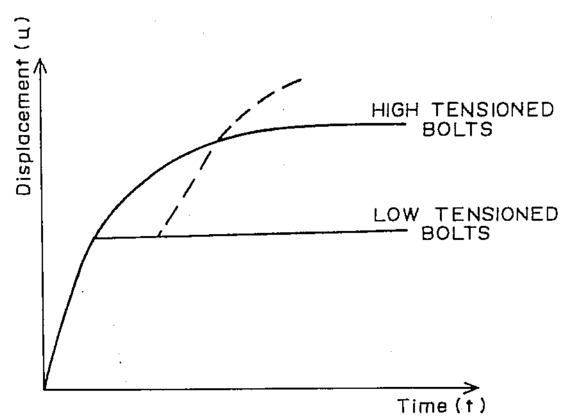
In a roof that is competent (i.e. contains no bed separation as the bolts are being installed), the nut only needs to rotate 0.5 - 1 turn (once the face plate makes contact with the rock) in order to generate >5t load. Compared to a situation whereby 30mm of bed separation exists at the point of bolt installation, for a 3mm pitch thread, 10 turns of the nut under appreciable loading are required to close up the existing de-lamination (i.e. to make a competent roof). Clearly, the heat generated in these two conditions will differ greatly.

The general finding is that for competent roof conditions, a teflon washer or thrust bearing will give a similar pre-loading result, as a low level of heat is generated and the teflon washer will generally remain intact during the tightening process. It is in de-laminating roof conditions, where significant heat is generated, that the thrust bearing has an obvious advantage as the unit is not sensitive to the build up of excessive heat and hence, maintains its ability to function as a friction washer over an increased range of loading. This results in higher bolt pre-loads (measurements and observations substantiate this) which as will be shown, significantly changes roof behaviour in de-laminating conditions.

EFFECT OF INCREASED PRE-TENSION ON ROOF BEHAVIOUR

Field trials have been conducted at Angus Place, Gordonstone, Tower and Teralba Collieries. The first two trials at Angus Place and Gordonstone did not show any 'obvious' changes in roof behaviour following the application of higher pre-tensions, whereas the trials at Tower and Teralba showed the same clear change in roof behaviour. When put into context, the nature of the change has been defined and the reason behind the apparent lack of change in the first two trials has become evident, however, that is not to say that higher pre-tensions are not applicable to the first two trials.

The essential difference in roof behaviour at both the Tower and Teralba trials when higher bolt pre-loads were applied is shown in the time dependent - total roof displacement plot (see Figure 2). In the case of the lower pre-loading, the time dependent - displacement plot shows a very distinct change in trend, whereas the pre-tensioned bolt plot exhibits a very smooth time dependent deceleration in displacement.



- - - Possible unpredictable path of low tensioned bolts

Figure 2. Time dependent displacement plots for low and high tensioned bolts

If one accepts the scenario depicted in Figure 3a, whereby horizontal stress causes a thin beam of material to buckle downwards, then the only plausible explanation for the almost instantaneous cessation of roof displacement is that the driving force (i.e. horizontal stress) has been rapidly dissipated. Considering the physics of the system, it is believed that the most likely mechanism behind such a phenomenon is the generation of a low angle shear plane across the beam such that rapid horizontal stress relief occurs. The rapid and frequent shearing off of a number of exto. holes in areas of low tensioned bolting is clear evidence of such a mechanism. Interestingly enough, the low angle shear plane has long

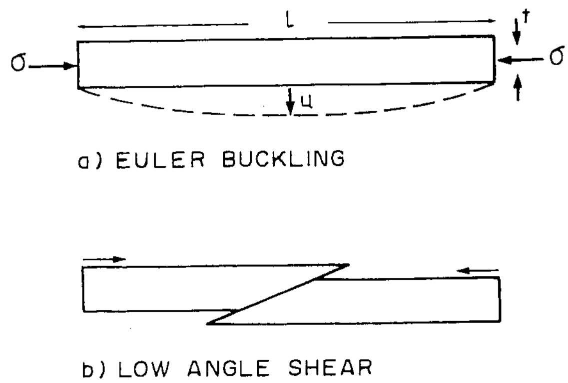


Figure 3. Fundamentals of beam behaviour.

been recognised as a phenomenon accompanying roof instability (Gale, 1986). Conversely, in areas of higher pre-tensioned bolts, the low angle shear has been limited and the whole system displaced as a buckling beam with the horizontal stress largely retained in the immediate roof.

A general back analysis of the trial data (using standard Euler Buckling theory) has shown that the measured displacements of the roof beams compare well to the predicted displacements when estimates of roof modulus and stress magnitude are used in conjunction with the measured beam length and thickness. Roof behaviour conforming to a standard analytical beam theory will therefore offer a significant avenue for an improvement in both support design and roof management.

Where the immediate roof is characterised by a low angle shear, long term roof behaviour is often very unpredictable (movement can start again for no readily apparent reason — see Figure 2) and is very sensitive to any stress changes, for example those caused by an approaching longwall. By contrast, when beam action is retained, roof behaviour has been seen to be far more predictable both during first workings and longwall retreat.

It is therefore suggested that once beam action is lost, roof stability relies primarily on the interaction of discrete pieces of sheared roof material (assuming no secondary support is installed) such that its behaviour becomes far harder to predict and is also increasingly sensitive to issues such as joint frequency and weathering etc. This obviously makes management of roof stability far more difficult when subtle features such as these become increasingly significant.

The application of a higher pre-tensions also altered the geometry of roof sagging. Figure 4 shows three distinct patterns that were observed according to either the pre-tensioning of the bolts or the location of lower tensioned bolts across the roadway. It is clear that with lower pre-tensioning the roof often sags from rib to rib (even with a fairly uneven spread of bolts), whereas higher pre-tensioned bolts clamp the roof together confining the sag to

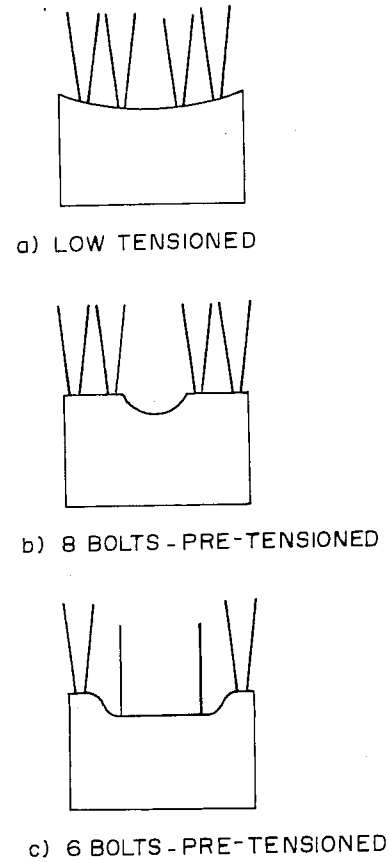
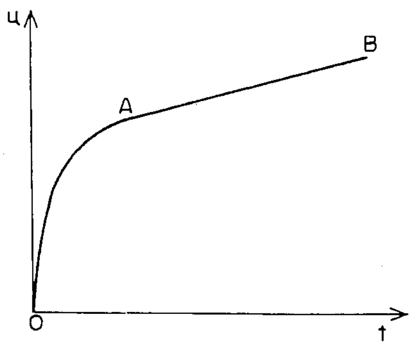


Figure 4. Various geometries of roof sag.

areas of lower bolting density eg, the centre of the roof. This action reduces the effective roadway width and has clear stability benefits through the generation of very stable flanks with low levels of roof softening concentrated in the centre of the roof.

The trials at Gordonstone and Angus Place showed that beam action was already in place with the lower tensioned bolts and that no change in roof behaviour occurred following the application of higher pre-tensioned bolts. This suggests that, providing the bolting/loading density can not reduce the inherent beam length (as the pre-tensioned bolting density over the roadway flanks was reduced in Gordonstone and the pre-loads achieved from the hand-held bolters at Angus Place were considerably lower), it is the material properties of that beam which will control its behaviour. This leads into two other areas of discussion; weak roof material and guttering.

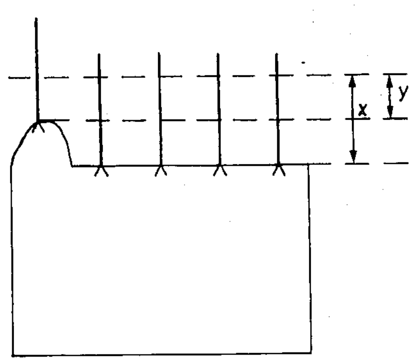
Two of the trials, which both had a weak immediate roof (coal or weak sandstone) showed a tendency for the roof to creep (i.e. displace at a low, but fairly constant rate) beyond the point at which it was apparent that the immediate roof was exhibiting beam action (Figure 5, point A). If one accepts that the horizontal stress has been maintained in the immediate roof, then even though a beam is present, if the strength of that beam in compression is less than the induced horizontal stress level, then one would expect the roof to



O- A Beam action
A - B Material creep

Figure 5. Time dependent displacement plot showing beam creep.

creep. Certainly, both the trial sites referred to, conformed to the basic criterion that the UCS of the roof < induced horizontal stress level and both consequently demonstrated evidence of long-term roof creep. The control of roof creep would therefore appear to be an issue of secondary support (extra bolts will not overcome a deficiency in material strength), as the suspension action of additional cable/flexi bolts would reduce any gradual deterioration in beam stability.



x - Maximum induced beam thickness
y - Effective beam thickness reduced by guttering

Figure 6. Effect of roof guttering on beam thickness.

The significance of a roof gutter can be seen with reference to Figure 6, where the height of the gutter failure clearly limits the thickness of an induced roof beam. Therefore, as the displacement of a buckling roof is very sensitive to the thickness of the beam one would expect the measured displacements to be largely controlled by the development of any guttering. This has proven to be the case at two of the trials (Angus Place and Teralba) where a significant roof gutter corresponded to an increase in roof displacement (150–200mm) on first workings with pre-tensioned bolts even though beam action was indicated in the time dependent displacement plots and a relatively thick roof beam shown in the exto strain plots. In the case of Angus Place, the gutter was associated with the roadway's intersection with a strike slip fault and an intense area of fault gouge which subsequently fell out of the roof immediately upon exposure by the C.M. In Teralba, it is believed that the slow progress of the C.M. during the first half of the pre-tensioned bolt driveage exacerbated the effect of the stresses ahead of the face making the guttering more severe and harder to prevent by bolting. When the speed of the C.M. picked up during the second half of the Teralba trial, the gutter was removed by the action of the pre-tensioned bolts and the measured roof displacements reduced markedly (Figure 7).

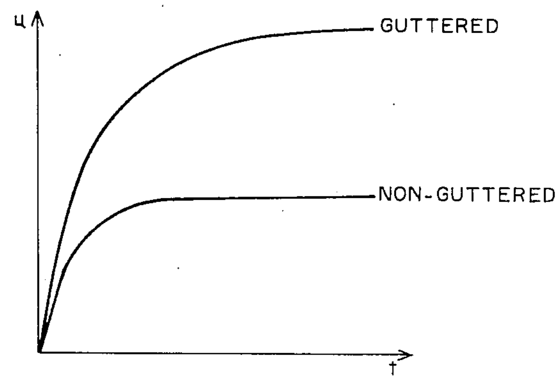
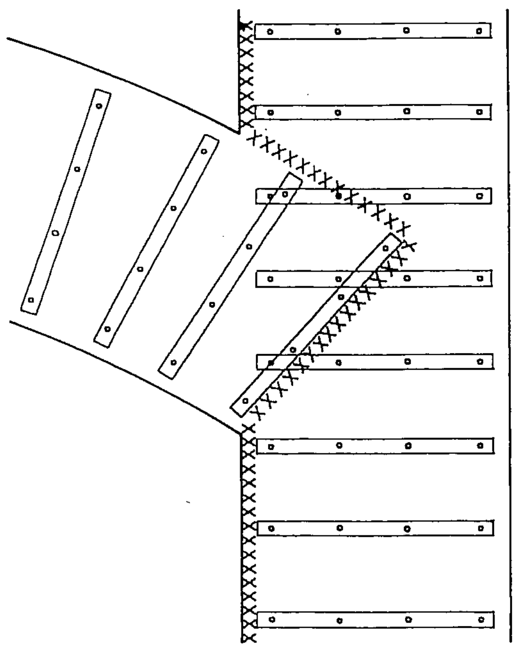


Figure 7. Effect of roof guttering on measured roof displacements.

Furthermore, at both Tower and Teralba the actual location of stress controlled roof guttering was altered dramatically by either the use of more low tensioned roofbolts on the guttered side of the roadway (Figure 8) or by pre-tensioning the standard number of bolts. This phenomenon is obviously related to the level of reinforcement applied to a given area of roof, as it is quite clear that by increasing the effectiveness of the bolting system (through either bolting density in a standard bolting system or loading density in a pre-tensioned bolting system) the common generation of the guttering over the roadway flanks has been constrained and transferred to an adjacent part of the roof where the bolting/loading density is lower.

Returning to the fact that both the low and high tensioned bolting at Angus Place and Gordonstone did not alter the inherent behaviour of the roof beam, it is worth re-iterating the fact that for the Tower and Teralba trials, by pre-tensioning to a higher level the same number of bolts/metre, the breaking down of beam action through low angle shearing was prevented and beam action was maintained. Consequently, where roof beams are inherent in the roof, if each bolt has



xxxx Roof gutter

Figure 8. Moving of roof gutter by additional bolts.

a higher level of pre-tension, it can be argued that beam action can also be strengthened by reducing the beam length or even maintained by the use of less bolts. The latter has yet to be proven through trialing, but if correct will be of significant benefit in improving roadway driveage rates.

OVERALL SUMMARY

It would appear that the level of bolt pre-tension at installation does indeed change roof behaviour on first workings. In particular, if beam action is being lost using standard bolting, it can be maintained if the level of reinforcement is increased (inferred from exto results) either by using more bolts or higher levels of pre-tension for the same number of bolts. From visual observations, guttering can be removed to a large extent by the application of higher pre-loads and, if the bolts are biased to the flanks of the roadway due to the rigs being fixed on the C.M., roof sag will only generate where there is a lower level of bolting density in the centre portion of the roof (i.e. the effective roadway width has been decreased).

It has also been seen that structural anomalies and a slow moving C.M. can make it harder for bolts to prevent guttering, while weak roof materials in relation to the induced levels of horizontal stress can cause long-term roof creep regardless of the presence beam action.

An unexpected aspect of the trials (and as yet not fully proven) is the close conforming of measured roof displacements where beam action is present to those predicted by the Euler Buckling Beam theory. If this can be better established through *in situ* and laboratory testing, there are two very significant benefits to roof control:

(i) a reasonable first approximation prediction or back check of roof stability can be made using a very simple and proven equation.

(ii) more importantly, if one looks at Euler Buckling it is clear that roof stability is primarily determined by beam geometrical factors (i.e. width and thickness) rather than its material properties and loading condition (modulus and stress level). This changes the emphasis of roof stability from rock mechanics to operational factors (width control and beam building by bolting) which are far more amenable to management and control.

The benefits to the coal industry of these two aspects as well as others detailed earlier are quite clear and ACIRL intends conducting larger scale trials over the next 12 to 18 months in order to better validate the comments made here and bring these benefits to fruition. A more detailed account of the trial results and interpretations is given in ACIRL's report to ACARP relating to the industry funded trials.

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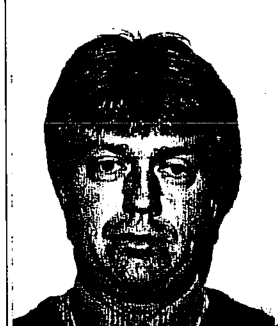

Russell Frith

Rod Thomas

Bill Lawrence

Gordonstone Coal Management.

ASSESSMENT AND MONITORING OF GATEROAD ROOF SUPPORT AT GORDONSTONE

Gordonstone has experienced maingate roof falls during extraction of its first two longwalls. The presence of weak roof combined with horizontal stress have been identified as the prime factors contributing to the maingate falls. The geological extent, geotechnical implications and support of this weak lithology had previously been derived from geophysical logging of surface exploration holes and underground roof condition/structural mapping. Observations and measurements taken during initial longwalling have indicated that the extent and behaviour of this weak lithology have been significantly underestimated.

Subsequent work has concentrated on better geological and geotechnical definition of current and proposed gateroad roof strata. The need to incorporate the assessment and placement of secondary support into the gateroad development cycle has been recognised. Tools required to achieve this objective need to be developed to a practical level.

GEOLOGICAL AND GEOTECHNICAL ENVIRONMENT

In the northern workings at Gordonstone the immediate roof can be broadly characterised as interlaminated and interbedded sandstones and siltstones. There are very few thickly bedded or massive units. The sandstones are typically fine to medium grained with a high clay matrix content, and micaceous bedding planes. Broad lithology variations across the workings due to changing depositional environments have been recognised from surface exploration drilling.

Numerous rock property tests from surface borehole cores have highlighted the weak nature of Gordonstone rocks and provided a consistent correlation with surface geophysical logs. Rock strengths are in the order of 5 to 40MPa unconfined compressive strength. Premining horizontal stresses in the range 10 to 15MPa are typical of the region. However, the horizontal stresses are high compared to the rock strengths. The Gordonstone area is relatively undisturbed, with respect to size and frequency of faults.

Initial geotechnical assessments, based on surface exploration drilling and underground experience, indicated the northern extent of the workings as the 'weakest' most interlaminated stratigraphy grading to stronger predominantly sandstone towards the South.

PAST GATEROAD PERFORMANCE

Up to the present time, Gordonstone has extracted two longwall panels and is progressing through a third. A maingate fall interrupting production occurred on both LW101 and LW102 (Figure 1). Gateroad secondary support densities, consisting of cablebolts and angled ribside roof bolts are also detailed on Figure 1. For LW101, gateroad (all maingate intersections were supported) secondary support was only installed after the fall, while for LW102 it was installed before extraction, except for some additional support after the fall.

The fall locations were not expected relative to the initial geotechnical assessment. Substantial geotechnical monitoring has been conducted on both LW101 and LW102. Primarily, the monitoring has consisted of sonic probe roof extensometer measurements and both *in situ* stress and stress change measurements.

LW102/103 GEOLOGICAL AND GEOTECHNICAL INVESTIGATION

In response to the maingate falls in the first two longwalls, a more rigorous and detailed geological/geotechnical investigation was required. The first priority was to assess the geological variability along the gateroads. An inseam drilling program was instigated along MG102 and MG103 where roof and floor cores were taken. Initially, hole spacing was expected to be in the order of 100 to 200m (one to two pillars). Unexpected and extreme geological variation resulted in significant infill coring, closing hole spacing down to 50m in places. The small scale representation of geological variability shown in Figure 2 does not adequately show the full picture. However, it does indicate that there is no recognisable correlation between stratigraphic units.

Strata Control Technology is engaged to conduct a program of rock property testing on the inseam cores. A geotechnical assessment

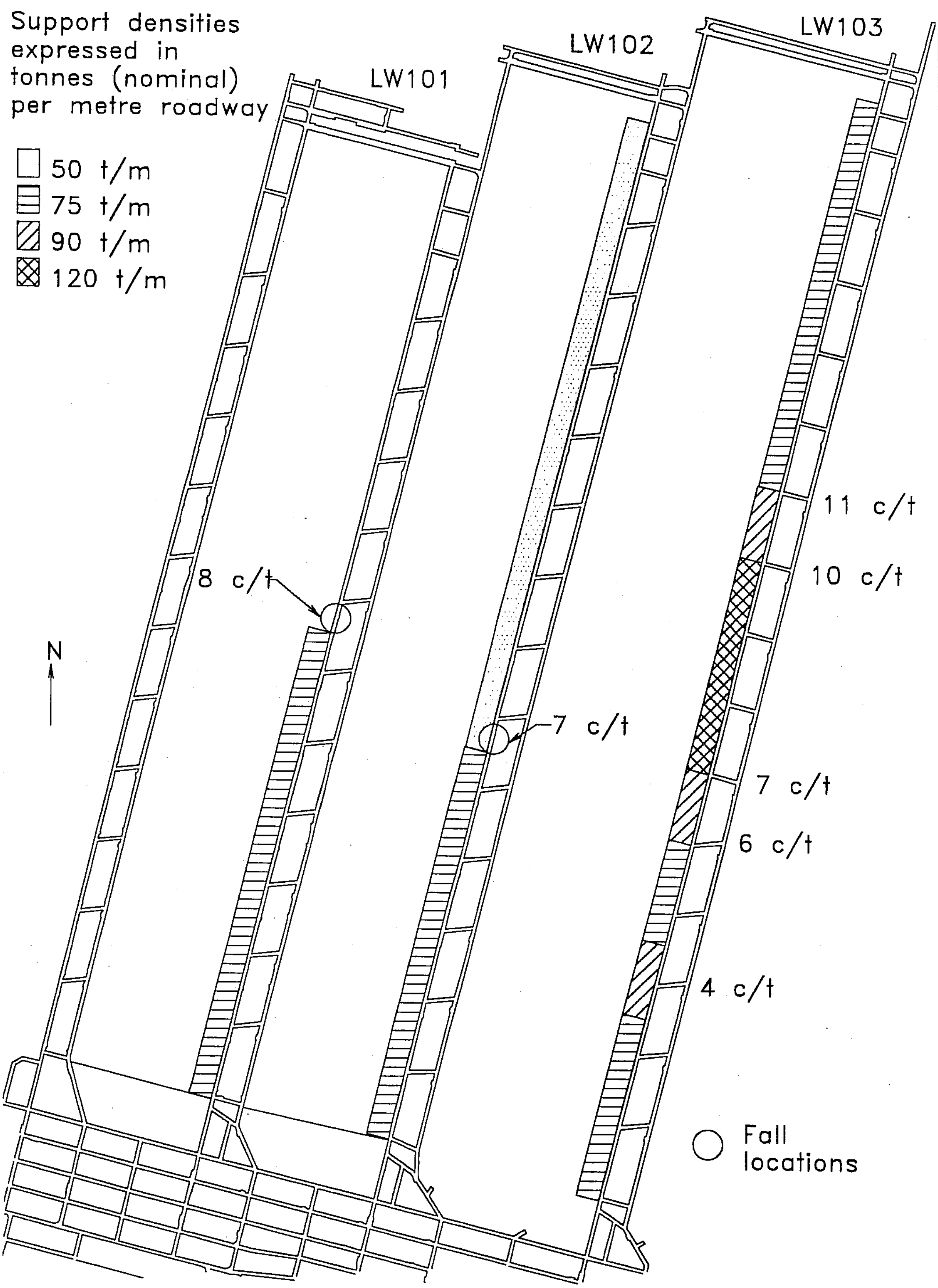

Figure 1. Gordonstone longwall gateroad secondary support densities and MG fall locations.

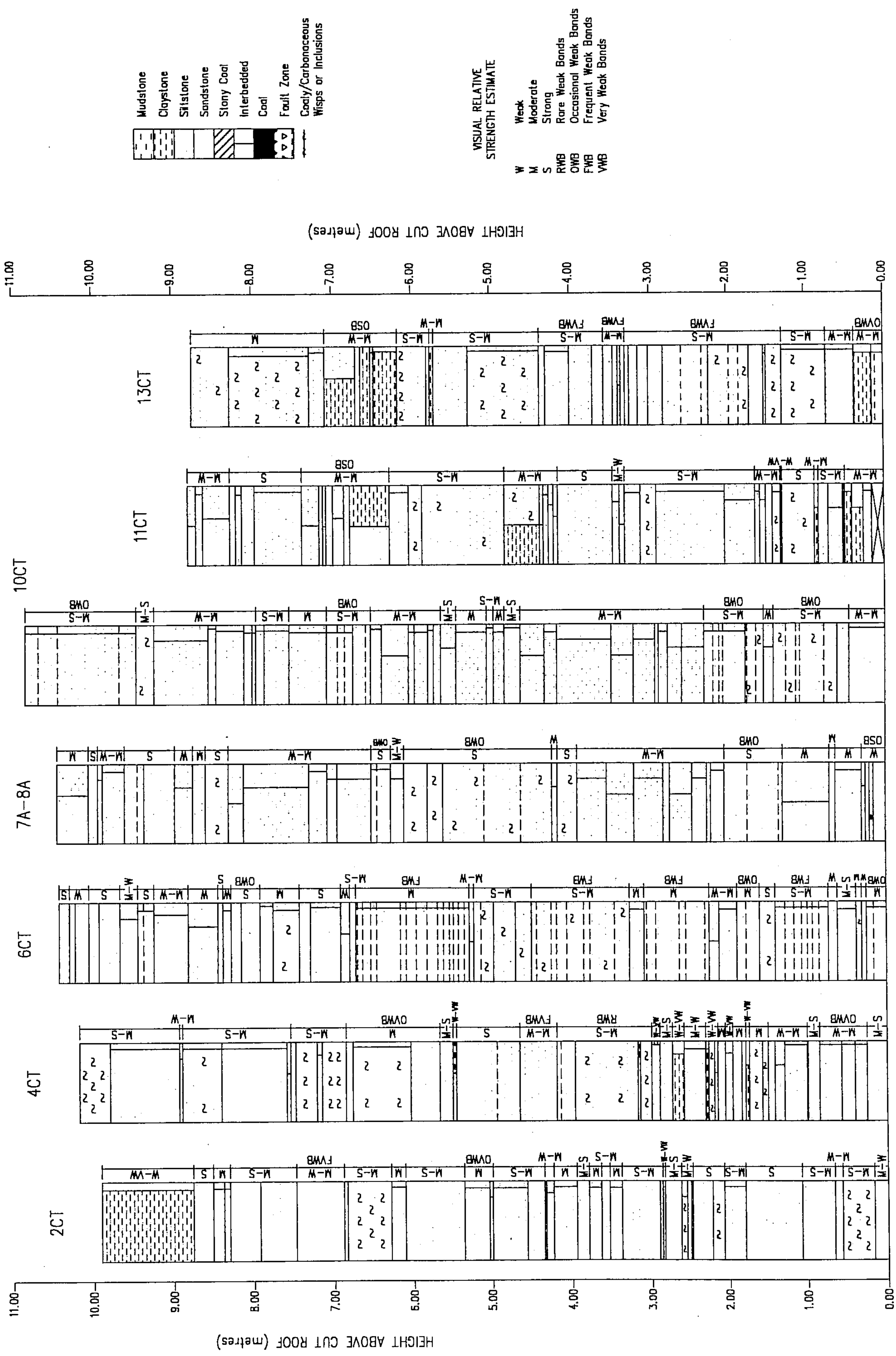

Figure 2. Stratigraphic roof section of 103 Panel.

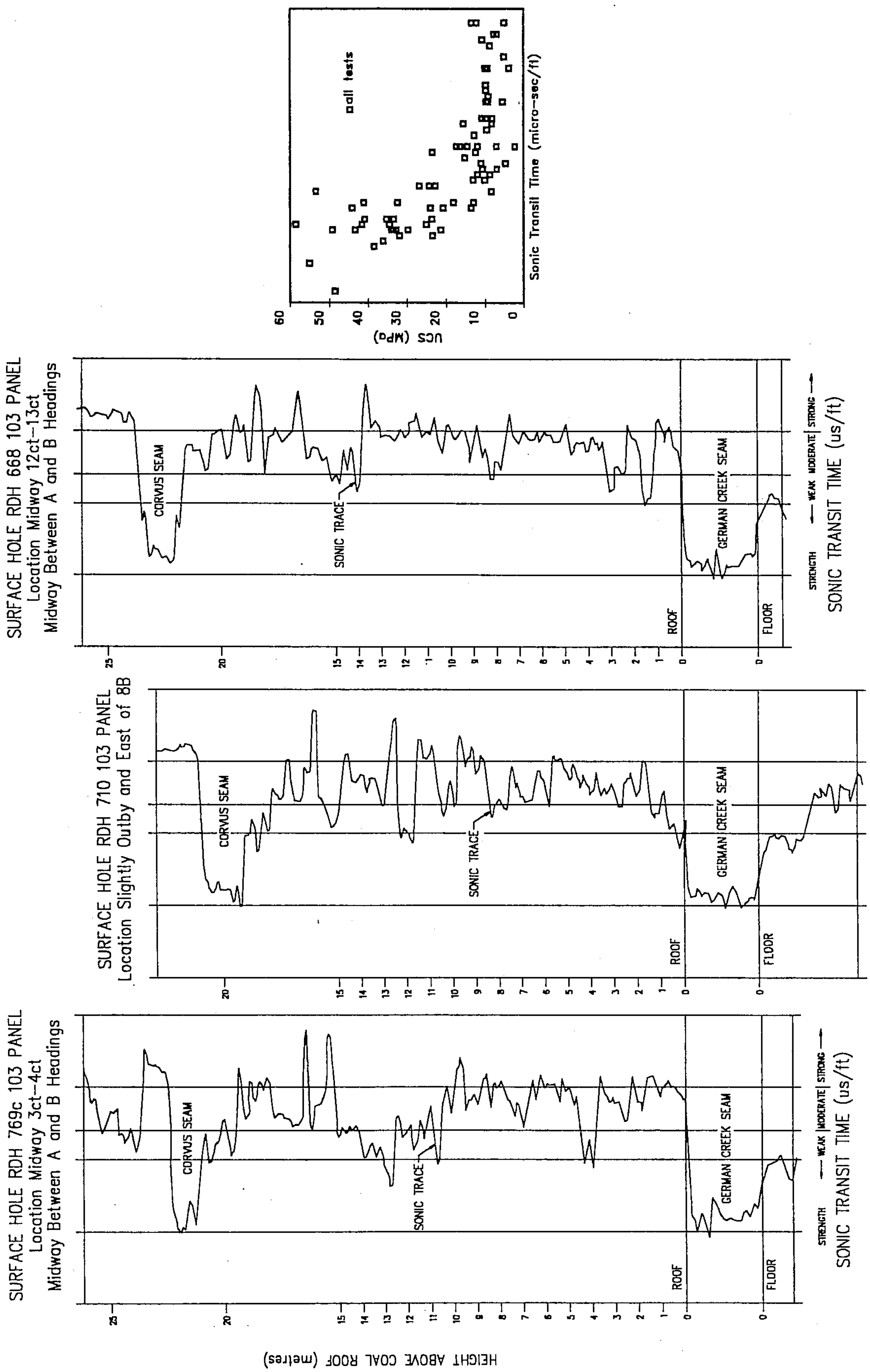

Figure 3. Correlation between sonic transit time and rock strengths.

of the secondary support density and placement for LW103 was based on a visual appraisal of the inseam roof cores. The appraisal of relative roof strength was subjective, but was required due to production schedules.

Objective rock property testing and subsequent computer modelling, currently being conducted by Strata Control Technology, will be used as a geotechnical basis for ongoing investigations.

In summary, the region between 7C/T and 10C/T was assessed as needing significantly increased secondary support density. Support densities were also increased in other regions (Figure 1).

To complete the current phase of the geotechnical investigation, extensive underground monitoring and observations are underway in order to correlate the geological and geotechnical results.

DISCUSSION

Ideally, roof support specification comes after a complete geological/geotechnical study, but production schedules require progressive specification and placement of roof support, i.e. during the gateroad development cycle. Obviously, the longer term consultative work of rock testing and modelling is required for planning, but tools are needed to provide an objective short term *in situ* assessment.There is a need for development of practical geophysical and rock property determination tools to be used during routine inseam evaluations. Borescopes would be limited in their application, as the visual assessment of rock strengths would still be subjective. CSIRO has developed a borehole penetrometer. A dry hole sonic velocity tool could have enormous potential, particularly as a correlation is being developed between sonic transit time and rock strength (Figure 3).

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NEW ROOF AND RIB SUPPORT TECHNOLOGY A MAJOR BREAKTHROUGH FOR ELLALONG COLLIERY

The recognition of the principle deformation mechanisms in the roof has led Ellalong Colliery to assist Barrett Fuller and Partners with the development of the Flexibolt. The impact of shear deformation on bolt loading was the major deciding factor in changing the primary support to Flexibolts. The ability to provide higher capacity reinforcement to the lower section of the roof has almost eliminated the need for 10 m long cable bolts in actively loaded areas.

The poor performance of the fibreglass rib dowels in highly loaded roadway ribs led Ellalong Colliery and Dupont to jointly develop a new yielding cuttable dowel that does not require resin capsules (RIMA Dowel).

This paper describes the various stages of practical implementation and results achieved together with the impact on roadway development productivity.

BACKGROUND

Mining conditions at Ellalong Colliery have traditionally been difficult due to an adverse ratio of ground stresses and rock strength around openings. In situ horizontal ground stresses as high as 40MPa have been measured at 500m depth. With the immediate roof rock strength of 20 to 30MPa, shearing of the roof on development is unavoidable and generally occurs ahead of the coal face. Sub horizontal shearing has been identified as the principal deformation mechanism (Figure 1). The shearing is locally phrased "Face Bumping" which can occur frequently during the cutting cycle.

As mining proceeds and the confinement is taken away from the surrounding strata, fingers of roof and floor stone are created which push into each other causing bulking and roadway convergence. In highly deformed roadways up to 4 fractures per metre have been measured with decreasing frequency into the roof.

Although not as severe or violent this type of shear failure has also been observed at Gretley Colliery at 100m depth. Once again it is the ground stress and rock strength relationship that is of prime importance. Depth of cover alone does not determine the conditions within a mine roof.

ROOF REINFORCEMENT PRINCIPLES

As is it impossible to stop the initial roof failure, roof support is primarily required to resist further horizontal displacement along developed fracture planes.

As ground conditions became more difficult at Ellalong gateroad roof support density went as high as 10 x 2.1m X-Bar bolts per metre supplemented by 2 x 10m twin strand birdcaged cable bolts per metre. Even at this density uncontrollable roof convergence was still experienced with "Iron Bound" conditions occurring frequently in the maingate. Cable bolts were often broken with severe bending of the X-bar bolts (see Figure 1). Cable bolt density went as high as 4 cable bolts per metre in the longwall take-off cutthroughs.

Instrumented roof bolts indicated that immediately after installation primary support frequently went into yield. This was a major concern as the capacity of the bolts to take further load on approach of the longwall was limited. Ellalong was at this time looking for a quantum leap in roof support technology and it wasn't until the early trials of the prototype Flexibolt that a breakthrough seemed possible.

RIB REINFORCEMENT PRINCIPLES

Although a difficult concept to understand by many, rib control is viewed as far more effective than roof reinforcement in controlling the roof. At Ellalong rib failure always precedes roof failure. It only takes a small amount of rib spall to decrease roof stability considerably. If the equation of a uniformly vertically loaded single beam is used $D_{max} = WL4/384EI$, it can be seen that even if only 0.5m of rib spall is allowed then the maximum roof deflection will increase by 400%. It is the recognition of this that has led to a major focus on rib control.

Unlike roof stone, coal is more nonhomogeneous and can exhibit granular

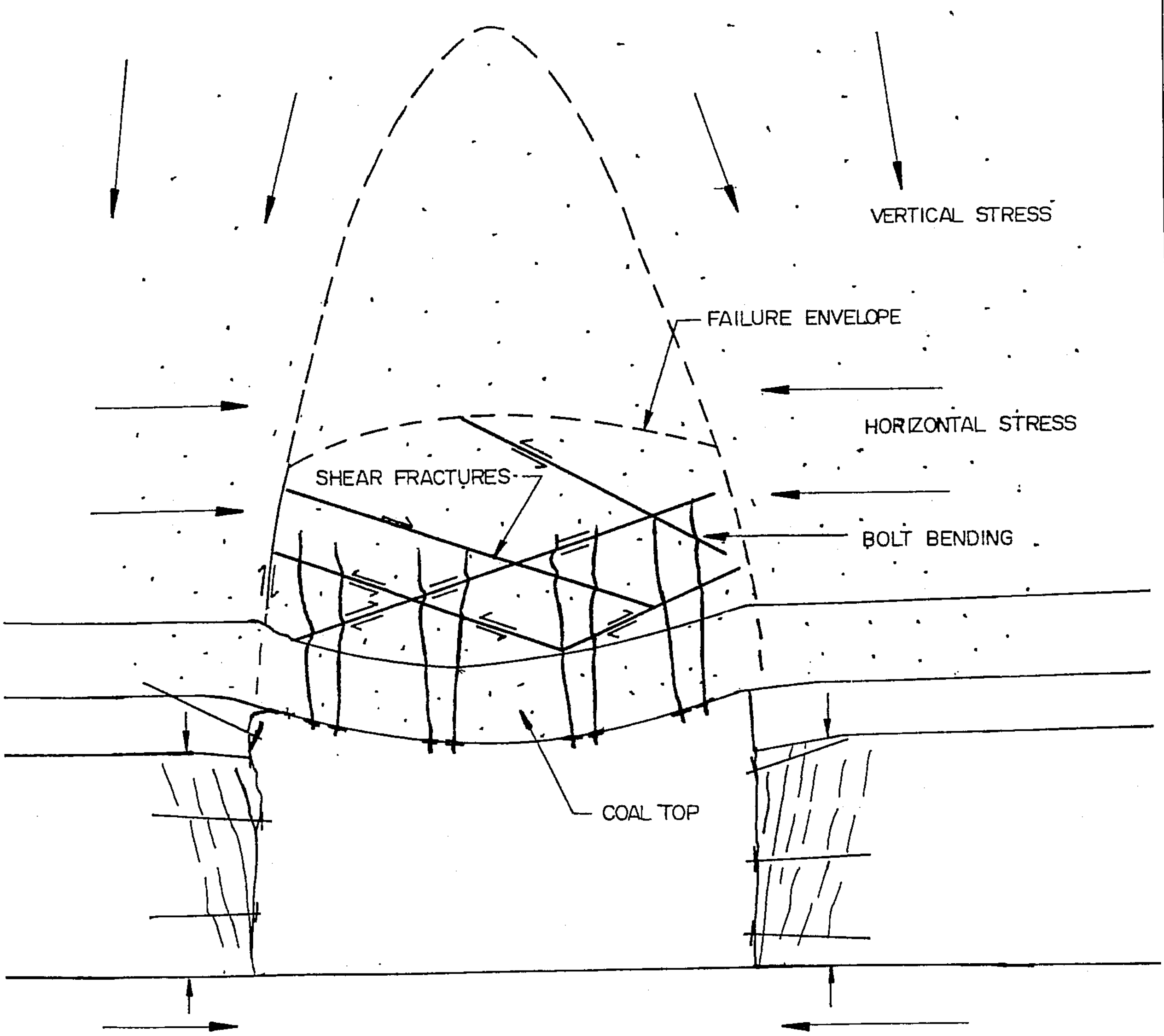

Figure 1. Failure mechanisms at Ellalong Colliery.

and/or buckling mode of failure (O'Beirne & others). The severity of which ultimately depends upon cleat intensity and loading.

Apart from the early stages of deformation a highly cleated coal rib is impossible to reinforce with conventional rib dowels as load transfer is not maintained during the latter stages of loading. The coal will fail around the bolt leaving the bolt ineffectual as a reinforcing tendon. The only work the bolt can do at this stage if the bolt head assembly is still functional is to hold the mesh against the rib.

The introduction of the Tensar cuttable mesh three years ago was successful, however, the inability of the stiff fibreglass dowels to maintain system integrity has been disappointing. Rib support failure is commonly through the nut pulling the wooden plate or the head of the dowel snapping off. This failure initiated the development of the RIMA Dowel, a rapid installation, yieldable and cuttable dowel.

FLEXIBOLTS AS PRIMARY SUPPORT

The concept, development and initial testing of the Flexibolt has been outlined previously in Fuller & others,1994 and McCowan, 1992. Conclusions from the comprehensive trials conducted at Ellalong were:

- Flexibolts at reduced densities restricted roof movement in the bolted horizon to lower levels than the X-bar bolts.
- the additional reinforcement provided minimised deformation above the bolted horizon compared to the X-bar bolts.
- the ground stability achieved with 4 Flexibolts was superior to that achieved with 7 X-bar bolts per strap.

These trials along with the laboratory tests confirmed the superior behaviour of the Flexibolts over X-bar bolts particularly in shear.

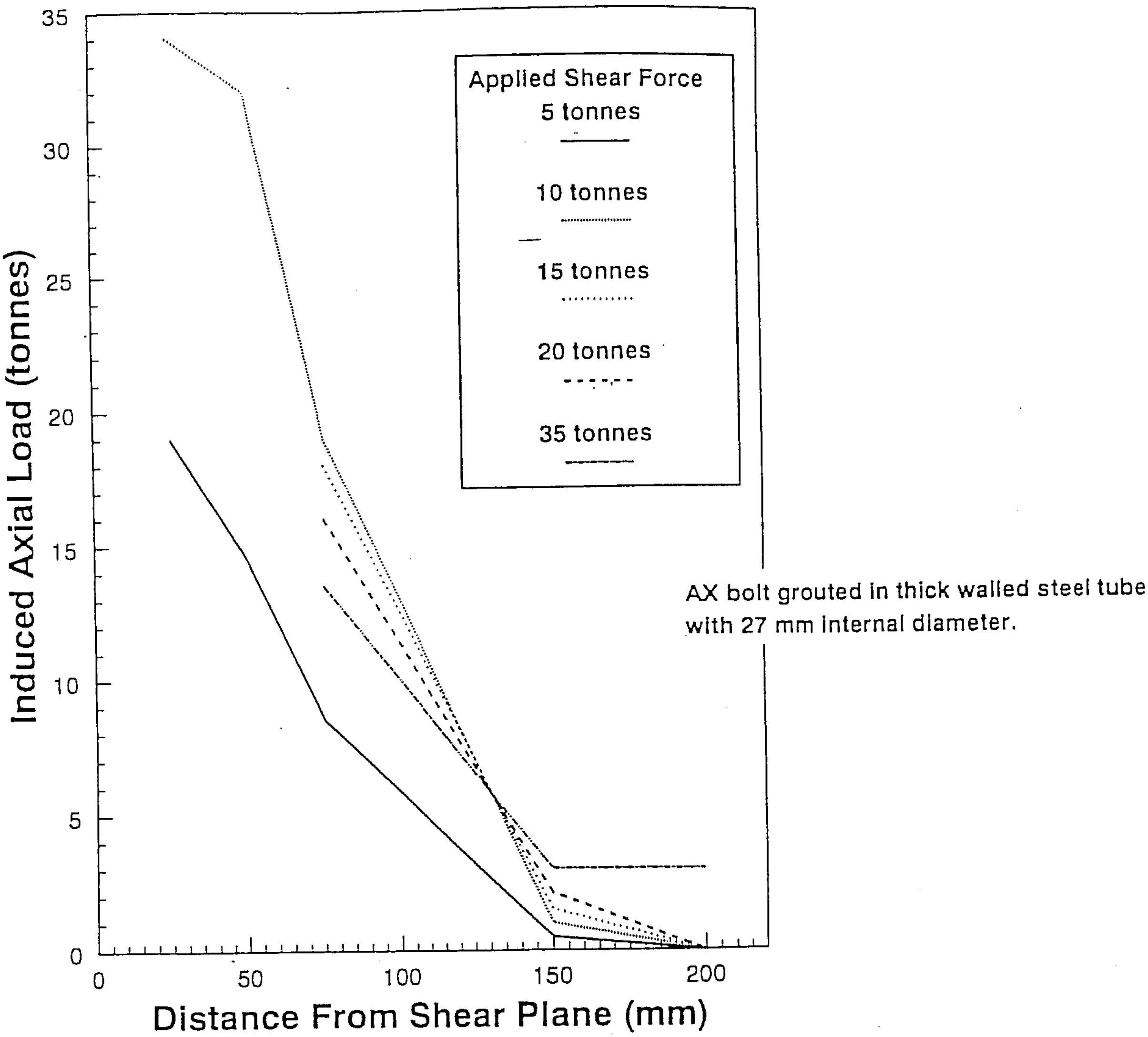

Figure 2. Induced axial loading from shear deformation (source Barrett Fuller and Partners).

Laboratory tests consisted of direct shear tests on resin grouted portions of the bolt. The results (Figure 2) indicate that an applied shear force of only 5t is able to induce an axial load approaching yield load of the bolt.

Underground and laboratory short encapsulation pull out tests confirmed that while the AX bolt displayed a reduced load carrying capacity after the peak pull out force was exceeded the Flexibolt displayed a near constant load capacity.

The above characteristics of the Flexibolt and a comprehensive testing program was sufficient justification to change the primary support system.

To change the roof support system all development crews were familiarised with the research undertaken particularly the adverse behaviour of X-bar bolts under shear and why the non-yielding Flexibolts at reduced densities would be appropriate for Ellalong conditions.

The workforce accepted the benefits of the Flexibolt and they were introduced in June 1994. Within a few days problems were being experienced with violent failure of the nut and they were immediately withdrawn from service. As the Flexibolt does not have a full thread around the circumference the nut has to be longer or hardened to achieve a similar strength to the X-bar bolt. It was during the hardening process that excessive brittleness was induced which resulted in the nut failures. The single nut assembly was eventually abandoned for a cone and nut assembly capable of withstanding a 40t load. The bolts were re-introduced in July 1994 and at the time of writing this paper over 500m of roadway had been developed with 3500 bolts installed.

FLEXIBOLTS AS SECONDARY SUPPORT

Since the development of the Flexibolt the number of 10m cable bolts used has been reduced significantly. The ability to provide a higher capacity reinforcement system to the

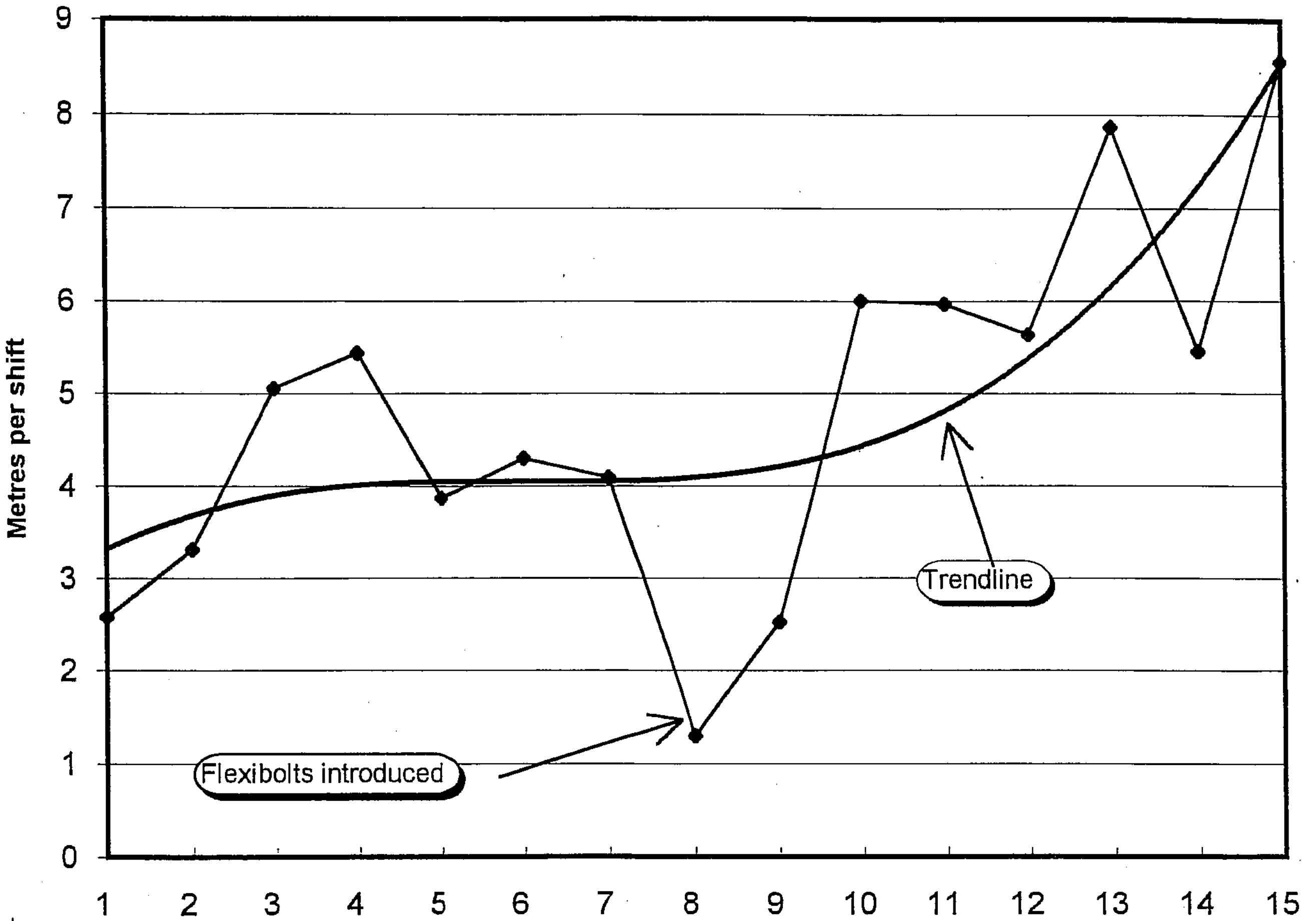

Figure 3. Development rate No. 3 Unit - Ellalong Colliery. **Week**

lower sections of the roof has reduced the need for long tendon support.

In the longwall take-off cutthroughs it is usual practice to install 3 x 10m cable bolts per linear metre. To trial the performance of Flexibolts as secondary reinforcement, 3 x 2.m long Flexibolts were substituted for the cable bolts in Longwall 12 take-off cutthrough. During the final approach of the longwall the cutthrough exhibited high stability with no requirement for supplementary support. Previous cutthroughs in the same panel underwent severe deformation and many of them required cribs to prevent them from falling. Longwall 12 was taken off successfully and after 12 weeks standing the cutthrough is still displaying high stability.

PRODUCTIVITY BENEFITS OF FLEXIBOLTS

At the present time 5 Flexibolts per metre are being installed as the primary support system. The initial design was to use only 4 bolts per metre, however due to the configuration of the three bolting rigs on the Joy 12CM5's, 4 bolts cannot be installed due to the excessive roof span between the inside bolts. Modifications are planned for these miners to allow the installation of 4 bolts.

Since the changeover from 7 AX bolts to 5 Flexibolts bolt per linear metre there has been a significant increase in development rate from 4 to 6 m/shift (see Figure 3). The actual benefit directly attributed to the Flexibolt is difficult to quantify at this stage as the miners were relocated to the other side of the colliery prior to the introduction of the Flexibolt. This means that there has been a change in some of the development system parameters as well as the roof support system. The predicted benefit using 4 bolts instead of 7 was 20%, however, even with 5 bolts it appears this might be conservative.

RIMA DOWELS

The RIMA Dowel was a joint development between Dupont Australia and Ellalong Colliery and is now commercially available.

The RIMA Dowel consists of a variable length 18mm diameter fibreglass rod. The anchor consists of three consecutive plastic cones each fitted with a tapered split ring. On installation the split rings slide towards the cones expanding radially providing a friction fit against the borehole wall.

The RIMA Dowel was developed to:

- provide a yielding capability to maintain rib mesh integrity,
- allow installation in broken or unclean holes,
- be self anchoring eliminating the need for resin capsules, and
- allow further insertion and retightening in the advent of rib spall.

The dowel has been extensively tested at Ellalong including a trial section in Longwall 11 which has been extracted. For comparison the dowel was installed alongside resin grouted AROA dowels and the behaviour monitored as the longwall approached. A number of the resin dowels loaded up sufficiently to fail the rib support system by pulling the nut through the wooden plate or snapping the head off the dowel. The RIMA Dowels in contrast yielded at around the designed 4t load thus maintaining the plate and mesh integrity at the most crucial period when the roof is undergoing maximum loading conditions.

As the RIMA dowel is a friction fit the use of hydraulic rigs for insertion is desirable. They can however, be inserted by a sledge hammer but the workforce at Ellalong has not found this to be desirable and at this stage the introduction of them has been deferred until the miners that are fitted with rib borers are returned underground.

Design changes are currently underway to eliminate the friction fit so that they can be installed by hand. This will improve the speed of installation as well as making them more acceptable to the workforce.

CONCLUSIONS

The additional reinforcement offered by Flexibolts in high deformation environments has led to improved roof control and major productivity gains. Long tendon cable bolting for active zones has been eliminated by short to mid length Flexibolts. Cost per bolt has been reduced by as much as 80% and installation time reduced from one hour per cable bolt to between 5 and 10 minutes per Flexibolt depending upon length installed.

Containment of highly loaded coal ribs which fail in a granular mode can now be achieved by polyethylene cuttable mesh and yielding RIMA dowels. Reinforcement by conventional stiff tendons has proven futile.

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Bob Butcher

Powercoal

EXPERIENCE WITH FLEXIBOLTS AT ANGUS PLACE COLLIERY

Angus Place Colliery Operates in the Western Coal Field of N.S.W. Development activities are carried out beneath 3.0m of high ash coal inter-bedded with claystones. Roof support consisted of 5 x 2.1m long AX bar roof bolts in "W" straps per metre with 1.2m bolts placed in ribs.

In 1990 both gate development units encountered zones of highly structured and stressed ground. Roof bolt densities were increased to 12 x 2.1m bolts per metre and cut out distances reduced to a minimum.

This density of roof bolts provided only marginal improvements in roof stability and with the advent of 3 roof falls, one burying a continuous miner, a difference approach was needed.

ESTABLISHMENT OF NEED

Roof instrumentation was commenced and indicated rapidly propagating strain zones 3m into the roof. Within a short duration deformation continued to 5m to 6m.

Cable bolting was introduced and became an integral part of each 100m development sequence.

Typically, development would continue for 20 to 30m allowing deformation of approximately 150mm to occur.

The miners were pulled out, the bords centre legged and then cable-bolted.

Figure 1 indicated that half the panel lost time was attributed to cable bolting. Cable bolting activities also affected the efficiency of belt extensions.

Pull out test performed up through the roof profile. The following table shows typical results:-

ROOF TYPE	PULL-TEST (Tonnes)
Coal Roof	5 - 10
Stone Roof	15 - 22

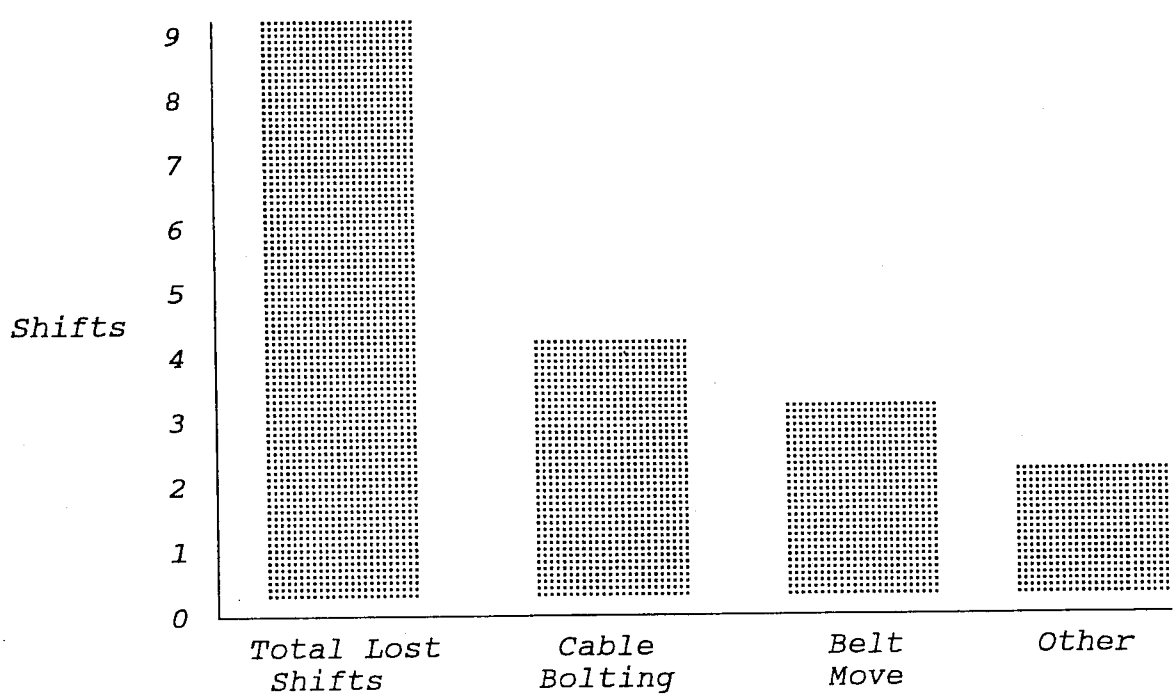

Figure 1.Breakdown of Average Number of Shifts Lost/Week.

THE PROBLEM

Production was continually interrupted to allow cable bolts to be installed up to the working face. The unit was typically down for 48 hours each application of cable bolts.

The cable bolts did not provide instantaneous roof support (cementacious grout).

By the time cable bolts were installed, roof softening had propagated 5-6m into the roof.

Serious shortfall occurred in the development schedule.

ROOF SUPPORT SYSTEM NEEDS

The needs of a roof support strategy were determined:-

(i) Maintain continuous production to achieve development rate of 7m per shift.

(ii) A type of cable support to be used in structured zones which:-

- *provided instantaneous support*
- *Installed as close to the face as possible in conjunction with normal face bolting activities.*
- *Have a capacity of 40 to 50t.*

(iii) Cost effective

(iv) Installed safely using available equipment

(v) Can be installed as a truss or tendon

(vi) 4 to 8m in length.

FINDING A SOLUTION

Various roof tendon systems were investigated and trialled. These included 4.0m slim-line coupled bolts and self-drill bolts. These were eliminated either by geotechnical or practical reasoning.

Early in 1983 the flexible roof bolt had emerged from trials under ACARP assistance.

Trial samples were manufactured and underground performance pull tests carried out. Different roof horizons and bore hole diameters were evaluated using a 1000mm resin cartridge.

Pull tests revealed that anchorage in excess of 40t could be consistently achieved in a 28mm nominal bore hole.

Installation was a relatively simple process, however cable deflection, when installing, became a problem with hand-held roof bolters.

A decision was made to integrate flexibolts with roof bolts as a primary support system.

BENEFITS OF THE FLEXIBOLT SYSTEM

Introduction of flexibolts provided many solutions to the roof support system deficiencies:-

- Could be installed at the face in lengths greater than seam height, due to its flexibility.
- Quick installation time (*5 - 10 minutes for a 4m bolt*).
- Uses existing roof bolting equipment
- Provides instantaneous point anchored support.
- Measured support capacity of 50t using a one (1) metre resin anchor.
- Installed in a 28mm bore hole.

IDENTIFIED PROBLEMS OF THE FLEXIBOLT SYSTEM

Several issues were identified during the introductory trials:-

- The point anchor (*40 - 50t*) had greater capacity than the collar assembly (*35 - 40t*). Roof strains experienced at Angus Place in individual horizons were measured in excess of 6%. Concern for catastrophic collar or cable failure became apparent.
- Severe cable whip on installation became a significant safety concern.
- In severely disturbed ground, the 4m cables slowed the deformation process considerably, however, additional cable support was required.
- The need was established to have the cables fully resin-grouted or post-grouted. Each of these systems presented operational problems that current technology did not address.

DESIGN OF THE FLEXIBOLT ROOF SUPPORT SYSTEM

The first consideration was determining an effective length that suited Angus Place's thick coal roof.

Several factors were considered:-

i) Softening occurred to a severe strain zone 3m into the roof soon after the roof was exposed.

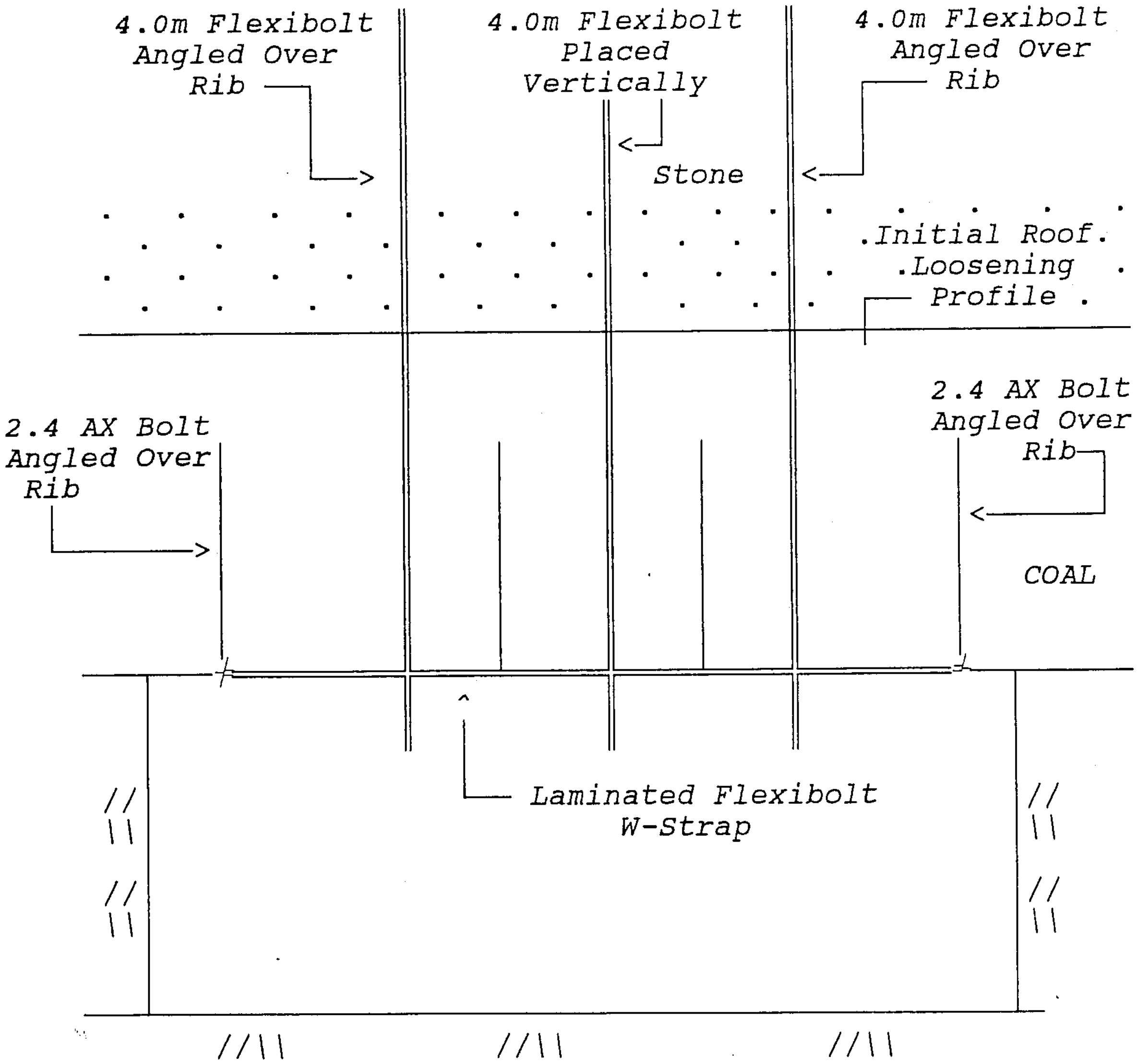

Figure 2. Shows a Section View of the Preliminary Flexibolt Pattern.

ii) A review of existing falls in structured ground revealed roof was displaced 3 to 3.5m in normal roadway widths.

iii) Bolt anchorage was very good in the 3 - 4.0m roof horizon.

A4m bolt length was chosen.

The second consideration was to design an anchor system that would yield at approximately the capacity of the collar assembly. A 29mm diameter bore hole was chosen after appraisal of pull test data, which indicated the anchor should yield between 35 and 45t while maintaining a high residual bond strength.

CONCLUSION

The introduction of the Flexibolt system has provided control of roof in severely disturbed weak strata at Angus Place mine.

Deformation rates were reduced dramatically. Deformation rates of 100mm convergence per day on standard roof bolts were reduced to 3 months exposure. This reduction in deformation rate enabled full length cable bolting to take place after the full panel advanced. The need for full length cable bolting was reduced by approximately 75%. Roof falls were eliminated and the propagation of roof instability was minimised.

The application of roof bolts, flexibolts and cable bolts is controlled by a roof control management system.

The various combinations of roof support are applied by means of a roof response system triggered by a roof control monitoring system.

The control obtained by this roof control strategy has provided a level of safety acknowledged by all employees at Angus Place.

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SIMULATION OF SOME GEOTECHNICAL ASPECTS OF ROADWAY DEVELOPMENT DESIGN

Roadway development design practice requires several geotechnical aspects to be considered. Standard engineering practice uses empirical design equations which have been developed based on mining experience. These conservative solutions are generally adequate for initial designs. However, roof-falls and pillar failures are usually associated with local inhomogeneities in geology and/or unfavourable stress conditions. The empirical design methods do not account for these details. Detailed roadway designs should therefore take into account:

- local geology
- *in situ* stress
- excavation dimensions

The use of computational methods as an aid to design analysis is well established though probably not as widely used as it could be. This paper presents a few select examples of the application of computational stress analysis to understanding specific aspects of ground behaviour during longwall coal mining.

MODELLING METHODOLOGY

Computational analysis provides the design engineer with a powerful tool to make better informed decisions with regard to the complex interaction of geology, structure, stress and installed support elements. Modelling provides insight to the active physical processes and ultimately an understanding of the mechanisms responsible for particular observed behaviour. However, care must always be exercised in interpreting results as modelling cannot and should not be used to predict absolute magnitudes of deformation or stress. It is important therefore that modelling be carried out interactively during the design and construction process. This requires close collaboration with operational sites.

The adage that "my problem cannot be solved by computers" is outdated and misinformed. Computational methods and power far exceed the current awareness and applications requirements. Indeed I challenge that if a "problem" can be defined, it can be modelled. And if a solution to a problem cannot be found then alternative solutions can be considered based on the improved understanding of the mechanisms of interaction. Probably the greatest weakness is with practicioners not being able to relate or translate their results to situation at hand. Hence, we often see significant efforts being channelled into instrumentation and monitoring programmes. These are important components of a design evaluation programme. Unfortunately, the interpretation of monitored data is often lost to descriptions of instrumentation performance (or lack thereof).

Computational analysis provides a focussed, cost-effective design method which can be used to augment insitu monitoring programmes.

Model Selection

A critical aspect of successful modelling is to select a constitutive model which adequately represents the material under analysis. Thus the use of elastic models to represent rockmass behaviour is quite unacceptable. Rock incurs damage and as such the mechanical properties and the distribution of stress in the body will be altered. More appropriate models would include Mohr-Coulomb elasto-plastic or strain-softening models. The presence of fabric in the rock will affect the behaviour. Laminations and cleat are best modelled with a ubiquitous joint model which allows for slip to develop within the microstructure without failing the intact material. Selection of an appropriate constitutive model must be based on an awareness of the actual expected physical behaviour.

KEY ISSUES IN ROADWAY DEVELOPMENT

Roadway performance and stability are largely controlled by the geology, *in situ* stress and excavation sequencing. Manifestations of these interactions can be observed in rib spalling, floor heave, guttering, surface subsidence, water inflow to panels or outbursts.

Initial driveage of roadways provides access to the reserve. It is important that these roadways remain stable throughout the mining life. Standard practice is to cut the excavation and to insert artificial support. Both operations present high cost activities. It is important therefore to optimise the support density for the specific local conditions. Various configurations of rockbolts, cable anchors, W-straps or meshing can be designed and their effectiveness can be assessed using an appropriate model. An example of one configuration of roof and rib support in a main development drive is shown in Figure 1. The section can be interrogated for stress and deformation in the rockmass and in the installed support elements.

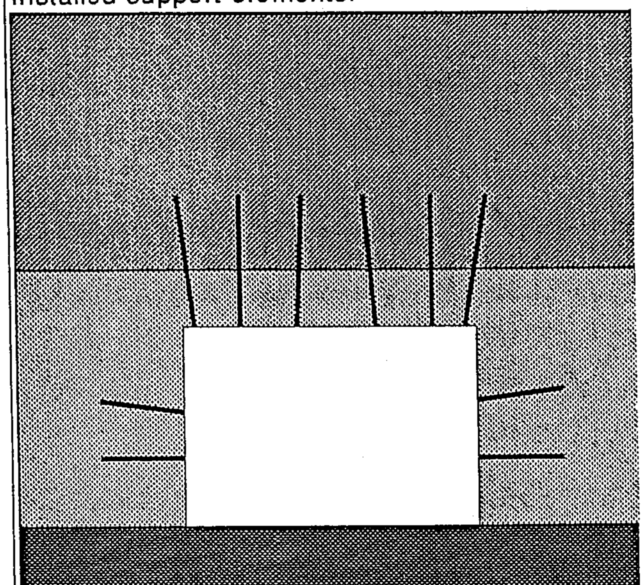

Figure 1. Main development drive with support elements.

During the next stage of mine roadway development, extraction areas are blocked out to define panels and chain pillars are created between intersections of roadways. Depending on the insitu stress conditions these intersections can present areas of high stress concentration. Using three-dimensional stress analysis methods the stress changes can be adequately modelled. The effect of artificial support can also be analysed. An example of vertical stress distribution at a roadway intersection is shown in Figure 2.

During subsequent longwall extraction, significant stress changes will be induced on the roadway, support elements and pillars. The response of each of these components can be simulated. Figure 3 shows the distribution of tensile fractures developed in the roof and overburden rockmass after panel extraction. Subsidence induced by goafing is indicated and its effect on overburden permeability can be inferred from the distribution of volumetric strain in the overburden. Damage extends to surface and presents a conduit for surface water inflows to the underground workings.

Flow of surface water through subsided overburden to underground panels is shown in Figure 4. From these analyses estimates of the potential inflow to an underground panel can be made.

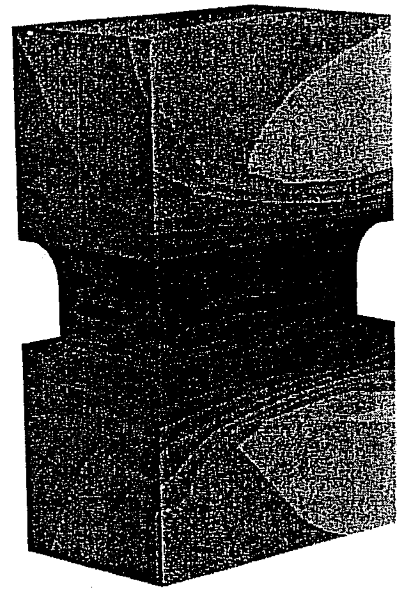

Figure 2. Stress distribution at roadway intersection.

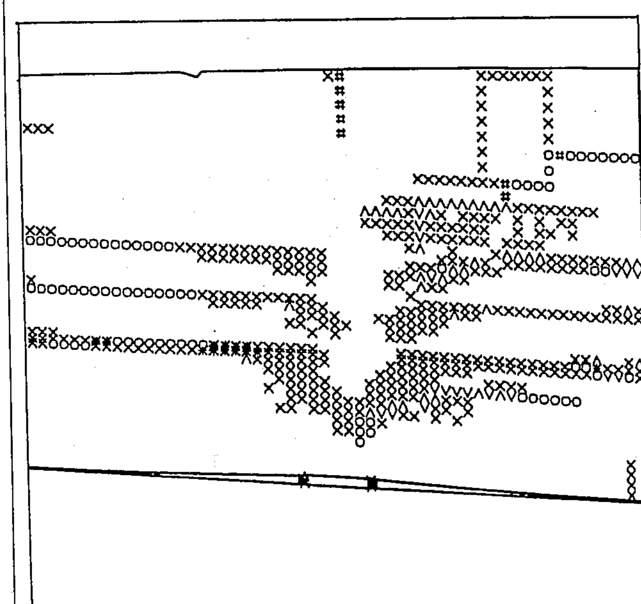

Figure 3: Rockmass fracturing induced by goafing.

The performance of pillars is a topic on its own and substantial investment is made on pillar design analyses and instrumentation monitoring programmes. As anyone with experience of instrument installation can testify, conditions are rarely consistent or conducive to providing reliable output. Data obtained from instrumentation programs requires substantial interpretation and analysis and ultimately represents at best a lower bound solution. Computational methods provide

ROADWAY DEVELOPMENT SYSTEMS - A

The central theme to the forthcoming sessions are to review and examine the current and alternate roadway development systems. Necessarily to achieve this we need to focus clearly upon what we, as an Industry, want and require - I would put to you that these issues have been proposed by the introductory session and are summed up as being a *development system* capable of matching the sophistication and potential of the *longwall system*.

Significantly the improvement of the longwall mining has occurred through several key factors being;

» a focussing of efforts in Research & Development,
» system scrutiny and innovation (under extreme cost forces);
» detailed discussion (such as this Workshop); and,
» commitment/perseverance between operators and manufacturers.

Roadway development is now playing 'catch-up' in consequence to the significant improvement in longwall performance. The consequence for roadway development mining is further magnified by the tighter constraints imposed by longwall mining. Development roadways must traverse previously considered poor 'ground' conditions - now we demand that the development mining match these more onerous conditions (as opposed to turning away to seek 'good' ground).

To better illustrate the diversity of approaches being applied to improving the lot of roadway development the forthcoming two sessions on the topic are divided into the *current roadway development systems*, to be followed by *alternate roadway development systems*.

The focus of this session is very much aimed at the development and refinements which have evolved from the bord & pillar extraction systems. Indeed during the last three decades the continuous miner has developed significantly to become a machine with high output capacity and greater reliability, but unfortunately the machine is under-utilised by the changed circumstances associated with the other elements of the mining cycle - it is not by any means *continuous*.

The adopted norm from the evolution of the continuous miner is that of a bolting platform (roof & ribs) capable of intermittently forming a well profiled gateroad. However, along the way some radical departures from this norm have been trialled under specific circumstances, such as in-seam miners and tunnel borers - these more revolutionary approaches are to be further explored in the following session. Presently, however, the inability to concurrently hew coal and roof support has been broached by the Satellite type miner (U.S.), the Alpine ABM-20 and other prototype machines being developed. Highly encouraging results have been achieved through simultaneous coal cutting and roof supporting as outlined in subsequent papers and presentations.

During the last decade evolution of the gateroad entry system has resulted in the two heading, long pillar layout becoming the accepted and most efficient layout - occasionally the use of single-entry has been employed and is now considered a usual event to avoid delay of the longwall. Such regression to twin headings followed the initial attempt to emulate multi-heading layouts which resulted in a thrust to gain metres at any expense with totally sub-standard gateroads being the result.

Whilst the twin heading layout is now the Australian norm, we cannot neglect the experience and practice of our overseas contemporaries. The U.S. systems, in particular, have evolved via different constraints to those of Australia with different norms resulting. Close attention is needed to assess whether these similarities and differences are conditional or cultural - certainly impressive output is being achieved by longwall mines employing concepts which seam to fly in the face of Australian practice. Currently attempts

to regain productivity are being trialled using the *Cut & Flit* method with encouraging results emerging. We look forward to the review by our Key Note Address related to the U.S. practice.

The development *system* of course extends beyond the coal winning and roof support elements to encompass the layout of the mining area. Furthermore, coal clearance from the face and the many support activities required to ensure the *system* operates also require application of Research & Development.

Coal clearance from the mining face to the conveyor has for years relied upon the shuttle-car. The concerted commitment of suppliers and producers has resulted in the reversal of the longwall boot-end by developing the Mobile Boot-End, the Flexible Train Conveyor and surge units. Whilst further refinement is edging closer in re-engineering these associated issues and the associated teething problems, we remain encouraged by the potential of such technology.

In closing, I would invite your attention to a retrospect of the aspirations and progress of the last decade. I have invited Mr Bruce Allan to provide a synopsis of the progress and developments of the past decade in order that we may better appreciate our present position. The following papers encompass a range of the innovations and developments being made to the *present systems*. Many of these developments once more prove that necessity is the mother of invention.

I commend to you the following papers and anticipate that they serve to stimulate your views and ideas, so that the session may provide a solid foundation to direct future research submissions.

STUART MIDDLETON
Coalex Pty Ltd

ROADWAY DEVELOPMENT SYSTEMS - A

Bruce Allan

Springvale Coal Pty. Limited

ROADWAY DEVELOPMENT - "The Vision Of The Past To The Present"

In July 1983 a joint initiative of the Australian Government and the Australian Coal Industry through the then National Energy Research, Development and Demonstration Council (NERDDC) developed a vision that through the automation of mobile equipment increased safety and productivity would result in underground coal mines.

Of primary focus was the potential contribution to rapid roadway development in support of longwall mining operations.

The vision which culminated in an overseas technical mission by representatives from ACA, ACIRL and Department Resources & Energy - July 1984, attempted to focus upon elements of the roadway development process which could possibly be automated both now and at some time in the future.

These elements should be considered around parameters for defining current mining systems (Table 1)

- Machine Cutting Profile
- Machine Guidance Systems
- Roof Support Systems
- Face Coal Transport Systems
- Monitoring and Control

MACHINE CUTTING PROFILE

The vision is for a heading development machine to simply and automatically cut a roadway in coal exactly to a predesigned geomechanical design appropriate to the mining conditions and suitable for future longwall extraction operations.

Although there are a number of fundamental principles involved in geomechanical design, there is still a large area of design, particularly in progressing from theoretical considerations to actual practice, where economic and operational constraints greatly override the geomechanical considerations. Obviously these latter constraints must take precedence, but to achieve optimum and well balanced design, all three constraints must be considered, and long term as well as short term economics assessed in conjunction with the other parameters.

An all too frequent example of this problem is the situation where excessive labour time and materials are committed to secondary support of development headings. Much of this expensive work could possibly have been avoided by greater recognition of the geomechanical factors at the time of initial development resulting in improved heading design and/or primary support systems.

Some of the more general or fundamental issues relating to geomechanics which must be considered at the outset, even if the solutions are often subjectively determined are as follows:

- What is the purpose of the roadway, i.e what use will it be put to and for how long will its stability need to be maintained - 6 months, 2 years, 20 years?
- Will you select a primary support system which is standardised throughout the pit and adequate for all geotechnical variations, or are the variations predictable enough (and operations/stores flexible enough) to enable alternate support systems to be introduced whenever difficult roof zones are being entered?
- Can you determine the extent of the abutment load (and possible damage) which stresses induced by adjacent extraction will cause, and if so, should you allow for this in the primary support design; install permanent secondary support; use temporary or mobile secondary support; or a combination of these?
- In planning a new mine, or upgrading an existing mine, do you have sufficient knowledge of geotechnical conditions anticipated in the lease? If so, will this knowledge be used to influence selection of mining equipment?

The answer to this last question five years ago was probably, no, not because of lack of knowledge or willingness of operators, but because of lack of suitable alternative equipment.

The major advance over the past ten years has been in the development of specialist full face heading machines such as:

- Joy 12cm 20/30
- Voest-Alpine ABM20
- Jeffery - 2048
- Anderson Kemcol Beaver II
- Joy Sump Shearer

All these machines have been focused upon high development rates, cutting stable roadway and providing a platform to bolt close to the coal working face.

The latter two machines, the Kemcol Beaver II and the Joy Sump Shearer are now possibly in he second phases of development.

Developments of the past such as the Dosco In Seam Miner, the Link Sump Entry driver and the Atlas Copco-Eickhoff ESA have not taken off in Australia. This is due to their operation within one roadway only and the limited transportability from one working place to the next and a possible requirement in Australian Coal Mines to drive both main and tailgate roadways at the same time.

As these new generation roadway development machines have developed, so has engineering complexity and physical size crept into the equation. At all stages in the future development of machines we must maintain simplicity and ease of maintenance foremost in our vision.

MACHINE GUIDANCE SYSTEMS

As we move to fully remote development machine operation, particularly in outburst prone areas, there will be an increased proven requirement for guidance and profile systems for continuous miners. The mission in 1984 looked in Europe and the USA at various techniques which could contain laser direction with that of identification of roof and floor. The development of this technology is being driven via automated tunnel boring machines and support initiated longwall shearers.

Development is expected to continue to increase in the direction for support of video operations for remotely operated development machines in outburst conditions.

Video monitoring and control has taken large steps forward on the last five years with small, robust and intrinsically safe equipment capable of being mounted on face cutting equipment in dusty and poor light conditions. This has been fully tested in highwall miner extraction units and underground outburst prone areas where the operation of the equipment is placed at some considerable distance away in a safe environment. This process will continue to be refined thus leading to improving safety and efficiency of the operation.

ROOF SUPPORT SYSTEMS

I suppose if there was one key area that was considered as having the greatest impact upon safety and productivity in the area of roadway development it was the activity surrounding roof support and roof support systems.

In the area of manufacture and defined maintenance processes through the use of robotics just about most activities can be carried out with speed, precision and safety.

In the area such as an underground development roadways in a coal sean a large number variables need to be contended with during the mining operation.

But clearly there would be a need for:

- Automated systems for roof bolting
- An upgraded and automated system for rib bolting

An initial step to automated bolting system have been the developments associated with miner mounted bolting and rib doweling rigs.

In recent years the new generation of hydraulic bolters have had significant impact on bolting cycle times and hence development rates. As well as the improvements in drilling performance we have seen improvements to engineering design which has removed a lot of hosing and thus reducing down time and damage.

Some of the miner mounted hydraulic drilling units now incorporate roof support jacks for use throughout the drilling sequence and the application of logic control enables the operator to simply initiate the drilling sequence with one touch, leaving the rig to automatically feed the drilling process, retract and stop awaiting the fitting of the chemical anchor and bolt.

The vision the industry had for the automatic bolters in coal mines took another step closer in 1994 through the Compact Autobolter project, which commenced under NERDDP as an ACIRL – SIG programme and has reached the practical demonstration stage. This project was targeted to produce a semi-automatic bolting machine suitable for attachment to such continuous users as the ABM20 and 12CM30. These units should be in commercial production by the end of 1994.

While miner mounted roof bolting and support systems may have been the vision of the past for development there have been a number of initiatives in the system of roof support such as "cut and bolting". In past these have been based on the American "cut and flit" technique which when adapted to the Australian scene have proved quite successful. Such equipment employed in this process can range from the small SIG Compact Bolter to the large twin boom mobile roof bolting machines.

New technology in the roof support system is now in use today in coal mines:

- Cable Bolts
- Flexi Bolts

This process while significantly improving roadway and face conditions has not been matched with the same speed of developed installation equipment as found in metalliferous mines.

Nevertheless as the demand continues to grow for these two techniques improved installation equipment will most surely follow.

In a large number of mines now rib bolting and rib support is an integral part of the face development cycle. This process continues to improve safety for employees working around the face area as well as to guarantee pillar stability for longwall and main heading development.

Research and development work in this area will continue to be of major importance.

FACE COAL TRANSPORT SYSTEMS

The vision held was that when face area operations become fully managed by process, how do we remove the stop start coal clearance process. The solution must be by some form of flexible continuous bridge from the continuous miner to the operating panel conveyor.

Many systems have been developed and many are continuing to be developed but as yet no real winner has appeared. It is still very much dependent upon individual mine operating conditions.

Possibly the most recent variation is the Meco Mobile boot end which is showing the most promise.

A limitation and restriction to this equipment is the managing of supplies and services along with the addition measures needed when driving the second adjacent heading.

MONITORING & CONTROL

For automation to be successful accurate data needs to be collected and processed in simple format with ease of interpretation.

Comprehensive data logging systems are available for most modern underground mining equipment and not unlike electronic technology today the rate of change is expodential, therefore we must never close our vision that it can not be done, only that the difficult will only take a little longer to complete.

Continued research in the area of monitoring and control will be necessary for the underground coal industry to move forward.

Table 1. Parameters for Defining Current Mining Systems

Definition of Mining Conditions	***Depth of Cover***	***Roadway Width***	***Roadway Shape***	***Max. Unsupported Distance***	***Number of Roof falls per metric of Roadway***	***Type of Production Equipment in Use***
Type A	Up to 300m	4.2m to 5.5m	Rectangular	Within 3 to 6m from face	3 to 8, depending on conditions	Standard CM or single pass miner
Type B	>300m	4.0 to 5.0m	Rectangular	1 to 1.5m	5 to 12	Standard CM Single Pass
		4.9m	Profiled Roof Rib Support		Depending on Roof Condition	Roadheading Machine
		4.47m	Profiled Roof & Rib			In Seam Miner
			Rib Support			Boring

CONCLUSION

The vision of the 1984 technical mission in looking forward to automated processes in the roadway development area has been instrumental in some of the technical achievements of present mining equipment such as the Kemcol Beaver and the Compact Autobolter. These innovations and others will continue to be refined and developed.

The Australian Coal industry has a strong will to continue to improve the underground mining of our coal in a safe and productive way. We must therefore continue to be innovative through well planned Research & Development.

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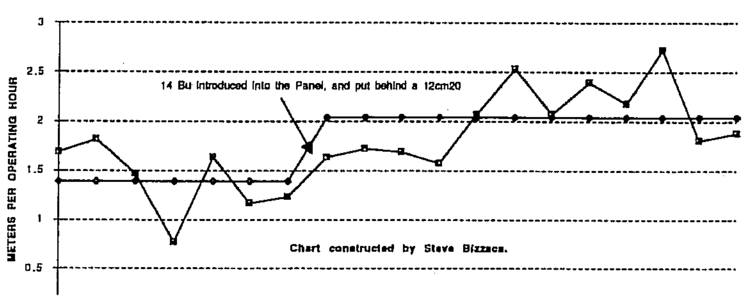
WADE KATHAGE

Gordonstone Coal Management Pty Ltd

APPLICATION OF SURGE SYSTEMS AT THE FACE

It is no news to anyone that most Longwall mines are development-constrained. Thus, most companies strive to keep the development ahead of the longwall without incurring unmanageable costs from labour or requiring too much capital from the shareholders. All, of course, affects the bottom line return on shareholders' investments.

TG101 METERS PER OPERATING HOUR



A Sililized Control Chart used to track the part of the development process in T/G 101.

It is important that the management of the development process know their process well and that they fully understand the effect that any change or modification to a process will produce. For example, if the company were to spend $4m installing a new continuous miner with a shuttle car, associated manning and support into an existing gate road development to create a super panel, would this give the process a cost-effective development increase or would it create a productivity and management nightmare? The coal industry faces the increasing need for concepts and systems designed to improve processes with minimal cost start.

The concept of surge capacity at the face is not a new one. In fact, a smaller variant of the 14bu was actively used behind 6CMs as early as 1979. However, the combination of the increased capacity of shuttle cars, the increased cutting rate of continuous miners, the trend towards multi-pass continuous miners, and decreasing roadway widths (leaving little room for coal on the floor with traditional brattice ventilation) meant that the 14bu concept was not used for roadway development for years until Gordonstone started experimenting operationally with the concept in 1992.

The Gordonstone experiment went one step further and incorporated a surge car concept into its development operations. Various case studies resulting from these trials are detailed in this paper.

INITIAL USE OF THE JOY 14BU LOADER AT GORDONSTONE

The development at the mine had started in earnest into its first gate road and already, due to problems at pit bottom, the mine was behind schedule in development. During feasibility studies prior to beginning operations, investigations were made into the use of surge machines in South Africa and the United States. It was then decided that a 14bu would be used to improve expected development rates.

The 14bu was introduced in the development gate road Tail Gate 201 (TG201) and used with a 12CM20 and Joy shuttle car. After a period of settling in and personnel training, the 14bu increased the productivity of the machine and decreased the overall cycle time of the panel advance. It should be noted that the improvements made in productivity and cycle time were not solely due to the 14bu but were part of the overall development process in the panel. The 14bu contributed to these improvements by taking out the variation in the haulage system (shuttle car) by reducing the waiting time for the shuttle car between bolt-ups. It became even more effective as the drivage became further away from the boot-end. The surge capacity at the face was more effective when the bolting density was low.

The 14bu also helped in the cycle time reduction by keeping the roads cleaner and by cleaning up the belt road ready for the belt prep.

Disadvantages and problems

- *Initial rejection by the crews*

  This was overcome by involving the crews in the development of safe working procedures and by high levels of communication between management and crews.

- *Dust control*

  Due to high water content and good auxiliary ventilation, this problem has been controlled.

- *Exposed conveyor and gathering arms*

  Procedures were put into place to prevent people walking or working between the miner and the front of the loader. A side railing was mounted on the side of the loader's shovel but it was soon damaged. No injuries resulted from this high risk area.

- *Cable damage*

  Quite a few cables were initially damaged by both the miner and the 14bu. At first, one man was assigned to look after the cables, but after considerable consultation with the crews, a system was developed to ensure that the cables were placed out of reach of the pinch points and the loader's angle of operation was changed to reduce the impact risk. It also proved to be more effective in its cleaning results.

- *Extra machine pinch points*

  As with the problem with cable damage, the original method used was changed to allow the loader to clean along the ribs at an acute angle to reduce the surface area of contact between machine and rib. By taking out the man looking after the cable and restricting access around the machine while it was operating, the risk to personnel was reduced.

- *Obstruction to supply and man movements*

  Positioning of the 14bu when supplies were brought helped but this problem remained a concern and actually delayed re-supply times.

- *Greater heat at the face*

  Even though a good ventilation system was maintained, the heat remained a problem, particularly during the summer months.

- *Added risk in terms of utilisation and equipment availability*

  For example, the 14bu increased productivity and 12CM20 utilisation but discounted this gain by its own utilisation time (breakdowns).

- *Poor water handling*

  The 14bu could not handle high water make. It could not load out a high water content slurry and, in fact, contributed to slurry generation. We limited the amount of coal on the floor during high water make, particularly when we knew the coal would not be able to absorb the excess water. At times, the coal from the coal cutter was directed straight onto the shovel of the loader to prevent coal getting into the water and forming slurry.

- *Restricted movement*

  The coal at the back of the miner restricted backward movement of the 12CM20 which sometimes restricted corrective action in cleaning up at the face and recentring of the roadway. Timing of when to dump coal onto the ground was important. This required crews to know their process well. However, some crews had difficulty in managing this system leading to some delays and some direction deviation.

- *Restrictive canopy*

  Operators found the canopy restrictive they could not stand up to achieve better all-round machine visibility. They soon got used to the loader and took breaks in the process to get out of the machine and inspect the area around the loader and the roadway to stay familiar with their surrounds. **As shown in Figure 1.**

- *Floor horizon damage due to poor use of the loader's shovel.*

  If the operator was not watching what he was doing, the shovel could easily dig into the floor, tearing the floor up and causing a hole to form. If the miner's floor horizon was out of alignment, the 14bu would continue to drive into the floor, making the hole worse. When the operators understood what was going on, the problem was brought under control.

- *Increase in manpower by one man*

Advantages

- *Cleaner roadway and rib areas*

  This helped with road building (which was essential at Gordonstone) thus reducing the resources needed to clean the roadway. Also, cleaning of the belt road was very useful to ensure that the belt preparation could be started quickly.

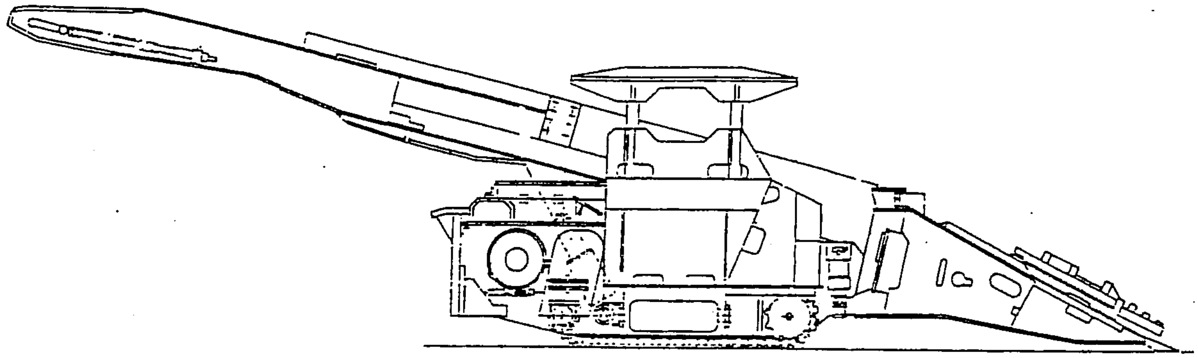


Figure 1. Side elevation of the 14bu Loader: note the exposed shovel arrangement and the restrictive canopy.

(Note: This was completed while the miner was finishing the 'break-thru'. The coal from this break-thru would be dumped onto the ground allowing the mining process to continue while the belt road was being cleaned up. This dumped coal would be picked up again when the 14bu worked its way back to the face. This removed the delay to the mining process and decreased the overall cycle time of the section advance.)

- *Towing ability*

  In the event of breakdowns, the machine was capable of being towed out of the way, thus reducing the delay to the development process.

- *Easy cable handling*

  The 14bu only required a cable the size of a shuttle car cable, making cable handling very easy.

- *Low ground pressure*

  The 14bu did little damage to the roadways with its weight. The shovel angle was the only concern regarding floor horizon.

- *Transportable*

  The 14bu was relatively ease to transport around the mine without electricity. It only required an Eimco to tow the loader.

- *Storage capacity*

  The 14bu was used to store some face supplies such as plates, chemicals, bolts, etc.

- *Increased efficiency*

  It reduced the impact of some short belt delays. It also reduced clean-up time by the miner, thus increasing the cutting availability of the miner.

- *Safety*

  It provided good protection for the operator, which is a welcome improvement on the older model of the 14bu.

- *Easier starts to break-offs*

  There was no problem in regards to shuttle cars trying to get under the tail of the miner.

Other trials

A second 14bu was also used in another gate road development (MG101). It showed similar results to those produced in TG101, but the lessons learned in TG101 helped to produce a quicker and more efficient utilisation of the second 14bu. The use of the loader with the belt prep and road works was expanded in MG101.

As we neared the end of the panel in Block 101 we found further uses for the 14bu in long drivages in both gate road development and mains development. The loader was used to act as a mid-surge point in the haulage system between the miner and the boot-end in:

1. driving the 200m face road, and
2. driving the main-gate entrance while the gate drive-head and belt were being installed.

These uses for the loader provided the following advantages:

- Increased surge capacity — up to 30 cars in the mains case which, in turn, reduced the delays in the haulage system. For example, there was no effect from short belt delays and no effect from shuttle car delays from loader to boot-end,
- No interference to the face supply system,

- Decreased wheeling distance of the face shuttle car,
- No effect on ventilation,
- No slurry generation problems,
- Elimination of most of the problems that exist with surging shuttle car to shuttle car.

Some disadvantages were found:

- Reduced working area in the panel and, in some cases, hindered access to the face,
- Increased manpower,
- The need for a second shuttle car,
- Floor damage caused by the 14bu being used in the same spot for a long period of time, requiring the alteration of the loading position or installation of a concrete pad,
- Tendency to overuse the surge point (storing too much coal at the loading point).

The 14bu was also trialled behind an ABM20 in one of the gate roads in an attempt to increase the productivity and help control a water problem in that particular area.

The results showed that the water was controlled better by keeping the floor and working areas cleaner, thus reducing the amount of slurry at the face area so that the water was cleaner and therefore easier to pump.

Productivity did not increase immediately, but the morale of the men increased due to the cleaner work area, thus increasing their productivity. Process delays decreased (due to water control) as the crews became accustomed to the 14bu. As the use of the 14bu became more efficient, the crews were able to use the cutting cycle of ABM20 more effectively and the productivity of the machine increased and stabilised. This resulted in an increase in production and a decrease in the section advance time cycle.

The 14bu still had similar problems to those that had arisen in other cases with Joy loaders. It should be noted that while the 14bu was being used in T/G 201, another ABM20 was being used in a gate road in similar but wetter conditions. It did not have a 14bu but the crews in that area had been following a process improvement program in which they fully understood their process and were actively improving their process to handle the water conditions.

They found that they were able to keep the area clean and the water pumpable and maintain high productivity by:

- adjusting their cutting cycle,
- cleaning up the face area more frequently,
- installing a rubber flap into the shuttle car (to prevent slurry running back out or the shuttle car),
- loading the shuttle car more effectively.

This showed that a crew with a superior knowledge of their process had, in fact, achieved greater productivity than the crew with the 14bu, without the inherent risks of an extra machine in the process and the labour drain of another operator.

The crew without the 14bu knew their process well enough. They knew what effect the 14bu would have on the process. They found the 14bu would not be worth the risk and labour drain unless the bolting density would decrease. However, at a bolting density of 8–10/m, the critical path of the face process was not the haulage but the bolting up to 160m away from the boot-end. At this point, the 14bu would be of credible use. In fact, it showed that the 14bu was used only as a Band-aid solution in the other ABM20 panel to hide the lower level of understanding that the crew had of their process.

The method used by this crew to improve their process is shown in Table 1.

NOYES SURGE CAR

Gordonstone had three multi-pass machines in use at one time, but because of their high movement in the cutting cycle, the 14bu was not suitable. This is why the surge car was trialled.

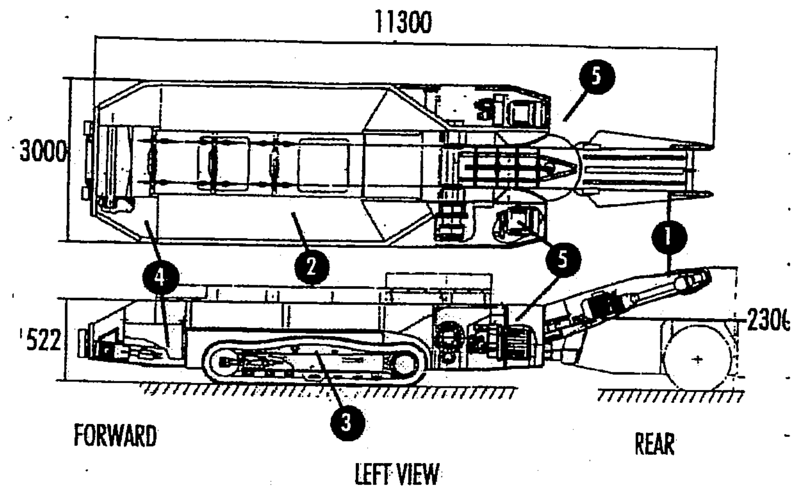
The machine was used in several areas of the mine, primarily with a multi-pass Jeffrey miner. The surge car's flexible conveyor tail (similar to a miner's tail) was track-mounted with a capacity of 25–30 ton. It was radio remote controlled.

The surge car was never used with the ABM20s.

The surge car successfully increased the capacity of the miner regarding productivity and utilisation but not to the same extent as the 14bu did with the single pass machines. This is because it did not have the same surge capacity and did not clean up the roadway which meant the miner still had to spend time cleaning when it could have been cutting.

Disadvantages

- There was limited room around the machine, therefore it was a high risk area for pinch points.
- Vision around the machine was restricted.
- There was limited access for getting supplies to the face area.



- The machine was difficult to move from the site when it broke down.
- The surge car proved to be as dirty as a shuttle car.
- There was no storage area for supplies.

Advantages

- There were significant productivity benefits to the cutting cycle.
- The surge car kept the coal off the ground and handled wet conditions well.
- Flexible operator positioning provided by the radio remote control meant that the operator was able to stand up in the cabin, allowing an all-round view.
- The flexible tail allowed excellent loading capability onto the shuttle car.
- There was capability to expand the concept by mounting rib support rigs onto the side of the machine or secondary roof bolting rigs.

CONCLUSION

In conclusion, depending on what process the surge capacity would be used in, it would be a definite help to the development productivity if the critical path of the development process is the haulage and not the bolting. Therefore, it is essential that a process improvement approach is taken to ensure that any proposed improvement such as a surge machine is a productive one. Because the surge machines are, in general terms, low-capital machines (as compared with FCTs for example), they would be, in most cases, cost effective solutions. Great success can be achieved in managing and operating an operation if a Total Quality Management approach is taken. This was done at Gordonstone by Malcolm Roberts, who I believe is one of few people who fully understands TQM.

Another person who was involved with surge machines at Gordonstone and also understands the continuous improvement process is Steve Bizacca. Malcolm Roberts, Steve Bizacca and I would all be happy to make ourselves available for further discussions with any interested parties.

IMPROVEMENT PROCESS

Understand the present system (Development) Keep future Goals in Mind.

Leader to understand the basic present system.

- Talk to the men and leaders in the process (in this case, Development crews and team leaders across the shifts, Service and Support teams + their team leaders associated with the area.)
- Construct the logic of the system (Network flow chart), include the variations across the teams. Best flexibility using a scheduling program.
- Construct the logic or structure of the management of the process, e.g. who makes the plans, who directs or leads, etc. LOGIC OF THE MANAGEMENT PROCESS e.g. Day shift team leader to set the 24 hour plan?
- Gather X data for the various parts of the process, e.g. average metres cut in different parts of the cycle.
- Insert the average times for the various activities in the process (logic network).
- Critical path the process.
- Identify the critical areas of the process, e.g. the cutting cycle.

Start to have an increase in predictability. (Know the degree of variation)

Key points:
- Understand the process. (logic)
- Gather data. (consult your men).
- Form a process of improvement. (basic direction).

Identify the critical areas in the process and the large variations in process.

From the activities identified in the basic process model, identify and prioritise the activities to be looked at more closely and in more detail.
- Communicate with the area's men on what has been found so far and how it was constructed (the logic behind the thinking) refer them back to the previous conversations which you used to construct the basic process model.
- Select the variations in the present process across the shifts
- Identify the activities of variation that are to be changed to the best activity as agreed by the Team leaders and team members, e.g. standard practice.

Start to study the Critical activities of the process, e.g. 1. Cutting cycle, 2. Belt prep and closures, 3. Supplies, 4. Water control, 5. Roadworks.
Process for each critical activity
- Talk to the Team leaders, miner driver, bolters and shuttle car driver, etc. about what we are presently doing.
- Construct the logic network diagram for the cutting cycle. (include the variations across the teams).
- Gather the data with the team members' help on the duration times for the logics activities.
- From the above study, identify the variations across the shifts and agree on the best logic, e.g. rib bolting.
- Determine the sensitivities of each of the critical activities, e.g. Cutting cycle — Bolting pattern, rib bolting, operators knowing the cycle logic and being capable of managing it, shuttle car wheeling time (does not become critical until 160 m from boot-end

Developed a reasonable degree of predictability. (Reduced variation)

Key points:
- Analyse the process. (Greater understanding)
- Reduce variation. (Everyone does the same)
- High involvement. (Increases in productivity)

Continue with the variations identification and reduction and start improving the sub-processes. (Unnecessary delays, resources, method)

Monitor the process, e.g. Plan versus Actual.
- Accuracy and collection of Data important.
- Use pareto charts to group the Data to help give a general direction for further investigations, e.g. Drill rig delays.
- Use Control charts to monitor the critical areas of the process and the process as a whole.

Attack and investigate the out-of-process delays.
- Ask Questions until you find the base reason behind the delay (dig).
- Use Control charts to monitor the delays.

Improve the in-process delays.
- Track and study the in-process delays and improve them as part of the process improvement method, e.g. logic (know it) strive to improve the logic (more efficient).
- Again ask the Questions until you find the base reason behind blow-outs of in-process delays.

Using the logic of the system, identify the areas of possible system improvements
- Possible better sequencing or logic changes. (process model will tell you the outcome, e.g. cost and benefits of doing a belt move without belt prep, using a 14bu loader in the process, etc.)
- Change in roof bolting pattern. e.g. reduction from 8 x 8 bolting pattern to 6 x 6 has benefits but going from 6 x 6 bolting pattern to 4 x 4 has a small benefit with present rib bolting pattern.

High degree of predictability (Variation understood and reduced)

Key points:
- High Involvement continues. (productivity)
- Increase in process utilisation
- Reduced risk when process changes are used. (predictability)

Process is Stable. Identify possible improvement — Technology, Methods, etc. Analyse the risk and impact of such changes. Use High Involvement to Implement, then repeat Process Improvement method.

From the previous investigations, construct the logic of proposed improvements

- Example: Reduction of roof support and rib support can be achieved with a Sump Shearer. You would construct the logic model and input possible values to assess to viability of the changes. Then you would use a similar method of process improvement as previously discussed.
- Another example: Reduction of rib support. Again, construct the logic and construct the model. This can tell you if it is worth spending time and money on the project or if it will hinder other operations, e.g. the roof support.
- Resource changes: what will be the effect of decreasing labour at the face and increasing labour to services or support , e.g. if you lower the manpower at the face to less than seven men, it will reduce the productivity of cutting coal and because this is the critical path activity of the process it will increase the duration of the process (cycle time). However, if you increase the resources of the belt prep (critical path), it will decrease the duration of the process (cycle time). Thus the management of the resources according the process needs is important to achieve the best possible results, e.g. process support crews.

DEVELOPMENT
A Case Example
ABM20 GATE-ROAD

You have Process Control
You have Continuous Improvement Framework.

Key points:
- Know what effect possible improvements can have. (good or bad)
- You have a clear picture on how one part of the process can affect the other in either present or future processes.
- You must have control over the process to effectively manage it.

Bernie McKinnon

Powercoal Pty Ltd (Wyee Colliery)

ROUND HOLES IN SQUARE PEGS

Stan Coffey has had a long and distinguished mining career and is well known for his visions with regard to technological development required by the Australian underground coal mining industry.

Following on from such previous visions as narrower roadways and sump shearers, Stan has focused his energies on the development of Underground Auger Mining. Stan has formed the "Auger Mining Dream Team" and he and a select band of others have invested their money and very professionally set out to make their dream come true.

Key players in the Dream Team are Mr. Coffey with forty-five (45) years experience in the mining industry (14 of which were as Superintendent for Peko Wallsend); Mr Charlie Deamer, the former Chief Design Engineer for Noyes Bros and Mr. Tony Stanley with more than twenty (20) years experience as an Auger Operator on large Civil projects both in Australia and overseas. Support players for the team include the NSW Department of Mineral Resources, Professors F. Roxborough and J. Galvin (University of New South Wales), Macquarie Manufacturing, Waratah Engineering, Powercoal and ACARP.

The Auger Mining Dream Team plans to develop augering machines that are capable of boring holes underground up to 2200mm in diameter. These machines will provide a safe and efficient tool for production of coal from areas otherwise rendered sterile and from longwall development roadways. Other applications are only limited by one's imagination.

The Auger Mining Dream Team is well aware that previous attempts to Auger underground in America, Germany and Russia have been unsuccessful. They are aware however, that surface coal augering has been performed successfully in the United States of America for over thirty (30) years and augering techniques have been successfully applied to large civil engineering projects. Holes have been bored through clay, stone and other unconsolidated material for 150m at diameters ranging from 250mm to 900mm and on grades of ±30°.

In order to succeed with Auger Mining underground in Australia, the Dream Team have undertaken a unique and novel research program. They have designed and constructed purpose built machines and cutting heads rather than make modifications to existing machinery.

The design criteria are based on initial testing carried out at Wyee Colliery on the Central Coast of NSW. A modified surface augering machine Coal Auger 1 (CA1) was used to bore holes in both the Great Northern and the Fassifern Seams.

Prior to using CA1 for the design testing a risk assessment was carried out and the need to consider the potential risk of frictional ignition was identified. The basis of assessing the potential for frictional ignition was a report prepared by Professor Frank Roxborough from the University of New South Wales. This report indicated that three (3) criteria were necessary to cause frictional ignition:

- Methane Air Mixture,
- Rotational speed of the Auger Head and hence the linear speed of the outer most cutting picks,
- Presence of quartzite rock or pyrite.

Discussions with Professor Roxborough indicated that the ignition source is provided by fine silica becoming incandescent and this meant that investigations into alternative ceramic picks was not warranted.

With regard to the Great Northern Seam, research has indicated that several frictional ignitions have occurred when miner picks have struck high silica content pebbles in the conglomerate roof. It was decided to carry out desorption and residual methane tests on freshly mined coal from standing pillars and also to bore a hole, seal it off and study the methane make. In order to carry out this testing the auger cutting head was reduced to less than 10rpm. A 900mm hole was bored part way through the pillar and the freshly bored coal was collected and sampled by ACIRL. The hole on withdrawal of the augers was sealed and a sample pipe installed. The quantities of methane in the standing pillars in the Great Northern Seam from these analyses were negligible and the remaining trials were conducted at cutter head speeds of 30rpm.

Methane is present in the Fassifern Seam and so examination of the coal seam including roof and floor was undertaken to establish the presence or otherwise of high silica content material or pyrite. An investigation and report by MEGS Consulting Geologists and the results of investigations conducted by the Institute of Coal Research at the University of Newcastle indicated that the propensity for frictional ignition of methane is non existent to negligible. Research also indicated that no frictional ignitions have occurred by the cutting actions of miners or shearers in the Fassifern Seam.

Trials and testing with CA1 have enabled:-

DESIGN AND CONSTRUCTION OF A 900MM DIAMETER CUTTING HEAD SUITABLE FOR PENETRATION LENGTHS OF UP TO 40M.

Several holes have been bored in the Great Northern and Fassifern Seams. Initial use of a 50mm pilot hole has been replaced by care and control of thrust and rotation to achieve accuracy. These holes have been strategically placed to enable future observation and convergence monitoring.

DESIGN AND CONSTRUCTION OF A 1500MM DIAMETER REAMING HEAD CAPABLE OF FOLLOWING A 900MM PILOT HOLE.

A very stable 1500mm diameter hole was driven 20m through a standing pillar in the Great Northern Seam at Wyee Colliery. Indications were that this length was about the limit on forward reaming due to the gyrations of the unrestrained 900mm auger string. Coal product size deteriorated with gyrations.

DESIGN AND CONSTRUCTION OF A BACK REAMING 150OMM DIAMETER CUTTING HEAD.

A very stable 1500mm diameter and 20m long hole was established through a standing pillar. Indications were that this method would be successful for holes up to 40m in length provided access is available on the other side of the pillar to attach the cutting head. Coal sizing was manageable.

General observations to date include:-

- If a pilot hole 50mm is used, it must be bored accurately and straight. Cutting head design can eliminate the need for a pilot hole.
- The revolution of the cutting head is critical and must be similar to continuous miners and shearers. Despite the fact that pick lacing is completely different to that of continuous miners and shearers, pick speed must be similar. Ideal revolution of cutting heads has been 30RPM, which gives cutting speeds in excess of 0.5m/sec.
- There is an optimum thrust required and it is not great. Too much thrust causes the auger to climb to the right.
- A leading cone on the front of an auger caused the level of thrust required to be too great.
- Best results have been achieved with the system in tension rather than compression. Tension is achieved by backreaming rather than forward reaming of holes.
- Auger sizes are critical for handling and storage. Important factors are length and diameter.
- Coupling of augers needs to be simple and quick.

COAL AUGERING MACHINE NO. 2 (CA2) has been manufactured and a risk assessment has been completed prior to underground trials at Wyee Colliery.

The focus at Wyee will be progressively larger diameter holes; coal clearance; modifications to cutting head to enable back reaming of a blind hole; modifications of hole profile and utilisation of system to drive cut throughs in conjunction with a mobile boot end.. See Figure 1.

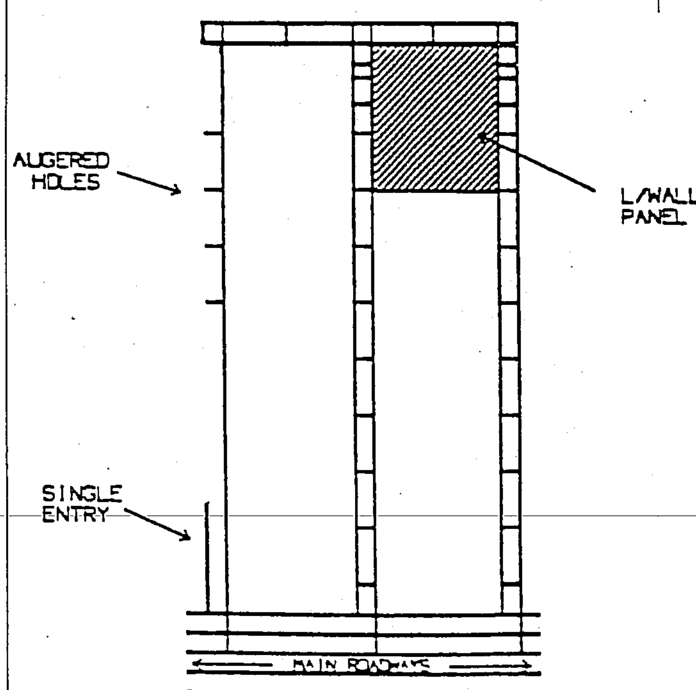


Figure 1. Augered cut throughs at Wyee.

The objective is to create a 24m long cut through within a 24 hour time period. The breakdown of time allocated to this process is as follows:-

Mobilise	1
Bore	6
Ream	10
Profile & Support	6
Dismantle	1

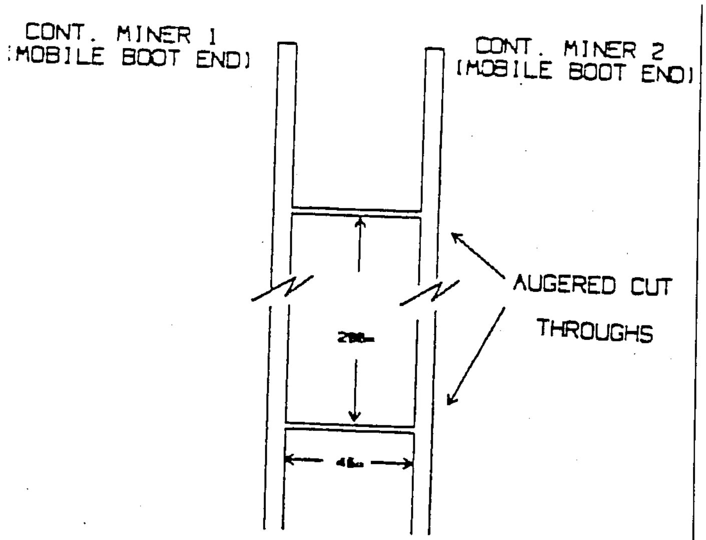


Figure 2. Augured cut throughs at Ellalong.

Ellalong Colliery mines the Greta Coal Measures near Cessnock in the Hunter Valley of NSW. It mines at great depth and develops under difficult circumstances. Ellalong is seeking to keep roadway driveage narrow and intersections minimal.

This machine is seen to have the potential to satisfy the needs of Ellalong Colliery also. See Figure 2.

COAL AUGERING MACHINE NO. 3 (CA3) has been manufactured at Waratah Engineering and surface trials have been carried out prior to a risk assessment and underground trials at Cooranbong Colliery.

Cooranbong Colliery works the Great Northern Seam on the Central Coast of NSW. It has a special need to access pockets of coal at shallow depths and under surface water bodies. Auger mining is seen as a means of enabling maximum percentage extraction of the Cooranbong resources.

The focus at Cooranbong will be to develop production potential in excess of 200t/shift, i.e. 3 x 30m holes at 1500mm diameter. This will be achieved by two (2) men and will require no roof support. There will be a concentrated effort on continuous system improvement for auger handling/storage and coal clearance. See Figure 3.

The ongoing objectives of the "Auger Mining Dream Team" include:

- Enlarging the boring capability of CA2 from 1800mm to 2200mm diameter.
- Modifying the process to give an 1800mm diameter hole a flat floor.
- Developing the ability to drill 200m x 900mm diameter holes between main and tailgate roadways for ventilation.
- Modifying the original CA1 machine to perform an automatic function that can be remotely controlled to reduce exposure of personnel to outburst conditions.

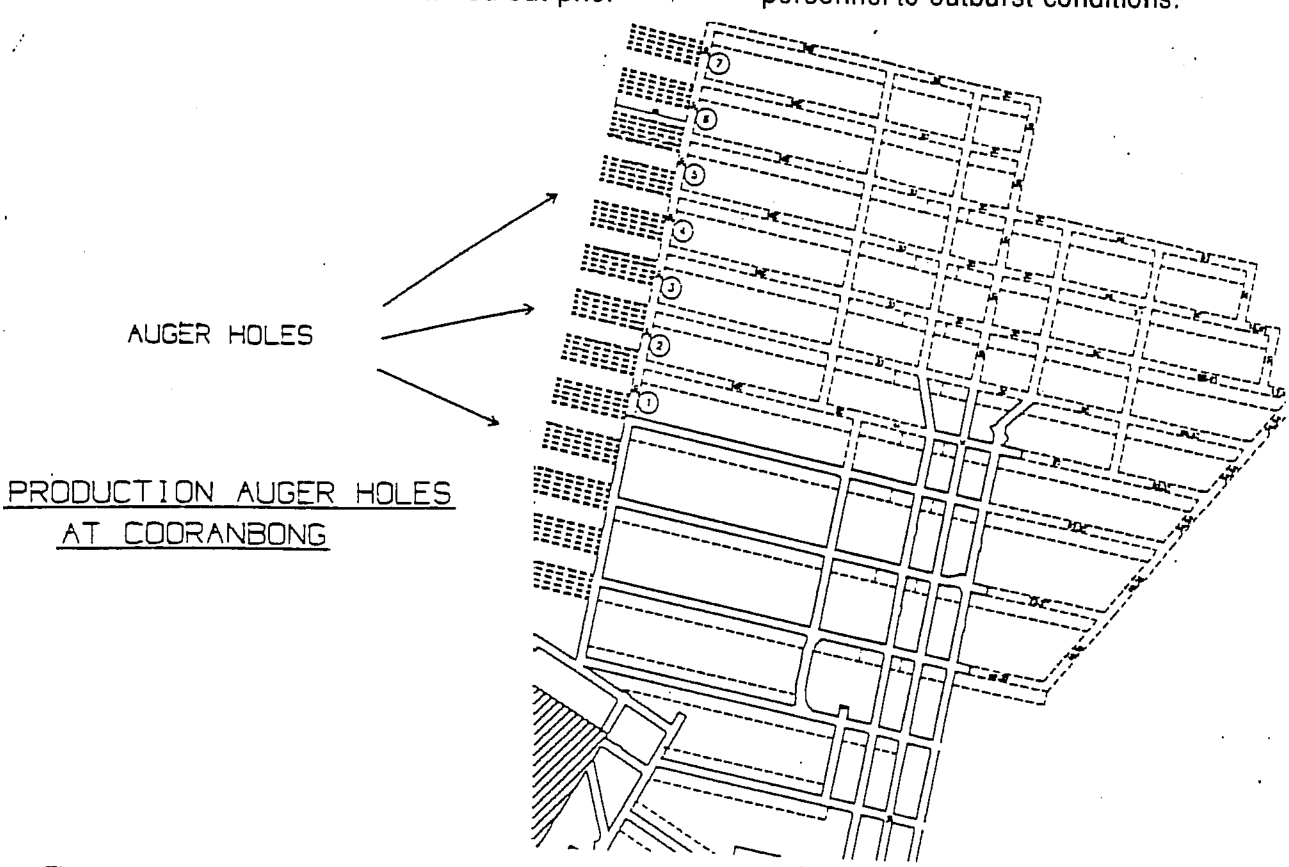


Figure 3. Production auger holes at Cooranbong.

- Investigating the feasibility of pneumatic or hydraulic transport of augered coal from its source to the mines conveyor system.

CONCLUSION

The concept of an auger capable of drilling holes up to 2200mm diameter is seen by the coal mining industry as a solution to a number of problems required to be overcome for Underground Coal Mining to remain Internationally Competitive.

The potential applications are only limited by one's imagination and include:

- An efficient alternative to the conventional cut throughs driven during longwall gateroad developments.
- A means of recovery of coal previously considered sterile due to physical mining conditions and/or recovery costs using conventional mining equipment.
- Deployment of remote control as a means of mining in potential outburst zones.
- Negotiation of fault planes by use of augered holes for 2nd Egress and Ventilation instead of heavily supported conventional roadways.
- Driveage of longwall take-off stubs.
- Negotiating fallen material to enable rescue of trapped men or provision of 2nd Egress/ Ventilation.
- Repairs to Longwall equipment when breakdowns are not opposite a cut-through.
- Single Entry Development ventilation and services.
- Geological exploration.
- Longwall ventilation when return is blocked.
- Major dewatering projects.

RICHARD PORTEOUS

Newcastle Wallsend Coal Company

SINGLE ENTRY MINING AT ELLALONG COLLIERY

Ellalong Colliery is situated 12km south of Cessnock NSW, and is the last mine still left working near the town after over a century of mining. It works the Greta Seam to provide a low ash high volatiles coking coal with exceptional fluidity. It is the deepest mine in Australia with current workings at 560m and experiences a high (28-40kPa) lateral stress. Intensive roof and rib support is necessary to maintain open roadways. It employs 283 persons at the mine with another 57 at Pelton Washery and produces 1.9mt ROM pa.

Between 1988 and 1991 an increasingly hostile geotechnical environment resulted in decreasing production from longwall panels. High stress concentration around the maingate caused dramatic roadway closures and falls, and in order to keep the maingate open in LW9 some $1.2M was spent on tertiary support. In order to enable the gateroads to be developed and the longwall extracted in a preferred direction the orientation of the next set of roads for LW10 was changed to be parallel with the principal stress direction.

LW10 DEVELOPMENT

Longwall 10 was developed on the other side of the main headings from the previous longwall blocks. Consequently there was no pre-existing roadway to be used as a longwall tailgate. Because of this requirement for a third roadway and accelerated development as well, the tailgate had to be developed as a single entry while the maingate and travelling road (trackgate) were to be developed as a two heading layout.

The tailgate of LW10 was developed using a miner and single shuttle car with belt extensions every 50m and an auxiliary fan at the start of the single entry. Supplies were taken to the face in MPV Pods after the MPV backed in beside the conveyor belt up to 1.2km. Man transport was by PJB cruiser although deteriorating road conditions and an increasing number of waterholes resulted in men preferring to walk in and out of the panel. Consequently development rates dropped from an initial 4m/shift to 2.4m/shift at the end of the relief heading and panel.

Ventilation did not present a great problem although increasingly larger fans and ducting were used as the roadway developed to its final 1.3km length.

The first Mobile Boot End (MBE) built was used in LW10 Maingate. The MBE was introduced into the maingate after some consultation with the workforce who designed the final layout of the machine and who welcomed it as an advancement. The companion roadway, the trackgate, was developed with another miner and car and the shuttle car side loading onto the belt with the MBE further inbye. Using the MBE in its pure designed role, that is with the miner loading directly into the MBE, resulted at once in a productivity increase. Over the life of the panel and subsequent panels this increase was measured at a consistent 28%.

Two problems had been anticipated prior to the use of the MBE. The first of these was that the distance to be driven by the CM in the maingate was much less than the CM in the trackgate as the trackgate miner had to drive all the cutthroughs — the Maingate CM could only drive ahead without a shuttle car. To avoid an imbalance between the length of the two headings, the number of cutthroughs was reduced by lengthening the pillar centres, and the trackgate miner was worked more often. Nevertheless at the completion of the panel the maingate was about 250m further advanced than the trackgate miner. The maingate miner holed into the relief heading and the shuttle car from that roadway was used with the maingate miner to drive the installation face of LW10 while the trackgate miner caught up.

The second problem anticipated was that of supplying the miner while it had a conveyor belt and MBE right behind it. The solution to this problem was to purchase a Gardner Denver air track and modify it so that material could be carried on two trays. This air track was narrow enough to fit into the space between the belt and the rib, and under the vent tubes. However a series of delays — delivery, mechanical failure of motors, process of approvals as a vehicle under CMRA — meant that the airtrack was not operational before the MBE was in a single entry. To overcome this a hydraulic motor was set up on the belt to drive it in reverse at approximately walking pace. Supplies were loaded onto the belt during the maintenance shift and then offloaded at the MBE onto the miner. This system worked well enough to continue its use for LW11 and LW12 development panels.

LW11 DEVELOPMENT

The maingate of LW11 was set up with the MBE once it was released from LW10. To overcome the problem of imbalance of drivage of the two miners a shuttle car was put in between the maingate miner and the MBE, with the intention of driving cutthroughs with the maingate miner. However after driving about 300m, having the car bogged, damaging and changing shuttle car cables in a 200m single entry and other inherent delays, the advantage of 28% quicker development was reduced to 10% and the car was taken out of the system.

Time pressures at this point dictated that the maingate be developed as quickly as possible, without regard to the trackgate. This was done with the result that when the panel was at its final length, the maingate was 320m in advance of the trackgate. So that a relief heading and installation face could be driven simultaneously, the belt and MBE were retreated from the face to the last open cutthrough, and another miner and two shuttle cars introduced. The belt was then rebuilt with a conventional boot end. This operation took four days, but the net result was that we were able to start the relief heading much earlier than if we had not used the MBE.

LW12 DEVELOPMENT

Given the productivity improvement when using a MBE, a second unit was bought in 1993. The intention was to install a second belt in the trackgate and then withdraw the belt when the panel was developed. So that cutthroughs could be driven, a shuttle car was narrowed to 2.0m wide by cutting 400mm out of its centre. This car could be driven up beside the belt, put between the miner and MBE when a cutthrough was due, and then withdrawn from the panel when the cutthrough was completed.

However a serious fall on the main track road of the mine prevented early installation of the second belt - the drivehead, loop take up and MBE were trapped outside. In this time the maingate was developed 750m inbye of the last open cutthrough. The gas content of the coal in this panel had increased significantly and composition was 80% CO_2 and 20% CH_4. Hence pollution of the intake air increased slowly over the 750m and approached the statutory limit of 1¼% CO_2. Added to this was another environmental problem — high humidity was generated by the heat of hydraulic motors and water from roof bolting rigs. Men wore raincoats to protect themselves from this water and just got hotter. It was not a pleasant environment.

At this point development was halted and the narrow shuttle car was driven in beside the belt and used in the development of the relief heading and installation face. The men were driven to the last open cutthrough and then they walked to the face. The crib room was kept close to the face.

The trackgate was developed another 250m but then also halted due to concerns with CO_2. This left us with a 500m single entry in which to install and run a longwall. Despite the limited access the installation was completed in the allotted time, the wall started and it has continued to run successfully.

THE NEXT STEP
LW13 DEVELOPMENT

Due to the high CO_2 content of the coal in the South East area of the mine a decision was taken in May of this year to transfer all operations to the northern side of the mine. A short block LW12A is being developed by grunching while this transfer takes place.

As with LW10, LW13 is in a new longwall area of the mine and so required a separate tailgate single entry and a two heading layout for maingate and trackgate. All three roadways are to be driven using MBEs. With a belt in both maingate and trackgate the layout is termed a "dual single entry system". With the benefit of our previous experience the panel will be laid out, set up and run to avoid as many of the pitfalls as possible.

The rail system will be kept up to the trackgate MBE. Although we currently don't own track mounted personnel cars these are being sought for transport for the crews right up to the face in the trackgate and close to the face in the maingate. Supplies to the face will be by rail in the trackgate and by reversing belt in the maingate. Cutthroughs will be at approximately 200m centres.

In the tailgate the face will be supplied by a reversing belt. Man transport into the face along the single entry is currently envisaged as also on the reversing belt though we have a process to go through to get an exemption for this. In terms of transporting an injured person out from the face the best option once all is considered also appears to be the belt though obviously not at 3m/s. A new type of litter for the care of such an injured person has been trialled in the most confined of space of the longwall and appears most suitable for a single entry belt.

The belt in the maingate and trackgate will be offset to the ribs to allow access for the narrow shuttle car for the driving of cutthroughs. Once a cutthrough is completed the car will be taken out of the face area.

A narrow Eimco, an EJC-60, has been purchased to be used in this panel beside the belt for service moves etc. This will be the second narrow Eimco at the mine. The first one is generally used in the longwall for servicing there.

OTHER SYSTEMS

A development system has been evolved at Ellalong over the last two years. Coupled with this evolution have been advances in other areas such as roof and rib support systems and machine reliability. Preventative maintenance schemes have resulted in 95% conveyor belt availability and 93% CM availability. In addition to other delays on a CM, as a rule of thumb availability decreased 1% for every hydraulic roof or rib bolter mounted on the machine. Higher reliability of bolters is keenly sought

WHERE TO FROM HERE?

The use of single entries, PM schemes and advanced strata support systems have delivered significant improvements. However there is still plenty of room for further progress to be made. An integrated supply system enabling supplies to be placed on the miner during the cut and support cycle without interruption to face activities is the next step for Ellalong. Monorail systems with purpose built cassettes carrying support supplies or other supplies are being investigated. This will also reduce manual handling type accidents in the roadway.

A continuous cutting and bolting machine will add the last piece of what is still a fragmented process.

At this point the process will consist of cut - convey - support - supply with all elements of the process capable of being carried out in conjunction with the others. Machine maintenance will then be the only function which requires the face to stop and other necessities such as stonedusting can be carried out during this stoppage.

Richard Porteous

ROWAN MELROSE

Voest-Alpine Mining & Tunnelling (Australia) Pty Ltd

THE APPLICATION OF THE ALPINE BOLTER MINERS IN COMPARISON TO STANDARD WIDE HEAD CONTINUOUS MINERS UNDER VARIOUS ROADWAY CONDITIONS

The main difference between a standard wide head continuous miners and Alpine Bolter Miners, is that the latter machines can simultaneously install roof support while cutting and loading coal. This process eliminates the 'stop-go' nature of conventional miners, and accordingly provides the potential for a more efficient operation.

The degree of efficiency varies with roadway conditions. This paper considers the potential increase in efficiency, using Alpine Bolter Miners in various roadway conditions.

OPERATING PRINCIPLES

The main advantage of the ABM-20 over conventional continuous miners, is the ability to simultaneously cut coal and install roof bolts. A slide frame allows the cutter drum and loading device to move independently from the machine chassis and crawler tracks, as shown in Figure 1.

The temporary roof support stabilises the machine while cutting and allows roof bolts to be installed 1.5m from the face. The relatively slow speed of the cutter drum reduces coal degradation and airborne dust generation.

The advantage of simultaneous cutting and bolting as opposed to a conventional wide head miner in relation to roadway development rates is shown in Figure 2.

This example considers the installation of 6 x 2.1m roof bolts, and 2 x 1.2m rib bolts for every one metre of roadway development. Both machines have four (4) machine mounted roof bolters and two (2) machine mounted rib bolters. Installation of all bolts is single pass. The simulation assumes that the drilling cycle for both machines is equal, and that both machines are serviced by two (2) shuttle cars.

The gap between the two lines, therefore represents the difference between non-simultaneous and simultaneous cutting and bolting. The possible development metres per operating hour for a standard wide head machine will, as shown reduce as the wheeling distance increases. The development rate for the simultaneous cutting and bolting machine, however, remains constant until the wheeling time in the operating cycle becomes critical. (In this case at the 70m mark).

ROADWAY CONDITIONS

Roadway conditions are however variable through-out both the world and within Australia, and it is therefore necessary to consider the application of simultaneous cutting and bolting machines over a broad range of roadway/ operating conditions.

Six categories of roadway conditions were considered, ranging from what would be considered excellent to poor for Australia's coal mines. The conditions for each classification are shown in Table 1.

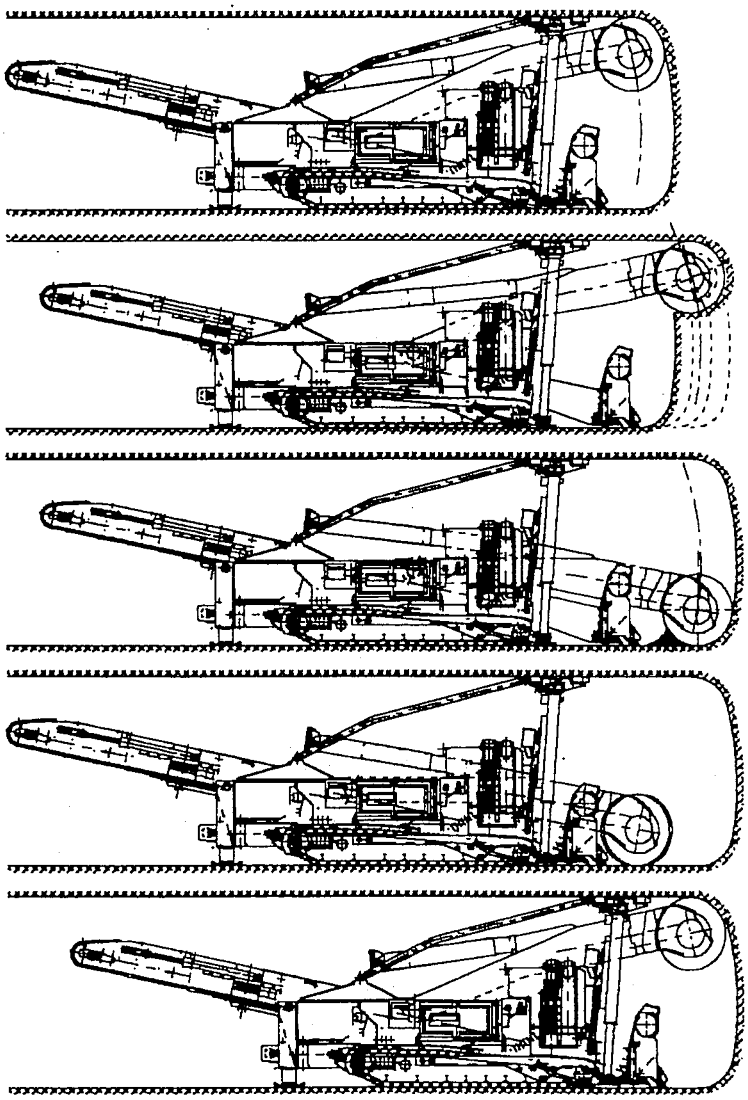
Table 1 - Roadway Conditions

CATEGORY	A	B	C	D	E	F
Number of roof bolts per linear metre	1	2	4	6	8	8
Number of rib bolts per linear metre	0	0	2	2	4	6
Distance (m) cut out before support, using a standard wide head continuous miner	8	6	4	3	2	1

The results for these conditions is shown in Figure 3.

As shown, the ABM-20 out-performs non simultaneous cutting and bolting machines in all categories, except category 'A'. The degree of out-performance varying with wheeling distance.

In category 'A', the bolting cycle time is not as critical for the development cycle, therefore aspects such as machine loading rates become more important. In these conditions the inability to use the steep apron as a bunker, sees the



Setting of bolting system to the roof stabilising of machine (rear and front)

Sumping (conveyor and gathering head are moved forward too)

Shearing down

Shaving floor by retracting cutting system

Starting off

Figure 1.

ABM-20 slightly under-performing a non simultaneous cutting and bolting machine.

In an effort to address this problem, Voest-Alpine have developed the ABM-30. This machine has a shallower and larger apron to accommodate coal storage.

As a result of the different apron design, the temporary roof support and bolting rigs are positioned approximately 3m from the cutting face. (The present record for this machine operating in 'A' category conditions in South Africa is 105m in one production shift).

To compare the performance of simultaneous cutting and bolting machines (ABM-20 and ABM-30) with non simultaneous cutting and bolting machines, the development metres per hour in each category were compared to determine an operating performance ratio. This ratio is shown in Figure 4.

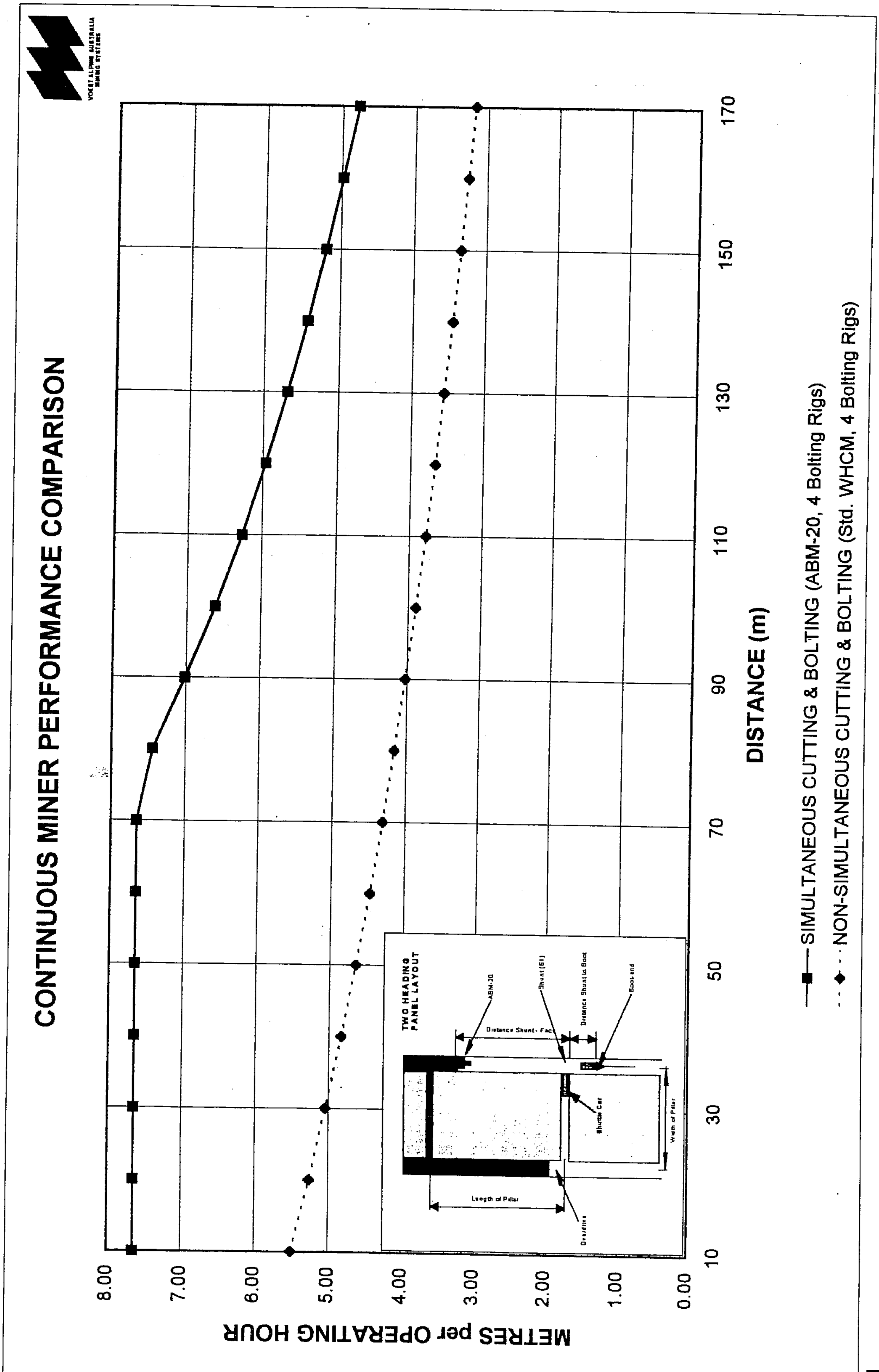
It indicates that for all categories the Alpine Bolter Miners will out-perform a standard non simultaneous cutting and bolting machine.

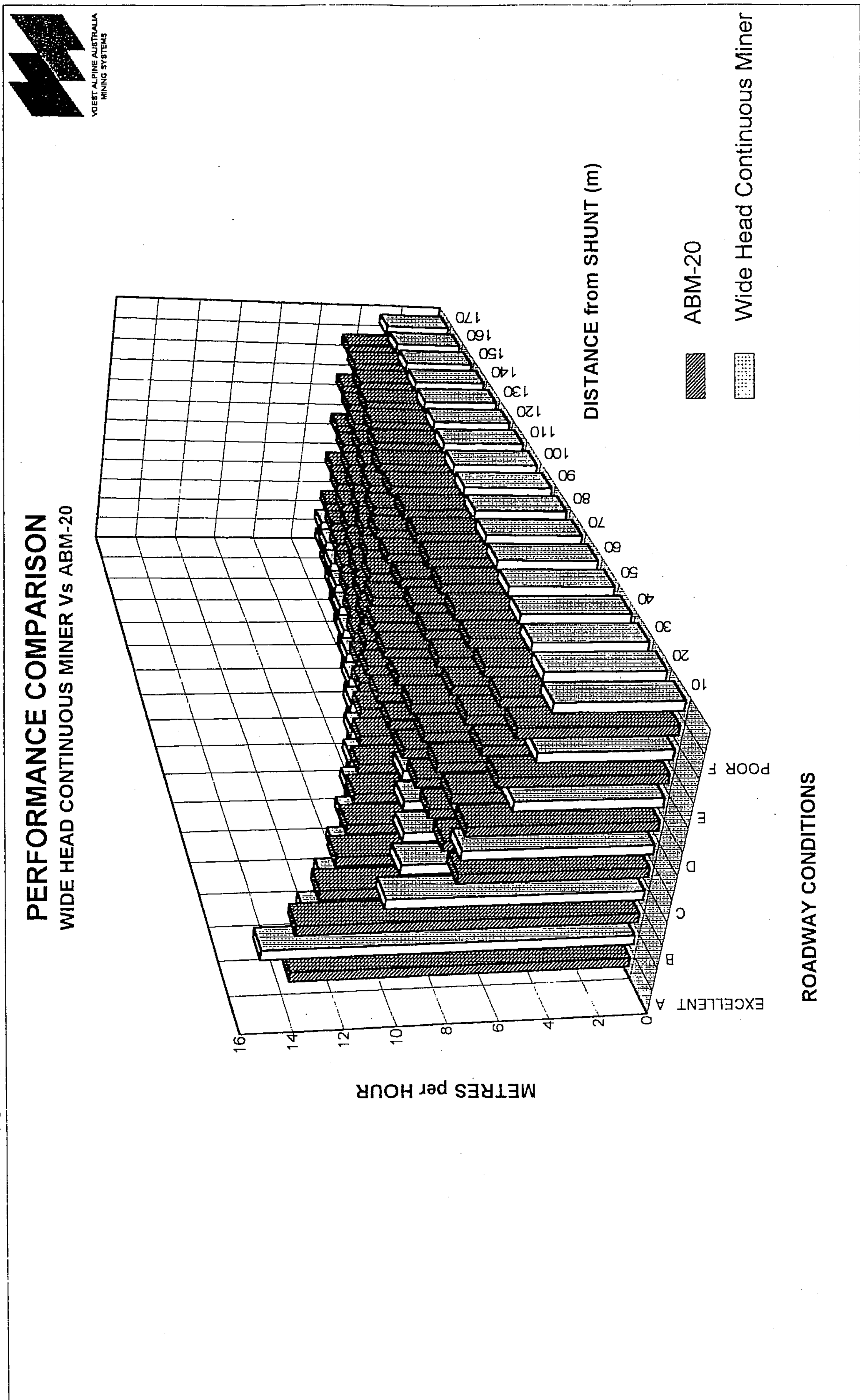
Depending on the category and the distance from the shunt, the development rate for ABM's can be up to double that of a standard wide head continuous miner.

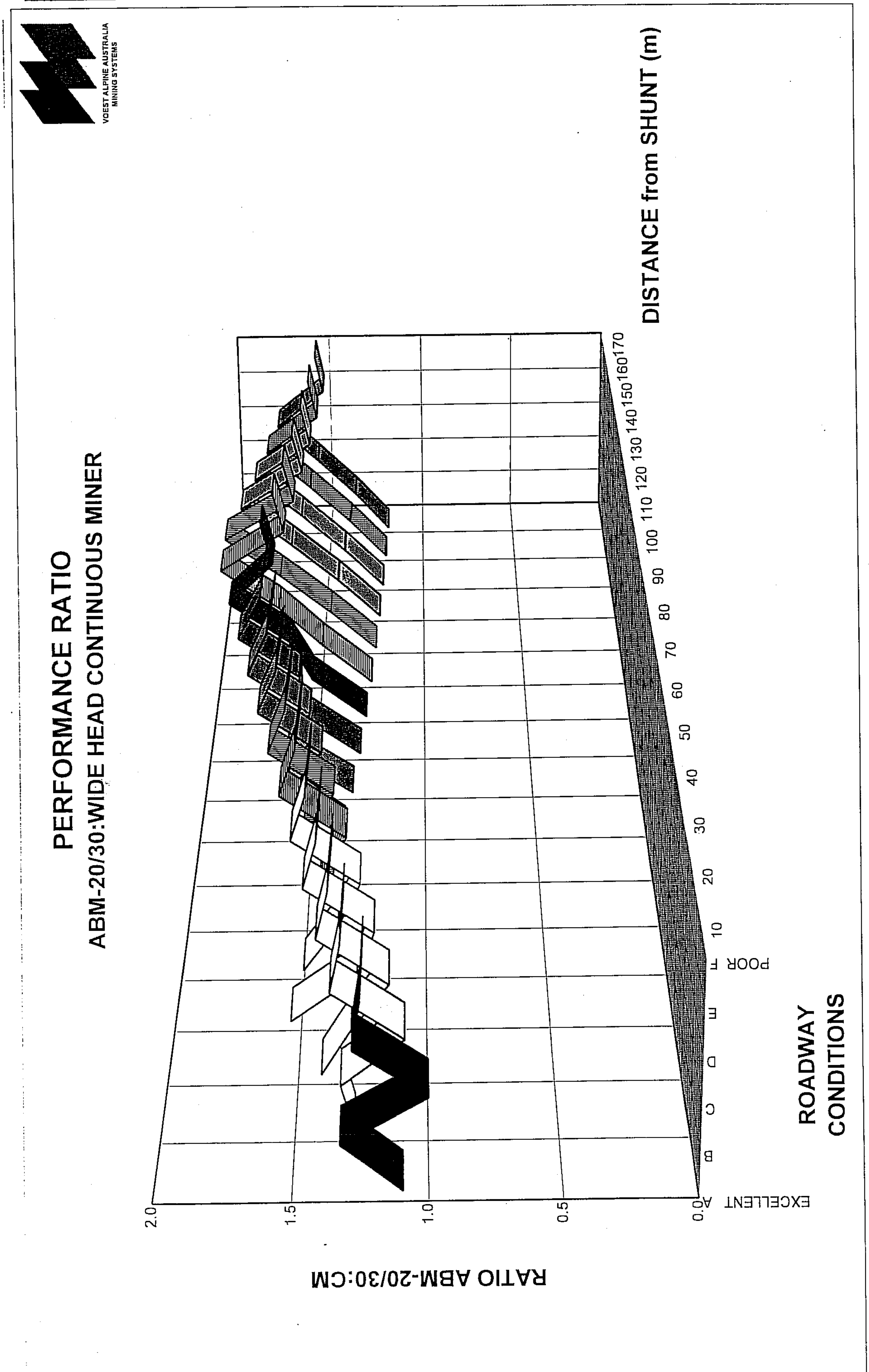
These figures assume that the machine can be operated at their full potential. In practice, however, people and people management play an integral role in determining machinery performance. This includes individual motivation and skill, but is also reliant on a work environment conducive to high performance.

Voest-Alpine's figures indicate that the largest gains for machinery performance can be obtained by concentrating efforts on uptime or cycle time.

The average downtime (including panel and machinery) for an operating panel is approximately 40%. While efforts will always be made to reduce this percentage, Voest-Alpine is also placing significant emphasis on the 60% uptime. The gains that can be made by concentrating efforts in each category are shown in Figure 5.


CONTINUOUS MINER PERFORMANCE COMPARISON
METRES per OPERATING HOUR
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
10
30
50
70
90
110
130
150
170
DISTANCE (m)
SIMULTANEOUS CUTTING & BOLTING (ABM-20, 4 Bolting Rigs)
NON-SIMULTANEOUS CUTTING & BOLTING (Std. WHCM, 4 Bolting Rigs)
TWO HEADING PANEL LAYOUT
ABM-20
Distance Shunt - Face
Shuttle Car
Width of Pillar
Length of Pillar


VOEST ALPINE AUSTRALIA MINING SYSTEMS
PERFORMANCE COMPARISON
WIDE HEAD CONTINUOUS MINER Vs ABM-20
METRES per HOUR
0
2
4
6
8
10
12
14
16
ROADWAY CONDITIONS
EXCELLENT A
B
C
D
E
POOR F
DISTANCE from SHUNT (m)
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
ABM-20
Wide Head Continuous Miner


VOEST ALPINE AUSTRALIA
MINING SYSTEMS
PERFORMANCE RATIO
ABM-20/30:WIDE HEAD CONTINUOUS MINER
RATIO ABM-20/30:CM
2.0
1.5
1.0
0.5
0.0
EXCELLENT A
B
C
D
E
POOR F
ROADWAY
CONDITIONS
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
DISTANCE from SHUNT (m)

ABM-20

PERFORMANCE MATRIX - "TYPICAL" COLLIERY

Metres per shift

TOTAL DOWNTIME (Mins) / CYCLE TIMES (Mins)

		16 (Av.)	12 (Peak)	8 (Target)
Target	45	17.3	23.0	34.5
	78	15.6	20.8	31.2
Av.	156	11.7	15.6	23.4

Source: Colliery Reports (assumes 390 mins per shift)

me, while the horizontal axis illustrates the potential of optimising the uptime. The ultimate solution shown in the top right head corner of the matrix, the result of efforts in both areas.

Voest-Alpine has recently helped develop a computer based real time monitoring system for its Alpine Bolter Miners that concentrates on 'process control' of the cycle time. This will assist in optimising machine performance, and help establish best practice operating procedures for all machine users.

Rowan Melrose

KEVIN HALL

Australian Longwall Pty Limited

SUPPLIER'S PERSPECTIVE

The purpose of this brief paper is to outline equipment manufacturer's objective for identifying products which are suitable for them to develop. The paper does not attempt to explore any aspects in depth, but to highlight significant points which may stimulate your thoughts.

Supplier's businesses are based on one or more of the aspects of design, manufacture, supply and service of products. These products may be hardware, software or pure service.

The objective of these products is to improve the productivity, safety and profitably of the end users business. Hence products must effectively fulfil the needs and wants of the user.

SUPPLIER'S OBJECTIVE

In the coal mining industry the objectives of the suppliers and mine operators are quite distinct. The mine operators objective is to create wealth by mining and selling coal. Supplier's objective is to take raw materials and add value to them to produce a product which can be sold for a profit.

For long-term viability, all businesses must return acceptable levels of profit to their investors.

Suppliers must focus on products and services which enhance the viability of the mine operator and the mine operator should strive to ensure that their suppliers are receiving sufficient returns to maintain long-term viability.

Some of the ways suppliers meet their objectives:

- Add or increase the value of raw materials by carrying out manufacturing processes on them, eg machining, fabrication.
- Provide expertise based on accumulated knowledge.
- Provide technical services in their field of expertise.
- Use modern, efficient manufacturing methods to keep the cost of manufacture as low as practical.
- Research and development of new products.
- Customisation of products to suit individual operators requirements.
- Provide services to support hardware supplied, eg training packages, service personnel, spare parts.
- Provide pure services such as on-call technical advice.
- Be technically and commercially competitive.
- Continual improvement of products based on operating experience.
- Seek features which enhance and differentiate their products from competitive units.
- Provide return on capital invested to shareholders.

NEW PRODUCT DEVELOPMENT

The main purpose of this workshop is to identify areas where products could be developed to increase the rate of roadway development in underground coal mines.

This I believe is an ideal environment to foster development of new products because of the interaction between the end user and the suppliers.

However, we must be mindful of the old adage 1% inspiration, 99% perspiration.

It is the 99% perspiration that needs to be understood and evaluated in the development of new products.

But in the 1% inspiration, people should not dismiss ideas because they think they are too simple or must have been thought of before.

Sometimes ideas seem simple to you because your the one who thought of them. The best ideas are mostly ***simple*** ideas.

Returning to the more difficult phase of how to implement an idea it is necessary to have at least one person to 'champion' the idea and have conviction to see it through to fulfilment. In fact, for effective product development I believe it is necessary to have at least two 'champions' — one from the supplier and one from the end user.

Evaluation and review techniques are critical for effective product development. Statistically in the order of one in thirty ideas committed for R & D evaluation actually make it right through to becoming a product.

The cost of R & D is high and funds available from both the supplier and industry is limited. Hence, the evaluation and review procedure must be effective and honest to ensure that the R & D dollar is spent in most affective areas.

It is poor R & D management to continue to invest money in project that may have demonstrated itself to be impractical part way through the development.

But now let us consider the projects that do reach successful completion.

In these cases the product must have a market, that is it must be useful and cost effective for the end user.

The project must have strong and effective project leadership from both the supplier and the user. Many products fail because the 'champion' or project leader on the users side is transferred from the project due to some immediate requirement in operations.

Products do take a significant time to develop.

The project must be focused and have clear objectives.

The hands-on operators who will be using the equipment must be involved to provide input throughout the project.

If the hands-on operators do not have involvement and a strong level of ownership, then even the best of ideas or products can be made to fail.

The product development must have a practical timescale.

PARTNERSHIPS

For individual products to be utilised to their optimum, it is necessary to consider the total environment or system necessary for effective roadway development.

This requires customisation of equipment to suit individual sites. This customisation should be carefully considered to ensure there are no radical changes to equipment which could affect its functions or reliability.

We are all individuals and want to feel that we have stamped our mark on a piece of equipment. We should be mindful of other peoples ideas and control or own egos to ensure the optimum equipment configuration is adopted.

No supplier I believe has all the necessary components to provide the optimum system for individual situations. Suppliers I believe have the opportunity to co-operate with each other to more effectively integrate individual pieces of equipment to provide a better system.

Effective co-operation in this area would relieve the pressure on the staff of the operator, allowing them to be more confident that the products being supplied will work together. This would be a further form of value adding to the products by the suppliers.

In conclusion, products must be developed to satisfy a real need of the end user. Products developed by suppliers in isolation based on a 'good idea' or on the belief that they can find a market are in most cases doomed to failure. Products require individuals from both sides to have commitment and be committed to the product development. Product development is a joint effort between the supplier and user.

Kevin Hall

TERRY O'BEIRNE

ACIRL, Brisbane

MOBILE BOOT ENDS - A REVIEW OF AUSTRALIA'S FIRST 3 UNITS

The mobile boot end (MBE) is an innovative device, capable of aiding productivity. In the initial 3 applications, and during a mid 1993 review by the author, their full potential was not seen, yet significant gains were made, and an indication of potential uses and problems gained. Oaky Creek gained around 30% by reducing time for regular belt realignments (on cross grades) and speeding up moves. Homestead included the MBE with a Matchappel collapsible belt structure for a gain of 25%, however the "bigger picture" benefits were small. The entire system was unfortunately not performing to expectation during the review period and has since been improved. Ellalong used the MBE in both continual advance and cyclic modes, giving a 35% gain in the initial panel, and from 10% to 30% in subsequent ones the author is aware of.

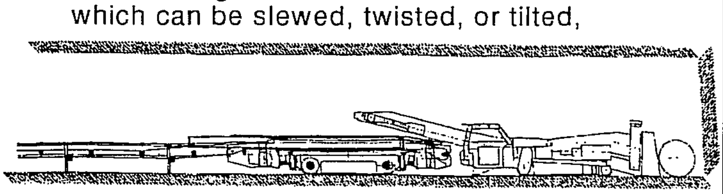
None of the mines reviewed had development rates that exposed a potential weakness of the MBE, that being its ability (because of width) to hamper access to the face. In addition, each colliery was able to introduce a shuttle car to mine the cross-cuts, and so the inflexibility of the MBE system was not fully displayed.

This paper is a synopsis of a comprehensive report for ACARP.

INTRODUCTION TO THE MOBILE BOOT END

The Mobile Boot End (Figure 1), as its name suggests, is a crawler mounted boot end, which can either remain static until a belt extension is required, or it can continually advance behind a continuous miner, pulling out the belt from a loop supply. The MBE can be loaded at one end only by a conventional shuttle car. The MBE has been technically described by several authors and so only a brief summary is provided here for completeness.

The Australian Longwall MBE has a crawler track arrangement located under a 'boot end' which can be slewed, twisted, or tilted,



MACHINE IN HEADING

independently of the crawler frame, to achieve optimum alignment for belt tracking. The MBE mass is around 35t and does not use any stacker jacks. It is hydraulically powered and controlled using a 37kW electric motor. Hydraulic controls can be placed in several locations. The MBE has a draw bar pull of 180kN which has proven adequate for advancing up to 2km of longwall panel conveyor belt. In some instances at Homestead however, it has proven inadequate to advance the Matchappel conveyor system.

At the outbye end of the MBE is a 'work area' in which 2 persons can stand whilst it is advancing and paying out belt from the storage loop somewhere in the panel belt. Provided there is sufficient clearance from the ribside, this area allows the persons to install all the belt structure and rollers as the MBE is advancing. Minimising the width of the MBE was a critical design objective, and a unit for a 1400mm belt is 2200mm wide. The extra width has impacted on access to the face in all applications. To date this has been overcome in various ways at each mine. More importantly however, because the MBE cannot 'turn' corners for cross-cuts, another machine like a shuttle car has to be introduced, creating its own difficulties.

INITIAL INSTALLATIONS

The authors' review was undertaken in early-mid 1993, and so the information on them is now dated. However, many of the observations are valid and they suggest some directions the industry should pursue (the basic objective of this seminar). Other speakers at this seminar will describe more recent applications of the MBE.

Oaky Creek

Oaky Creek introduced the MBE onto its 1800mm panel belt as part of a package of improvements. It succeeded in achieving 2 purposes:

a) speeding panel move times, and
b) simplifying the ongoing alignment of the tail end on a cross-graded floor.

The Oaky Creek 2 heading development system at the time utilised the traditional 1 continuous miner and 2 shuttle cars. The boot end (MBE) is unable to accept side loading, so it was placed outbye the nearest cross-cut, requiring shuttle cars from one heading to wheel around the pillar and approach end on. When driving the belt heading, only one shuttle car is used.

The result in its first panel of use was a gain in metres/worked shift of 30% over the prior system using a Stamler feeder and fixed boot end. There is inadequate data to conclusively prove if all this change was solely due to the MBE introduction, as other changes were made at the same time. Development productivity has continued to increase at the mine.

Ellalong Colliery

Ellalong Colliery acquired the first MBE, unused from Cook Colliery. The mine has traditionally had poor development conditions, necessitating considerable roof and rib support to be erected at the face. A relatively soft floor was continually damaged by wheeling cars.

This study covered the MBE in 2 panels, and the data are summarised below. The MBE has been very successful at Ellalong, however there were problems with spillage on loading and access around the MBE. Their generally low development rates however (caused by extensive bolting) probably masked the extent of these problems.

The MBE was first placed in placed in the LW10 Maingate directly behind a Joy 12CM20 Miner and followed its advance, pulling belt out of a 170m loop take up. The Maingate showed a significant advantage over the companion road (trackgate) driven with shuttle cars, as the table below illustrates:

In LW10 the Maingate (MBE) progressed so well in comparison to the shuttle car based companion road), that it resulted in the MG being far in advance of the last cross-cut and it had to be finished under single entry conditions. This imbalance created several unique scheduling problems, and effectively meant the entire system was limited by the slowest unit. Therefore, the MBE did very little to advance the overall rate of development of the 2 headings. Using the MBE also resulted in a very good road surface by the elimination of the shuttle cars. In this first application the conveyor structure was Meco-Garland, with 1200mm wide belt fitted. The structure was advanced through the shift by the crew. When the miner stopped for bolting, two of the crew advanced the MBE sufficiently to install 1 X 3m section of structure. With the relatively slow advance rate of only 4.6 m/shift, this task did not present any undue problems.

Table 1. Ellalong Longwall 10 & 11 Performance.

GATEROAD	WORKED SHIFTS	METRES	M.P.U.S.	MINS/ METRE
LW10 MAINGATE (MBE)	233	1076.2	4.6	62.8
TRACKGATE (SC'S)	243	933.5	3.84	81.3
LW11 MAINGATE (MBE)	223	818.7	3.5	82.3
TRACKGATE (SC'S)	239	913.4	3.9	87.2

In LW11 a Joy 22 shuttle car was initially used between the MBE & miner, with the MBE & belt advanced daily. This was done to allow the faster developing MBE road to drive the cross-cuts, but overall, this was unsatisfactory and the system returned to original specification. The table above shows the productivity performance of these gateroads. A detailed analysis showed a marked gain once the shuttle car was removed.

Homestead

Homestead mine (Wambo) introduced a new development system for Longwall 9 Tailgate and Trackroad. This comprised 4 main items:

a) Australian Longwall MBE

b) Matchappel quick set conveyor (concertina conveyor)

c) ACE loop take up system with PLC winch control

d) Pantechnicon mounted services

This was the first concurrent application of all these technologies/machines in Australia, though all of them had been separately demonstrated over the past couple of years. The system was first installed in a conventional 2 heading gate road, 19 West.

The mining system required the Return road (No. 1 heading) to be advanced 2 pillar lengths with the MBE towing and expanding 5 "sleds" (a sled being a sub-unit containing 44m of structure) of Matchappel belt system. A Joy HM9 miner cut a 5.3m wide place, and loaded directly into the MBE. The MBE begins the cycle very close to the HM9, allowing the HM9 to advance the length of its loading tail (around 3.5m) before the MBE has to advance.

Once the HM9 has advanced the required distance, the MBE is reversed back to the nearest cut-through, pushing and reloading some 44m of Matchappel structure ahead of it. At the cut-through, old sleds are removed and full ones reinserted, ready for the next cycle. Services are also advanced at this time.

A summary of results for 19 West is shown in the Table following:

Homestead, 19 West Performance

	MBE road (No. 1 heading)	Shuttle car road (No. 2 heading)
minutes/metre, average	20.5	25.6
minutes/metre, best	13.5	18.7
metres/shift, average	10.8	8.5
metres/shift, best	31.0	22.5

All this data clearly shows an average gross productivity gain of 25% by using the Matchappel system plus the MBE. However, the real gain to the mine was significantly less, as these values do not reflect the extra shifts taken to install the additional 180m of Matchappel belt structure and air/water/power reticulation systems. Since this study the author is aware of many changes to the system to address problems with the MBE pulling out the Matchappel structure, and the time taken to reload the structure.

IMPROVEMENTS FOR 2 HEADING DEVELOPMENT

As this workshop is attempting to consider the future development needs of the Industry, it is appropriate that the historical position with the MBE's, and other devices be put into context, and some views offered as to where success may lie.

Heading development has absorbed an extraordinary amount of attention over the past 10 years, spawning developments such as high capacity roof bolting, continuous haulage in various guises, single pass miners, and continuous cutting and bolting machines, and now the MBE. Despite the generally positive results from these developments, most Australian longwall have suffered to some degree by the inability of the development to keep pace with longwalls retreat. Of course, longwall extraction technology is also increasing, with daily output rising about 10% per year recently. In most locations, mining conditions are also deteriorating. Despite these comments on individually significant developments, they have been just that, individual. Little real effort appears to have been placed in the overall development system.

To help consider where the MBE and other developments may 'fit' into an overall picture for development needs, a number of simulation models of development systems were created. The objective of the simulations was to identify the sensitivity of development systems to various variables. In this case the variables examined included: haulage system, systems availability, shuttle car wheel speed & loading rate, bolting time and panel move time. The full details of the simulations are provided in the reference, however the following table (page 4)is presented as a synopsis, and hopefully will provide some 'food' for discussion.

SUMMARY AND CONCLUSIONS

This study confirmed the MBE is a useful device however its initial trials reported here have not shown its full potential benefits or problems. In most applications the MBE has created some difficulty in moving supplies to the face, though the full impact of this will not be seen until the unit is employed in development with much higher advance rates. In the 3 mines, the MBE has created a range of difficulties with clean unloading of cars, at worst, promoting bogging at the boot end. This is unfortunate because it otherwise improves road conditions by reducing wheeling. No side loading is possible onto the MBE, instead a belt side loader offered by the manufacturers. Its original design is far from ideal, with spillage from it and ease of moving needing to be solved.

The MBE has to be matched with a continuous cutting and bolting machine to find its true limits and range of application. In a 2 heading development this will probably mean the roadway using the MBE will eventually operate under single entry conditions, as its development rate will outstrip that of the companion road. Other papers on Ellalong colliery at this workshop will address this issue.

The MBE should be applied behind a true continuous mining machine at the earliest possible opportunity. The demonstration site should ideally be one with demonstrably high unit-shift availabilities, and not unduly constrained by support practises. The objectives should be not only to increase development rate at the host site, but also to identify other weaknesses bought about by very rapid development, and how a conventional panel belt system and MBE will react to continuous advancing at a high rate. This would also provide an opportunity to examine the many issues associated with supplies and servicing logistics to a rapidly developing face. ACARP should actively canvass interested mines for a demonstration project which would have wide ranging impact on the entire underground industry.

The MBE was unfortunately hampered when linked to the Matchappel belt system. Together they could form the hub of a relatively cheap and simple continuous face haulage system and ACARP should encourage development to achieve that end. All previous attempts at continuous face haulage have essentially failed because of exotic technology. The MBE uses very simple technology and this should be

Table - Summary of Modelling Results.

Reduction in 1 pillar development time	How	Base Case (worst scenario)	Research needs
3.2%-4.7%	2.5t/minute faster loading by miner	5t/minute	contrast reliability with loading rate
6.5%-8.5%	1 minute less spent bolting each metre of roadway	12.5 minutes spent for each metre advance, in the form of 25 minutes for a 2m cut out	* less face bolting * faster roof bolting * easier roof bolting * solve rib bolting hassles * more reliable bolters
7%	increase shuttle car wheel speed by 50%	75m/minute	complete re-think, maybe eliminate rubber tyred cars
8%-10%	1 shift less panel move time	5 shifts/pillar of 100x30m	* supplies logistics study * ability to duplicate gear * better vent and elec supply allowing longer pillars * rethink crewing applied to moves * better pit planning tools
9.2%-10%	improve unit shift availability 10% (points)	55% availability	* adopt world best practise in reliability engineering * buy/specify better gear * improve repair & maintenance 'tools'
9%-10%	add a 10 tonne surge car without increasing panel move time or decreasing system availability	2 shuttle cars	* rethink size of surge cars and the way they can be moved about * what else can replace a car?
20%-28%	change from 2 shuttle cars to continuous haulage with no loss of reliability of increase in panel advance time	2 shuttle cars	* engineer FCT's to be more reliable and less costly to own * expand capacitates of MBE & Matchappel
41%-76%	don't mine cut-throughs and don't incur any delays because of this	mine cut-throughs	* more effective auxiliary vent systems * simple cut-through drivage machine * improved supplies logistics * more reliable face equipment

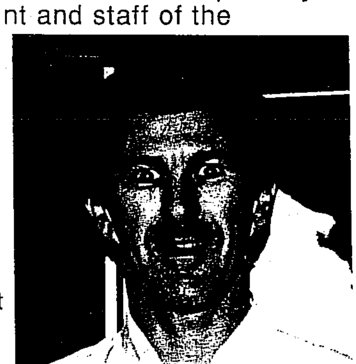
exploited to its limit. The Mobile Boot offers a unique capability as a platform for outbye work or services. For example, it could possibly be used as a bolting platform, or as the site for the auxiliary fan. Applications like this need to be actively canvassed and properly evaluated for their impact on mining system efficiency. Such reviews are probably best performed by individual mines. In a situation of two parallel single entries the use of the MBE as a 'platform' could take on a greater importance.

There is currently extensive research into increasing the efficiency of the mining process and machines. This however only applies whilst they are working. This project has demonstrated how other issues like equipment reliability and panel move times play a major role, sometimes greater than other issues which are being extensively researched.

All roadway development is currently hampered by panel advance times, and will be more so as the average development rate increases (i.e. it will effectively cost more to be not producing). There appears little practical research being undertaken to address the need to increase the efficiency of moves, and so this does not catch the industry unawares, studies should be instigated by ACARP as a matter of some priority.

ACKNOWLEDGMENTS

This project was funded by ACARP and initially scoped by a group from the Roadway Development Taskforce. AMIRA provided ongoing project liaison. Thanks are especially due to the management and staff of the Ellalong, Oaky Creek and Wambo and Australian Longwall for their assistance and permission to undertake the review.



Terry O'Beirne

REFERENCE

Critique of the Mobile Boot End Report on ACARP Contract ACIRL Report CMU2224, July, 1993.

ROADWAY DEVELOPMENT SYSTEMS - B

This session is the second part of Roadway Development Systems and perhaps addresses more where we are going than where we are at present.

The fact that roadway development is the most significant constraint on underground mining has been recognised by this Workshop. I am pleased with the term Roadway Development Systems, because a development panel is a system consisting of the cycles of mining, roof support, coal conveying and advancing infrastructure.

The first three of these cycles are addressed in this session with papers on continuous miners which can mine coal while simultaneously supporting the roof and continuously conveying the coal. We are fortunate that today we have a choice of such machines available which we did not have ten years ago. The status of the development of an automatic roof bolter is also covered which will greatly assist in maximising the efficiency of any of these machines.

I am pleased also to say that the last cycle, ie advancing the infrastructure, is also addressed. If we are successful in improving development rates will we be able to support the panels infrastructure? Can your operation handle two or three belt moves per week instead of one together with the advance of pipes, ventilation and power? Are we a dog chasing its tail who will not know what to do when we catch it?

Papers of interest here include the use of the mono rail at Oaky Creek and the one describing the concept of 'Simultaneous Development Systems'.

The tunnelling industry often takes on projects involving kilometres of single entry drives where the infrastructure is carried with them. They have contributed a paper to this session on where they see their technology is applicable to our industry. Our breakfast speaker, Allan Rossiter, will also address us on tunnelling.

Gentlemen, I trust you will enjoy what is intended to be an informative and interesting program of presentations.

SEAN EGAN
Oakbridge Ltd

BRUCE JARRETT

Oaky Creek Coal Pty Ltd

DEVELOPMENT OF MONORAIL SYSTEMS AT OAKY CREEK No.1 COLLIERY TO ENHANCE THE TWO HEADING SUPER SECTION DEVELOPMENT CONCEPT & LONGWALL MINING OPERATIONS

The utilisation of an Australian Longwall Mobile Boot End for its intended purpose required the re-thinking in the way the longwall services would be handled. This then gave the opportunity to develop a Monorail System with face services and ventilation requirements for the development panel.

BACKGROUND

Oaky Creek Coal Pty Ltd is a wholly owned subsidiary of Mount Isa Mines Holdings Ltd operating a modern underground coal mine in the Bowen Basin of Central Queensland.

The Colliery utilises:

1. Development

(a) Two (2) Jeffrey 2048 Continuous Miners

(b) One (1) Jeffrey 1036 Continuous Miner

(c) One (1) Joy 12CM20 Continuous Miner

(d) Six (6) Joy 15SC Shuttle Cars

(e) Two (2) Australian Longwall Mobile Boot Ends

2. Longwall

(a) Longwall International 800+ 4 Leg Chock Shields

(b) Longwall International Armoured Face Conveyor with twin 34mm inboard chains

(c) Longwall International Swivel Beam Loader with twin 26mm inboard chains

(d) Klockner Becorit Crusher

(e) Longwall International Waltzing Matilda Boot End

(f) Anderson AM500 900 kw DERD Shearer

3. Coal Conveyancing

(a) 1600mm Trunk Conveyors rated at 4200t/hour

(b) 1500mm Development/Longwall Conveyors rated at 3500t/hour

THE CONCEPT

The concept of utilising the mobile boot end to its full potential in gateroad development panels required the movement of the 1500mm panel belt conveyor structure to the centre of the roadway. In previous panels the belt was situated close to the longwall block to allow the pantechnicon to be installed in the conveyor road.

The existing monorail concept which carried 350m of type 260.11 70mm^2 H.V. cable, 32mm Fras air hose, 32mm Fras solcenic and 75mm raw water hose would not fit in the conveyor heading after moving the conveyor to the centre of the roadway.

Monorail systems were investigated and found to be heavy and cumbersome to handle. We then decided we would adhere to the existing light weight monorails would be utilised and the trolleys would be designed to handle all longwall drive motors, cables, shearer cable and associated services. Macquarie Manufacturing, who supplied the original monorail and trolleys, were again called on to assist and with close liaison with Oaky Creek the longwall system gradually started to take shape on the drawing board.

At this early stage, the thought of the extra work of removing the cables and reinstalling during longwall transfers was imminent. The utilisation of the monorail installed while the development panel was advancing became a distinct possibility. This then became reality when the super panel concept of two continuous miners operating in tandem would be used. Trolleys were then designed to carry flexi ducting ie. miner cables and water hoses for the inbye system for ventilation of the working faces and associated services and a 100mm water line, DCB feeder cables and 50mm Fras air line on the outbye system in the companion roadway to the conveyor heading.

THE METHODOLOGY

As stated previously the trolley design became critical. This necessitated the installation of a surface monorail system at the Macquarie Manufacturing factory at Rathmines in New South Wales for compatibility trials and testing of monorail trolleys. This proved an intricate part of the systems development. The monorail beams therefore had to be compatible for the development and longwall systems. The distribution and control boxes were mounted on an air track and the auxiliary fan was mounted on a mine toad. This was to provide ease of movement during section advancement.

DEVELOPMENT SYSTEM INSTALLATION

The initial installation of the 'A' Heading system required sufficient panel development to be completed to allow the monorail beams to be installed from the substation cut-through to the distribution and control box position. The monorail beams and services were then installed from the substation to the DCB approximately 150m inbye. This allowed the 'B' Heading companion roadway system to be installed from the auxiliary fan and concertina to allow production to begin per mining sequence.

MINING SEQUENCE

1. Coal production in 'B' Heading as a priority to the 92m mark, when a breakaway to the left is taken to ensure the cut-through is on 99m centre from the previous cut-through. Breakaway is to be driven to a depth of 6.5m. The miner is to be pulled back and 'B' Heading is to be driven to the 103m mark from the centre of the previous cut-through. Flexi ducting is pulled out to a point which allows attachment of temporary flexi ducting to the tee piece on the monorail.
2. Flit the continuous miner back to the previously started cut-through and drive to a distance of 29m from the centre of 'B' Heading.
3. On completion of the cut-through flit the miner back to 'B' Heading and continue driving the face to the 150m mark. All monorail services will be fully extended.
4. 'B' Heading miner is to be maintained and the heading cleaned ready for the belt extension.
5. 'A' Heading is to be driven to the 85m mark where the DCB cut out is to be taken to a depth of 3m and fully supported.
6. Continue driving 'A' Heading and overdrive to the 140m mark, holing the previously driven cut-through at the 99m mark. All monorail services will be fully extended.

SECTION MOVE

1. Set up shuttle cars in overdrive and using an Eimco clean 'A' Heading.
2. (a) Disconnect 'A' & 'B' Heading miner cables from the DCB's.
   (b) Disconnect fan cable from DCB's.
   (c) Disconnect the mobile boot end cable from DCB's.
   (d) Disconnect water and air hoses at outbye intersection from the monorail manifolds.
   (e) Disconnect two (2) feeder cables from DCB's.
3. Using air motors concertina 'A' Heading flexi ducting, miner cable, water and compressed air hoses, with the outbye end to be positioned at the new DCB cut out position.
4. Reconnect flexi ducting via inbye cut-through using elbow assembly to rigid vent line.
5. Move DCB to the new position and reconnect inbye services ie. miner cables etc.
6. Advance outbye monorail services to the new DCB position and reconnect DCB's, connect air and water to monorail manifolds.
7. In 'B' Heading:
   (a) Disconnect air and water hoses at outbye cut-through.
   (b) Disconnect lengths of rigid ducting (hang in cut-through) at inbye and outbye cut-throughs from both tee pieces.
   (c) Concertina 'B' Heading flexi duct and reconnect outbye tee piece via inbye cut-through to rigid ducting running through the cut-through via the telescopic tube.
   (d) Reconnect the miner cable, air and water hoses via inbye cut-through.
8. Run auxiliary fan inbye to the new location.
9. Run mobile boot end cable around, couple and carry out belt extension.

10. Set up shuttle cars.

LONGWALL MONORAIL SYSTEM

The development of the longwall monorail system came out of the MM Cables type 260.11 cable being unsuitable for monorail use. The initial system used at Oaky Creek utilised the above cable which was difficult to install, difficult to retrieve and virtually blocked the walkway along the conveyor belt due to the inflexibility of the cable.

The initial development required the sourcing of a suitable cable to be installed to the monorail carriages to enable the conveyor belt to be moved to the centre of the 'B' Heading, allow sufficient room for the monorail installation and also a walkway to allow conveyor belt inspections.

A MM Cables type 241.3 monorail cable was selected to bring the power requirements for the longwall from the outbye pantech to the longwall drive motors and shearer. With the suitability of this cable it was then the development of the monorail carriages to carry the services utilising the light weight monorail beams.

This development took place at Macquarie Manufacturing with the guidance of Oaky Creek and again the surface mock up ability was invaluable. The major requirements of the monorail beams was to move from 'B' Heading to 'A' Heading around a 90° bend utilising a type of points system where special straight and bend rails were developed.

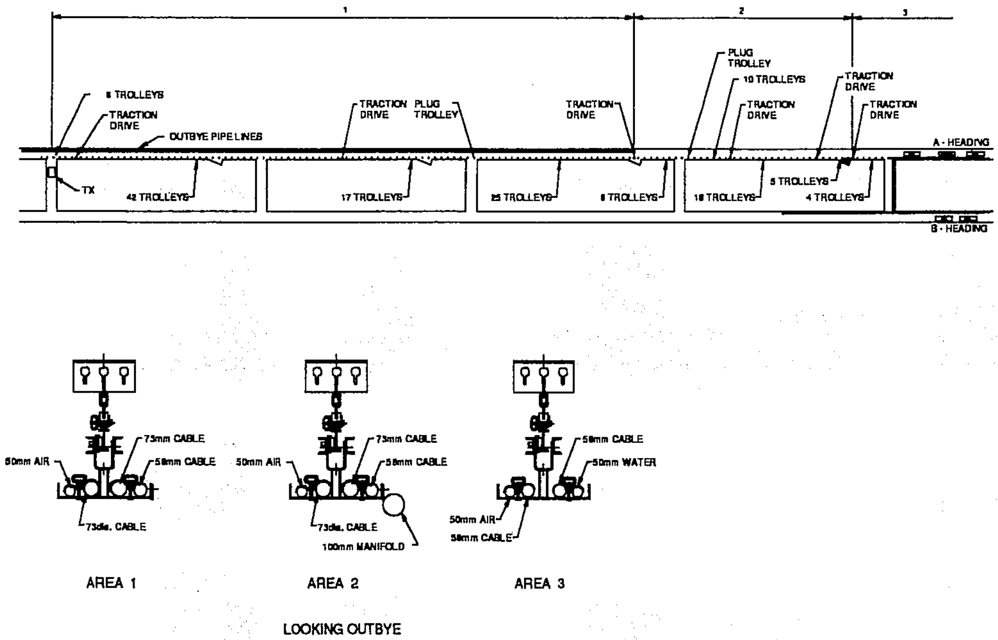
The pantech development was done on the Australian Longwall CAD system in Mackay. This required the sourcing of wheels, redesign of the pantechnicon sleds to stop flexing of the steel, joining link modification, wheel protector crash bars and pantechnicon pump and tank relocation. All sled modifications were carried out during the Longwall Four (4) to Longwall Five (5) face transfer.

A MM Cables PLC enclosure was installed along the face next to the maingate drive. This unit communicated with the PLC on the control pantech via a data highwall twin axial belden type cable which allows the switching of the contactors in the substations. The inbye unit receives information from the methane monitors, DAC 321 lockout and communication system, and the status of the outbye pantech substations and pumps and receiving information from the surface giving conveyor belt monitoring and status which is screened via a Allen Bradley T70 computer.

LONGWALL SERVICES RETRACTION

1. Disconnect the longwall drive and shearer cables from the pantech manifold and No.1 substation.
2. Disconnect control and communication cables.
3. Disconnect face high pressure hydraulics, face hydraulic return line and shearer high pressure water line.
4. Disconnect high voltage cable from No.3 substation after H.V. isolation procedure has been carried out.
5. Utilising the air drive motors move the monorail carriages back up the cut through and then inbye until the last carriages have past the bend section and lock the brake on the air drive motor.
6. Remove the curved section of rail and install the straight section joining the monorail beams in 'B' Heading.
7. While Item 6 is being carried out utilise the Domino Myne Dozer, 936 Eimco or a 913 Eimco to pull the pantechnicon to the next pantech location ensuring the pantech manifolds are in the correct position relative to the monorail beams coming through the cut through.
8. Using the monorail carriage air drive motors move the services to the pantech position.
9. Reconnect all services to the appropriate manifolds.
10. Reconnect the high voltage cable to the No.3 substation and repower the longwall substations as per procedure.
11. Check all services and the longwall drives are sequenced and running correctly.
12. Longwall production.

ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.



TOLERANCES:UP TO 300 ±1.5 U.N.O.
300 TO 900 ±2.0 U.N.O.
OVER 900 ±3.0 U.N.O.

REMOVE ALL SHARP EDGES & BURRS.

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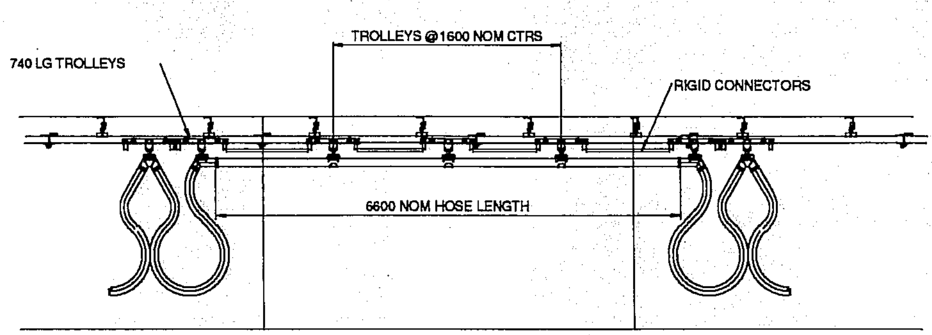
OAKY CREEK OUTBYE SERVICES MONORAIL ARRANGEMENT			
CHECKED: .		SCALE: N.T.S.	DATE: 18/10/93
DRAWN: PECK	REV. A		DRG No. MM10100

ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.



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300 TO 900 ±2.0 U.N.O.
OVER 900 ±3.0 U.N.O.

REMOVE ALL SHARP EDGES & BURRS.

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OAKY CREEK OUTBYE SERVICES CUT THROUGH ACCESS ARRANGEMENT			
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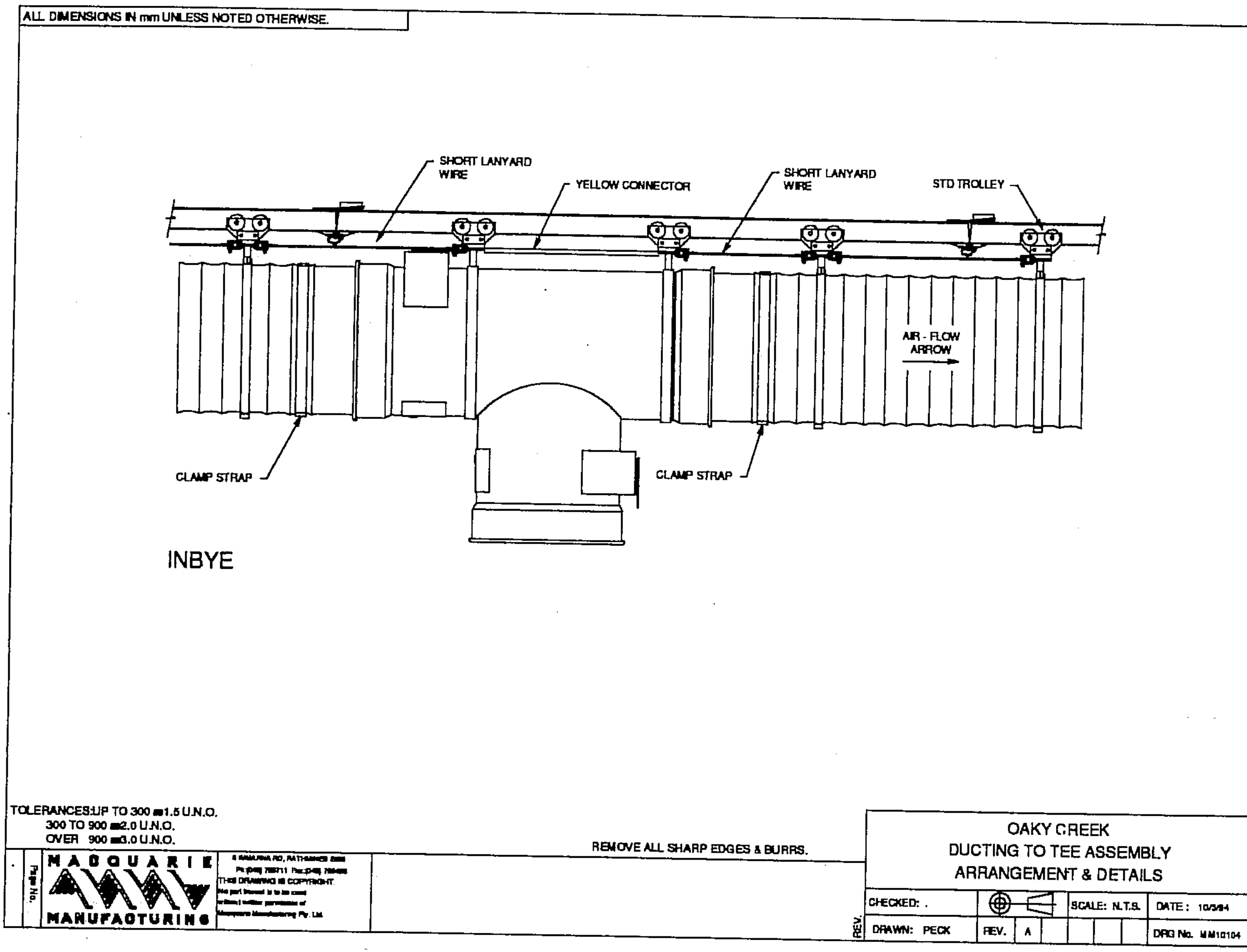
ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.
SHORT LANYARD WIRE
YELLOW CONNECTOR
SHORT LANYARD WIRE
STD TROLLEY
AIR - FLOW ARROW
CLAMP STRAP
CLAMP STRAP
INBYE
TOLERANCES:UP TO 300 ±1.5 U.N.O.
300 TO 900 ±2.0 U.N.O.
OVER 900 ±3.0 U.N.O.
REMOVE ALL SHARP EDGES & BURRS.
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OAKY CREEK
DUCTING TO TEE ASSEMBLY
ARRANGEMENT & DETAILS
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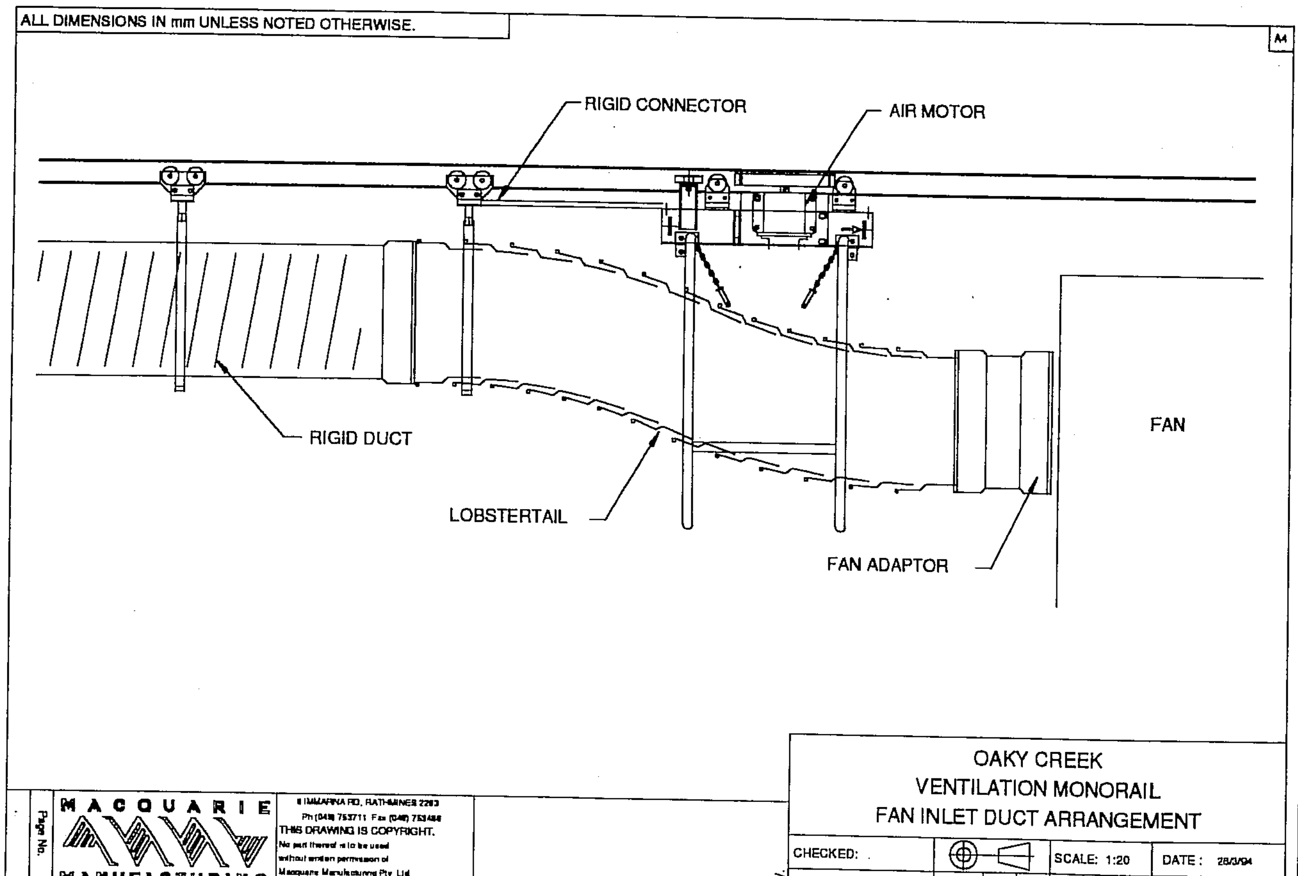
DATE: 10/3/94
DRAWN: PECK
REV. A
DRG No. MM10104


ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.
A4
RIGID CONNECTOR
AIR MOTOR
RIGID DUCT
LOBSTERTAIL
FAN
FAN ADAPTOR
MACQUARIE MANUFACTURING
6 IMMARNA RD, RATHMINES 2283
Ph (049) 753711 Fax (049) 753488
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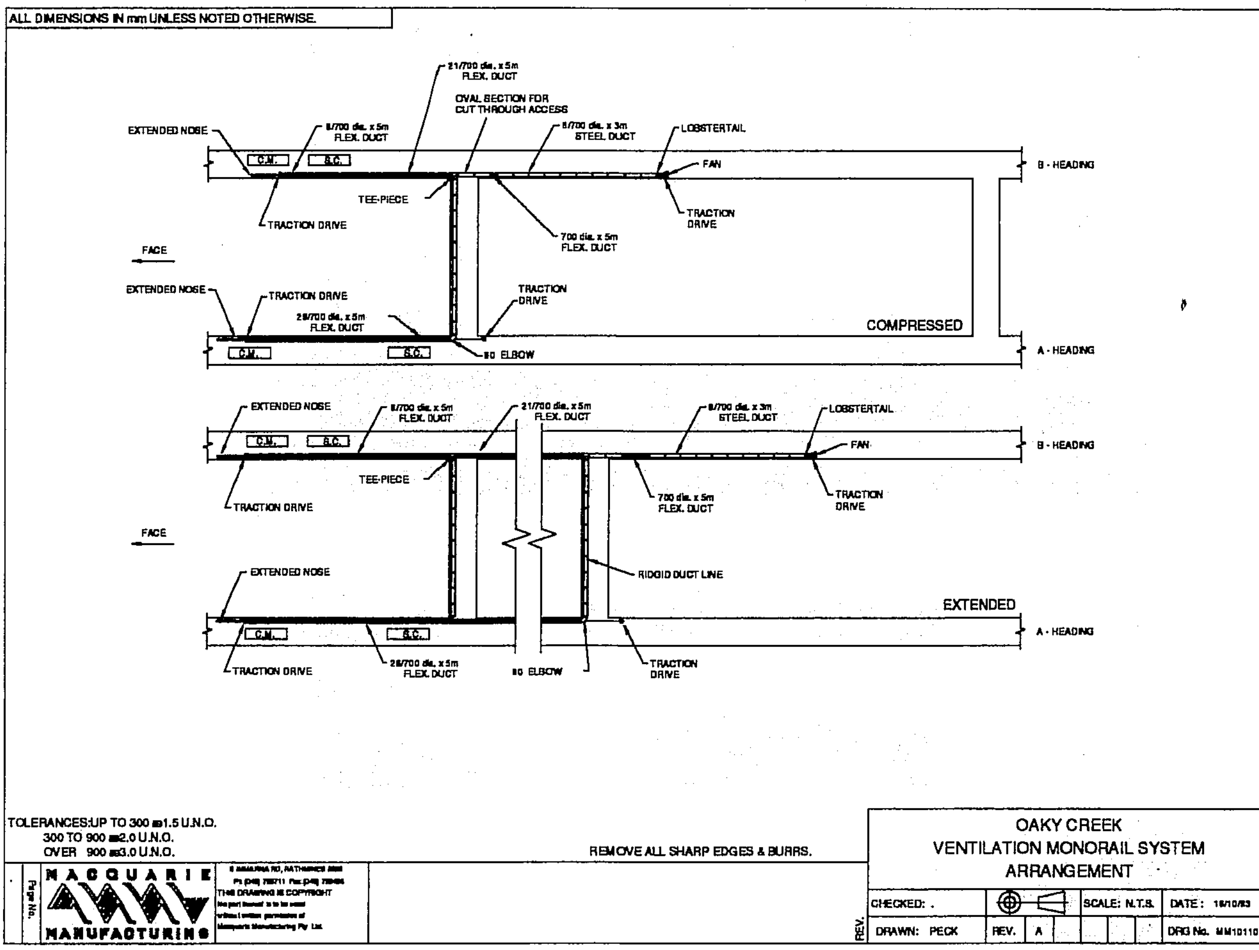
OAKY CREEK
VENTILATION MONORAIL
FAN INLET DUCT ARRANGEMENT
CHECKED:
SCALE: 1:20
DATE: 28/3/94
DRAWN: R.T.H.
REV. A
DRG No. MM10087


ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.
21/700 dia. x 5m FLEX. DUCT
OVAL SECTION FOR CUT THROUGH ACCESS
EXTENDED NOSE
8/700 dia. x 5m FLEX. DUCT
8/700 dia. x 3m STEEL DUCT
LOBSTERTAIL
FAN
B - HEADING
C.M.
S.C.
TEE-PIECE
TRACTION DRIVE
700 dia. x 5m FLEX. DUCT
FACE
28/700 dia. x 5m FLEX. DUCT
90 ELBOW
COMPRESSED
A - HEADING
RIGGID DUCT LINE
EXTENDED
TOLERANCES:UP TO 300 ±1.5 U.N.O.
300 TO 900 ±2.0 U.N.O.
OVER 900 ±3.0 U.N.O.
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OAKY CREEK
VENTILATION MONORAIL SYSTEM
ARRANGEMENT
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DATE: 18/10/93
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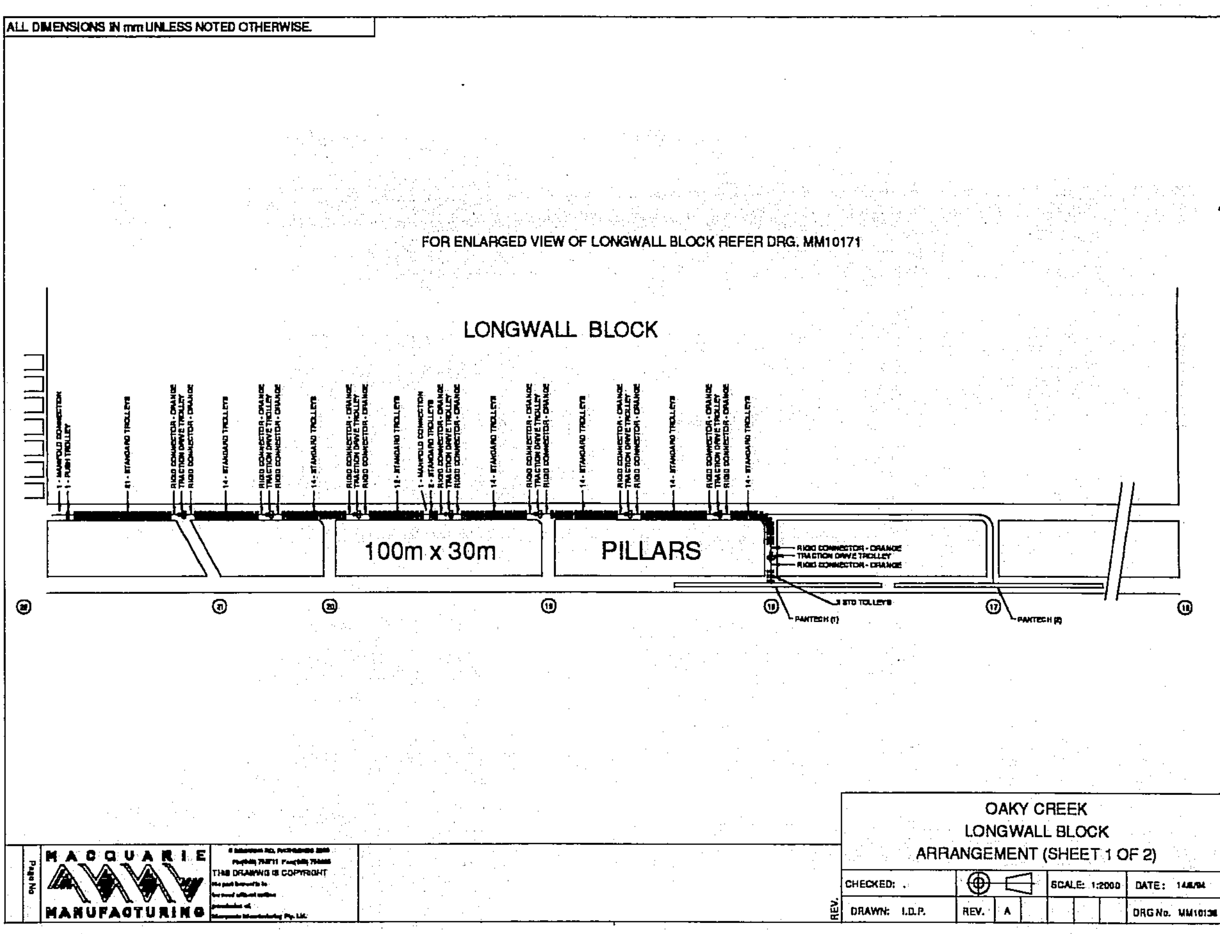
DRG No. MM10110


ALL DIMENSIONS IN mm UNLESS NOTED OTHERWISE.
FOR ENLARGED VIEW OF LONGWALL BLOCK REFER DRG. MM10171
LONGWALL BLOCK
100m x 30m
PILLARS
RIGID CONNECTOR - ORANGE
TRACTION DRIVE TROLLEY
RIGID CONNECTOR - ORANGE
3 STD TOLLEYS
PANTECH (1)
PANTECH (2)
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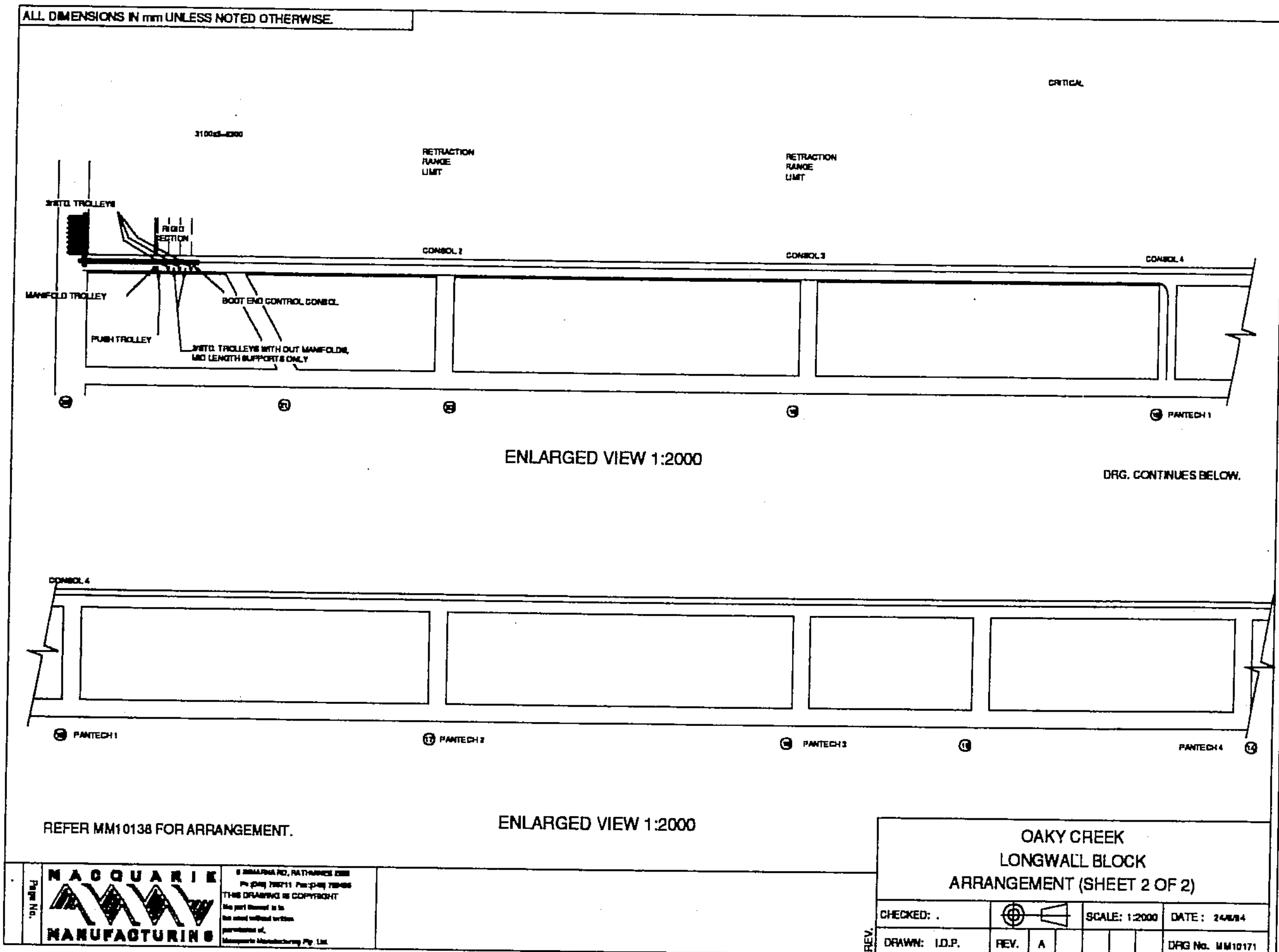
OAKY CREEK
LONGWALL BLOCK
ARRANGEMENT (SHEET 1 OF 2)
CHECKED:
SCALE: 1:2000
DRAWN: I.D.P.
REV. A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CRITICAL
RETRACTION
RANGE
LIMIT
RETRACTION
RANGE
LIMIT
RIGID
SECTION
CONSOL 2
CONSOL 3
CONSOL 4
MANIFOLD TROLLEY
BOOT END CONTROL CONSOL
PUSH TROLLEY
MID LENGTH SUPPORTS ONLY
PANTECH 1
ENLARGED VIEW 1:2000
DRG. CONTINUES BELOW.
CONSOL 4
PANTECH 1
PANTECH 2
PANTECH 3
PANTECH 4
REFER MM10138 FOR ARRANGEMENT.
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LONGWALL BLOCK
ARRANGEMENT (SHEET 2 OF 2)
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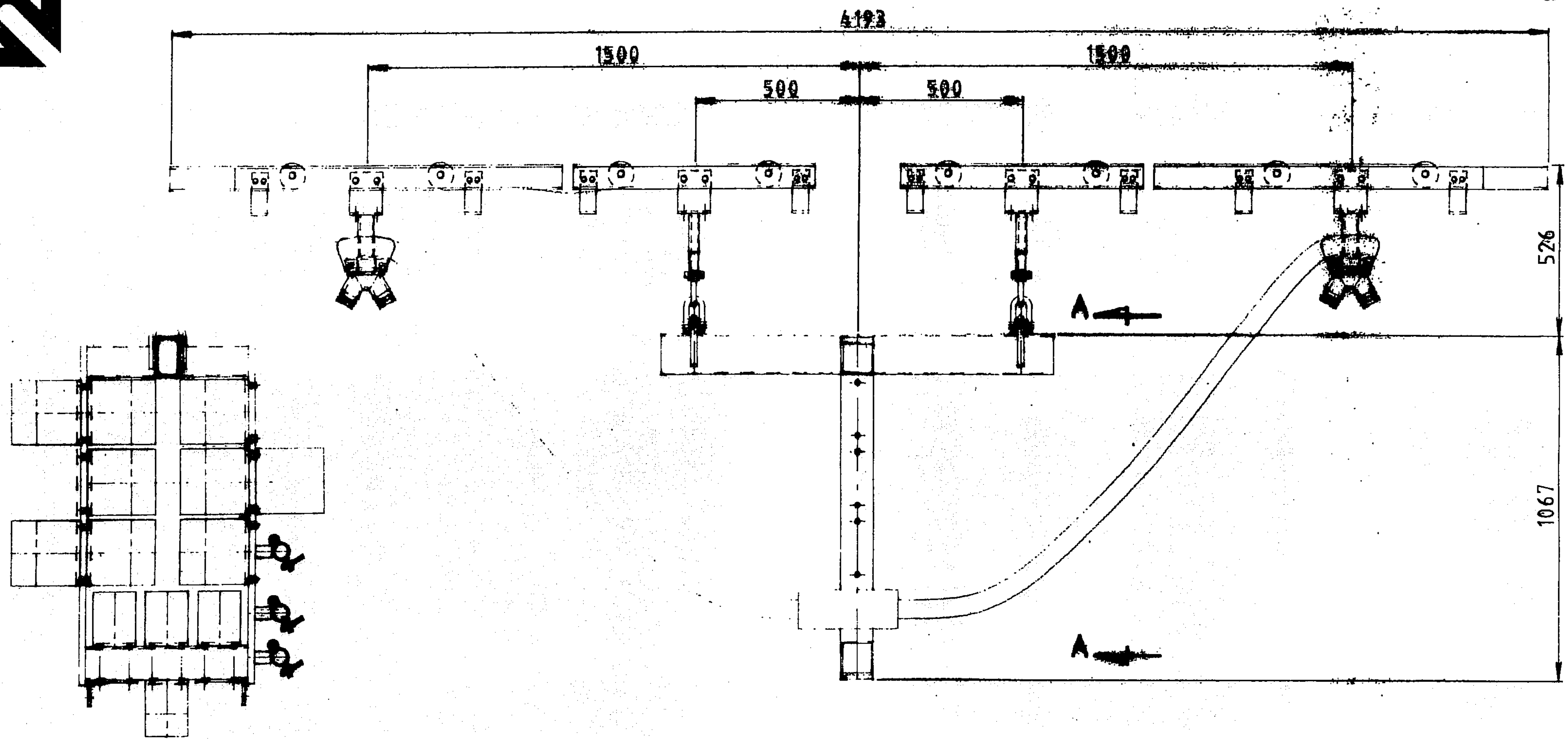
REV.



VIEW A-A.

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Page No.

OAKY CREEK
MANIFOLD TROLLEY
ARRANGEMENT & DETAILS PROPOSAL

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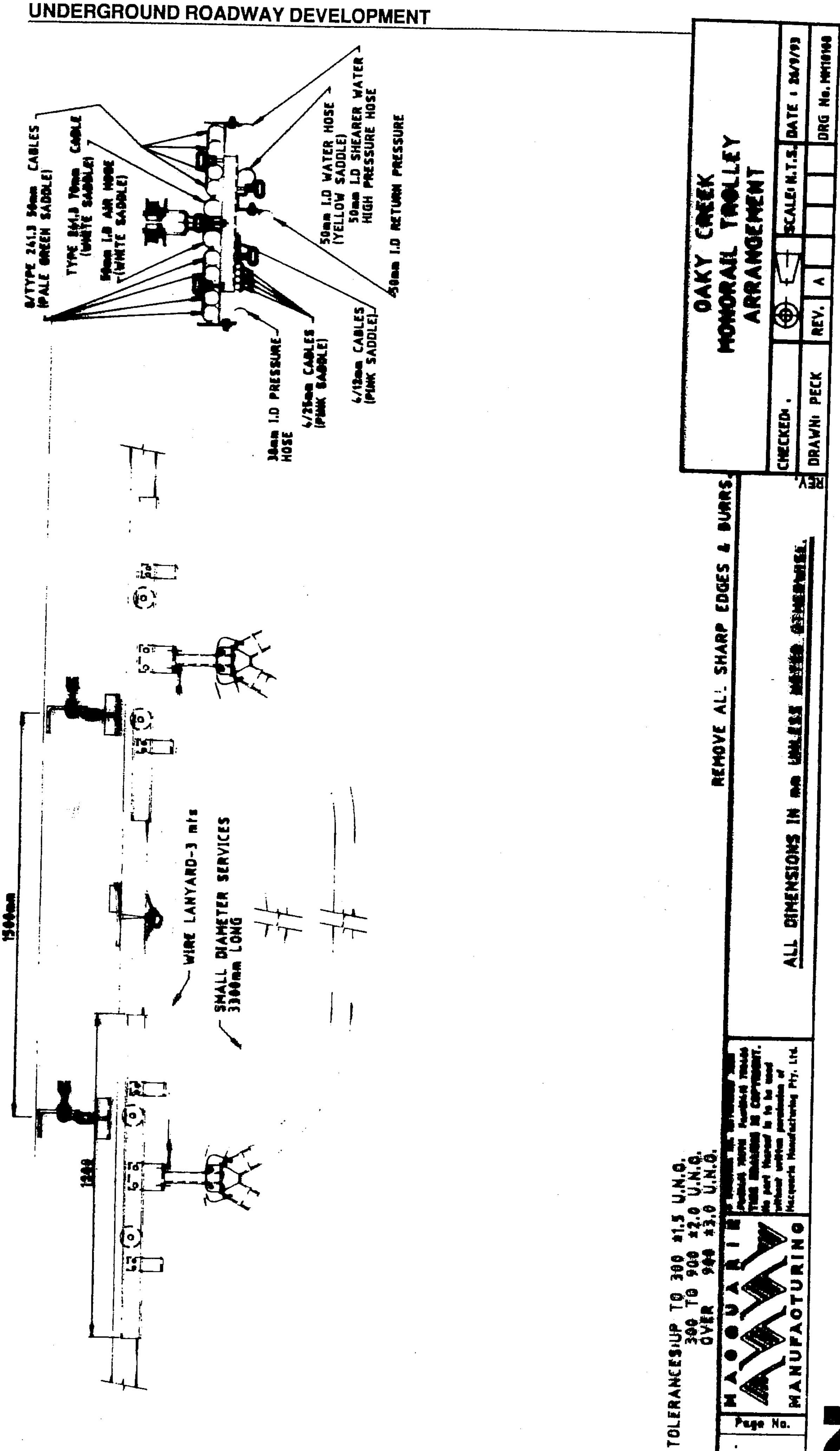
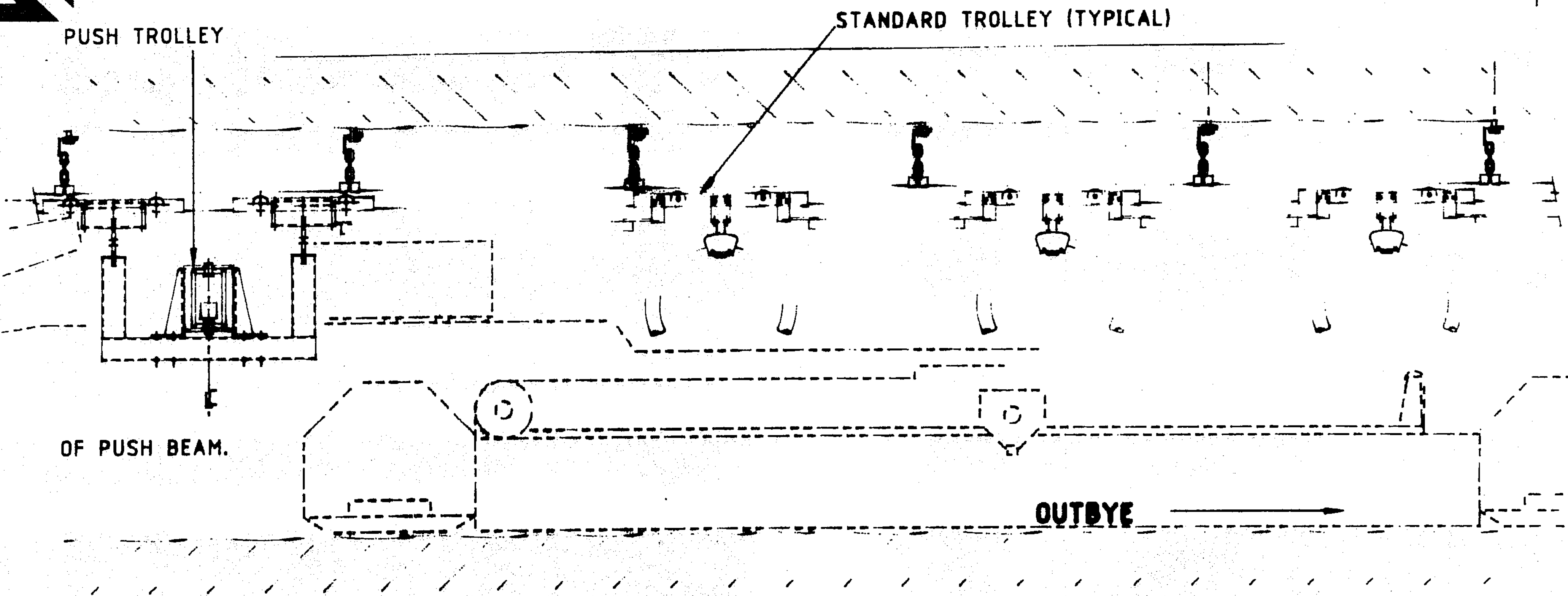

8/TYPE 241.3 50mm CABLES (PALE GREEN SADDLE)
TYPE 241.3 70mm CABLE (WHITE SADDLE)
50mm I.D AIR HOSE (WHITE SADDLE)
50mm I.D WATER HOSE (YELLOW SADDLE)
50mm I.D SHEARER WATER HIGH PRESSURE HOSE
50mm I.D RETURN PRESSURE
38mm I.D PRESSURE HOSE
4/25mm CABLES (PINK SADDLE)
4/12mm CABLES (PINK SADDLE)
1500mm
1300
WIRE LANYARD-3 m/s
SMALL DIAMETER SERVICES 3300mm LONG
OAKY CREEK MONORAIL TROLLEY ARRANGEMENT
CHECKED:
DRAWN: PECK
REV. A
SCALE: N.T.S.
DATE: 26/9/93
REMOVE ALL SHARP EDGES & BURRS.
ALL DIMENSIONS IN mm UNLESS
TOLERANCES: UP TO 300 ±1.5 U.N.O.
300 TO 900 ±2.0 U.N.O.
OVER 900 ±3.0 U.N.O.
MACQUARIE MANUFACTURING
Page No.



OAKY CREEK
LONGWALL MONORAIL SYSTEM
PUSH TROLLEY ARRANGEMENT

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REV.

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Page No.

MARK MATHESON & GARY EVANS

Capricorn Coal Management Pty Ltd

TOWARDS CONTINUOUS ROADWAY DEVELOPMENT UTILISING A JOY 1SS SUMP SHEARER MINER AND JOY 2-FCT ROOF MOUNTED CONTINUOUS COAL HAULAGE

OVERVIEW

Australian Longwalls have demonstrated the potential for annual production significantly greater than the mine operator's original forecasts. Improvements are being achieved through the application of technology, higher productivity equipment, greater consistency of production, roster variations affecting equipment utilisation and improved maintenance reliability.

In 1993 two underground mines with older technology longwalls achieved ROM production of 3MT. To sustain production at this level or greater, typical development requirements would be between 12 000 and 15 000 m/year.

For the five week month of July, Capcoal's Southern Colliery development performance was 1355m. This was close to budget and on target for approximately 14 000m/year. For the corresponding period, the Longwall achieved above budget performance of 410 000t ROM. This is consistent with an annual longwall production rate of at least 4MT, compared to the annual budget of 2.7MT.

A 30% increase in current development performance would be required to sustain annual longwall extraction of at least 4MT. A target of 5MTPA would require a 60% increase. Other examples of consistently high production can be identified for various longwall blocks over periods of one month or longer.

A number of options exist for improving total development. Progressive improvement of the existing methods of development are capable of delivering small annual increases in productivity. Increased utilisation may be available to mines not utilising seven day production options. One immediate option to provide the capacity is through the introduction of additional continuous miner panels. There are practical constraints for all mines, due to coal clearance limitations, ventilation capacity, mine electrical supply or load capacity, infrastructure requirements, priorities for capital expenditure, etc.

Capcoal's response has been to pursue a number of alternatives to improve development productivity. Central Colliery has demonstrated gradual improvement, achieving average gate road development rates of 18 to 22m/ unit shift worked. The potential of continuous coal haulage has been evaluated by experience at Southern Colliery, with a Joy FCT and Klockner Becorit Mobile Conveyor. Both underground mines have worked several distinctly different 5 and 7 day continuous rosters over a six year period.

PROBLEMS IDENTIFIED

The potential of existing development systems and equipment to achieve increased rates is limited. This is due to the discontinuous operation resulting from disruptions to coal haulage, bolting requirements, ventilation and other support activities.

The confined environment, moving equipment and manual handling requirements result in high safety costs. The potential for improved safety of existing systems is also limited by the type of equipment and the way it is used.

At present there are no operational development systems which have been able to eliminate all the causes of interruptions to continuous cutting.

A number of development panels currently operate with simultaneous cutting and bolting miners. The performances of the miners have not demonstrated sustained, consistent or reliable high development rates in average conditions.

One of the principal alternatives to existing miners, the Joy 1SS Sump Shearer has only been trialled on a limited basis. The trial excluded some operational features, including options for onboard materials handling and hydraulic roof bolting equipment. The Sump Shearer requires evaluation in a full operational environment with a system capable of providing continuous operation.

New equipment has often been introduced after relatively short operational trials of the prototype. This may not allow adequate evaluation of the performance or assessment of reliability. Inadequate information on equipment costs and performance has resulted in conservative approaches to equipment replacement. Future decisions are required to be based on the 'Whole of Life Cycle' costs.

ALTERNATIVES CONSIDERED

Other approaches to improving panel productivity have attempted to reduce non productive development time. The approaches have included:-

- reducing coal transport delays (FCT, surge cars, loaders),
- simultaneous bolting and cutting miners (ABM20, DOSCO In-Seam Miner, Joy 1SS),
- changes to methods of operation (Cut and Flit, e.g. North Cliff Colliery 1992), and
- changes to ventilation systems (Oaky Ck proposed suspended ventilation system, onboard ventilation fans e.g. United Colliery)

IMPROVEMENT ISSUES & APPROACH

Planning for Southern's future production has included consideration of issues common to other underground mines. Significant development issues include:

- frictional ignition hazards,
- constraints on capital expenditure,
- restrictions on ventilation capacity,
- limited coal clearance and trunk conveyor peak capacity, and
- previous experience with a Joy 2-FCT continuous haulage

While the FCT proved itself to be a successful continuous haulage system, it did not consistently provide the improvements in gate road advance rates originally envisaged. A review of the system of development in the FCT panel (May 1993) resulted in an initial proposal to optimise the efficiency of the FCT as part of the gate road development system. The proposal included five base criteria to:

- Mechanise materials and monorail handling,
- Utilise the FCT to support the ventilation system,
- Purchase a new continuous miner with improved production capability and ergonomics,
- Eliminate disassembly and reassembly of the FCT for panel relocation, and
- Decrease the time and resources necessary for panel extensions.

A review of the May 1993 proposal highlighted the need to adopt an approach which considered the development process as a whole, i.e. system of working, equipment, safety requirements, etc. The review led to a preliminary proposal to ACARP for partial funding of the proposed development system.

A detailed submission was provided to ACARP in Oct 1993. In preparing the submission Capcoal employees were involved through discussions, meetings and presentations. A three day HAZOP study was completed prior to the submission. The HAZOP included a review of ergonomic and safety requirements for panel operation and equipment functions.

PROJECT OBJECTIVES

Summary

The Southern Colliery project involves the extensive operational trial and evaluation of an integrated continuous roadway development system. This will be achieved through the introduction of new technology mining equipment including a Joy 1SS Sump Shearer. The equipment and system of development has the potential to significantly improve development performance.

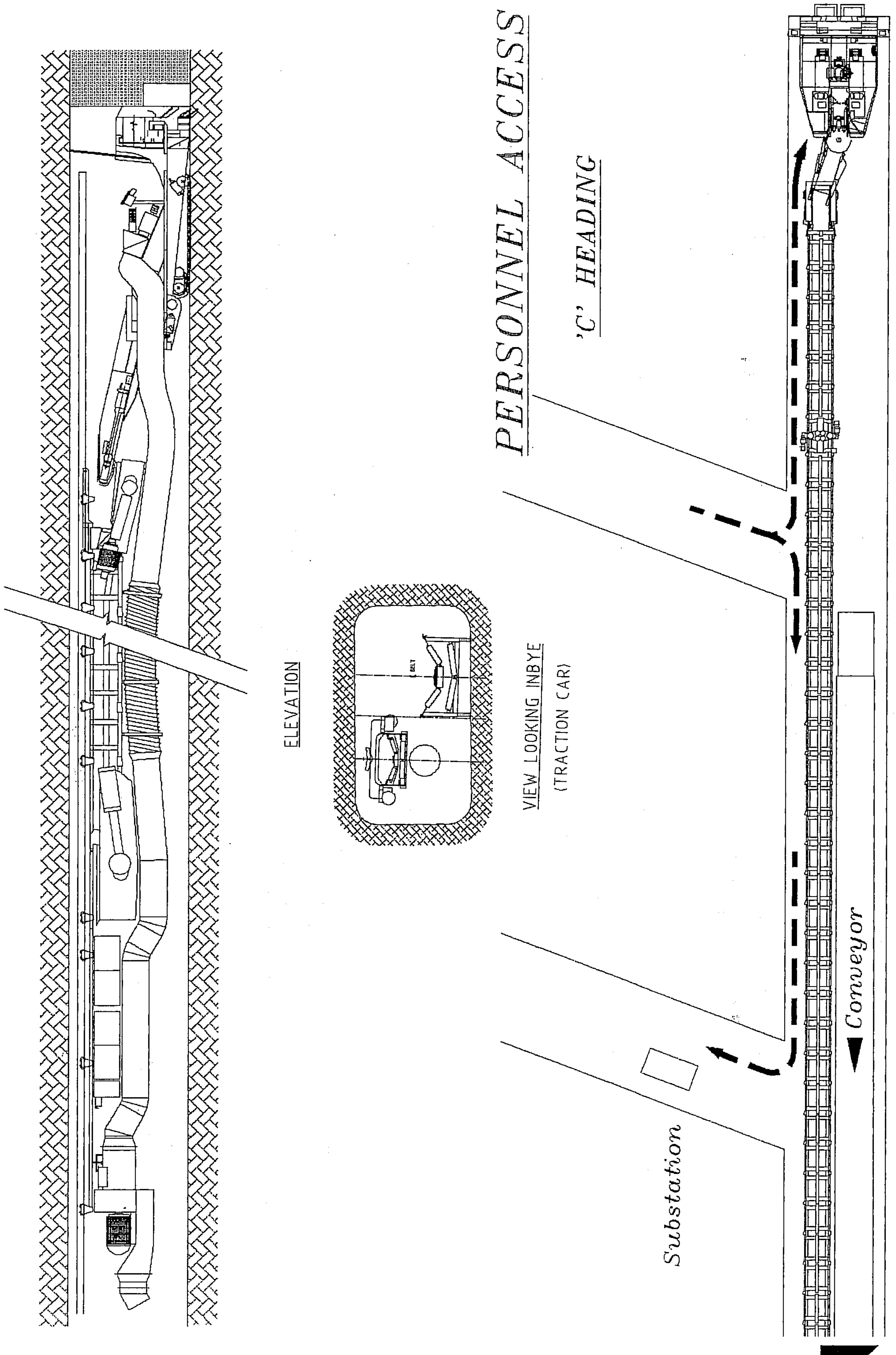
The project has an estimated capital cost exceeding $4.0M, and associated operational expenditure estimated to be up to $9.0M.

It is planned that the system will be available for demonstration on the surface in February 1995. The first 1,000 metres of development is scheduled for completion by June 1995. On the basis of satisfactory progress towards the stated targets, evaluation of the system will continue until December 1995.

Outline Of Project

The system integrates a Joy 1SS Sump Shearer, FCT continuous coal haulage, ventilation equipment, services and materials supply. The ventilation equipment and services are integrated with the FCT and monorail system so that they advance with the FCT.

Substantial labour efficiencies and employee safety improvements are anticipated. These are possible through the elimination of manual


PERSONNEL ACCESS
'C' HEADING
ELEVATION
VIEW LOOKING INBYE
(TRACTION CAR)
Conveyor
Substation

materials handling requirements, reduction of the causes of delays to production, and more effective gas or dust management as a result of:-

- elimination of manual erection of ventilation,
- elimination of manual erection of FCT monorails,
- elimination of waiting time for coal transport,
- reduction in lost cutting time to allow roof support,
- reduction of lost time for extension of feeder cables and services, and
- provision of direct effective ventilation at the working face.

The Joy Sump Shearer trials at Angus Place Colliery indicated substantial improvements in operational safety are possible using this machine in an engineered work environment.

The project involves a commitment from Joy Manufacturing to supply a 1SS Sump Shearer. Significant engineering support has been provided by Joy. This has been in assessing and resolving the design requirements, necessary to fully integrate the FCT, Sump Shearer and ventilation equipment.

The performance targets for the system include a consistent cutting rate of at least 8m per cutting hour. The system performance is targeted at a production shift average of at least 25m. This level of performance is consistent with a maingate development rate of at least one (100m) pillar per week, inclusive of panel extension and maintenance.

The system will be trialled and evaluated on the 702 LW main gate development at Southern Colliery. This represents a substantial commitment as the panel will become the priority development panel on completion of commissioning.

Standard engineering practice and proven equipment design is being applied to the development of the system. The intent of the initial system evaluations and detailed engineering has been to minimise the level of risk.

Management of the risks associated with the project is a key issue. The principal risks are with the reliability and production performance of the Joy 1SS Sump Shearer. The capital expenditure and equipment costs are approximately 250% higher than a conventional system. The system provides significant benefits to mine and employee safety. The ultimate acceptance and viability of the system depends on its cost effectiveness. The system must deliver a proportional productivity benefit to justify the initial expenditure in new equipment and training.

Summary of Benefits

Short Term

The successful introduction of the development system will assist Southern Colliery achieve improved safety, production and cost performance. Occupational Health and Safety benefits will be delivered through a reduction in manual handling. The potential production benefits are a result of improved face utilisation of the Sump Shearer and FCT combination. Cost performance benefits are a consequence of the reduction in heavy physical demands on employees, improved equipment utilisation and more continuous production cycle.

Long Term

Benefits such as safer access to underground coal deposits will follow from successful implementation of the new development system. The Bowen Basin measures include areas of higher risk conditions, e.g. poor roof conditions, inseam methane, or possible outburst areas. The design of the Sump Shearer is suited to the management of these risks. The economical development of mines in the Bowen Basin requires the continued improvement of safer and more productive drivage systems.

Without improvement in both safety and productivity, the viability of existing or future mining operations will be restricted. As an example the future of Capcoal's Central Colliery is in part dependent on the ability to mine the deeper 300's series of longwall blocks. Development of the German Creek seam at greater depth of cover requires consideration of limitations due to seam gas and possible outburst conditions.

Mark Matheson

Gary Evans

Bill Kathage

Mining Consultant

SIMULTANEOUS DEVELOPMENT SYSTEM — "S.D.S."

The S.D.S. provides 'time gains' that relate directly to 'metre gains' that relate to 'cash flow gains' that relate to higher profitability and finally and most importantly, 'contracted consumer commitment'" is positive.

The S.D.S. also provides for positive operating savings that are directly attributed to the 'Simultaneous Development System' and new methodology.

(a) The development of the S.D.S. as presented today, has been the result of many years of intermittent thoughts, planning initiatives and hundreds of sketch drawings which commenced in 1984.

(b) Westfalen Collieries, a fully owned subsidiary of Bundaberg Sugar, initiated talks with Eickhoff and Atlas Copco in relation to a full face mining machine that could mine a face 7m in width and 3.2m high. The criteria we required was a full face miner that would allow roof bolting to proceed as the face was being cut.

The ESA-60-L was the machine we selected to achieve this task and our only essential requirements was a "Sumpin Shearer" should be developed for the ESA-60-L operation. Shield supports were designed to complete the ESA-60-L face mining machine. Two visits to Eickhoff in Germany were made.

(c) In 1985 - 1986, the installation of a Flexible Conveyor Train - F.C.T. was discussed with Joy Manufacturing in Australia and the United States a decision was made to install an F.C.T. in Boxflat No. 9 Colliery. The F.C.T. was very efficient with high availability and increased R.O.M. production from 300 tonnes per unit-shift to 900 tonnes per unit-shift and in one 7hr shift produced 1972t.

The mono rail system to support the F.C.T. was also a totally new method and a tremendous amount of work was carried out in relation to roof bolting, no roof to floor support, and suspending the F.C.T. from a roof bolt that were part of the primary roof support system.

The mono rail system and erection of mono rails was not a problem, and as far as I am concerned the mono rail system allows many other operational functions and is a system that should be embraced by the underground mining industry in the development of panel headings as the efficient use of a total mono rail system is unlimited in its practical operational functions.

(d) In 1986 an application was prepared by our Company for a N.E.R.D.D.C. grant to develop a "Single Entry" heading 1,000 metres in length and 7 metres in width. The equipment to be used was an ESA-60-L with shield supports and drill rigs and a flexible conveyor train. Westfalen, Eickhoff and Atlas Copco were the joint applicants and I "headed up" the proposals.

We were informed in 1987 that we were successful with our N.E.R.D.D.C. grant and two weeks later our contract to supply coal to Swanbank Power Station was terminated - 6 years ahead of the contracted period.

The above is a basic background of my involvement with the ESA-60-L – the flexible conveyor train and rapid single entry development and the mono rail system. My next association with the mining industry was in 1992 when I carried out Recognition of Prior Learning (R.P.L.) assessments for Gordonstone Coal Company Pty Ltd.

When talking to the Miners of Gordonstone, who were being assessed, it became very evident to me that the existing system of heading development had to be reviewed. In making that statement I was not critical of the miners or of management, as Gordonstone was achieving greater panel development advance than most other collieries.

The roof conditions at Gordonstone are generally the worst in Australia and to achieve this advance, was indicative to me, that the miners and management were dedicated people. The introduction of the Voest-Alpine A.B.M.-20 was a major planning decision that allowed continuous cutting while primary roof support was being carried out, and this prepared the way for a development review of panel development.

As I had known Malcolm Roberts since he was a cadet, gaining experience in our mine, I stated at that time if any cadet manager was to attain a senior position in the underground mining industry it would be Malcolm, and I was right.

A meeting was held and the introduction of a F.C.T. into Gordonstone was discussed. Malcolm had set views on heading development for Gordonstone and as his views and mine were identical, a philosophy was prepared:-

- Keep It Simple
- A mining machine with a cutting drum with roof bolters attached – the roof bolters the main feature of the Face Mining Machine (F.M.M.)
- The F.M.M. – Roof Bolter to provide steady cutting – Low Capacity – high reliability – high rates of panel development.
- Total cycle – cutting – bolting – loading – conveying – floor surface – roof stability – cable bolts.
- Ventilation – Queensland Mines Department – rib bolting. Safe environment.
- Face cycle development.
- Longwall block cycle mainly longwall development.

In May, 1993 Malcolm asked me to prepare proposals along the guideline as set out in the philosophy. Proposals "A" To "L" were completed in November 1993 and submitted to Malcolm before he left Gordonstone.

I have named the final proposal the "Simultaneous Development System" - S.D.S. and as the name implies, the system allows for the development of the main and tail gate headings simultaneously.

In my opinion the development of the panel headings needs to be recognised as the most important link in the production chain in the overall context of the longwall system.

The existing continuous mining systems and associated equipment should not be considered when planning the development of main and tail gate headings with heading widths of 5 to 5.2m.

- The face mining machine should be a full face machine that will allow roof bolting to be carried out while the F.M.M. is cutting and the dimensions of the F.M. should be minimal and should provide a drilling platform for the required number of drill rigs.
- The cutting and loading capacity of the face mining machine should be 4 to 5t per minute and profile curved cut ribs on a right angle cut profile can be introduced to suit varying roof conditions.

The F.M.M. will be fitted with the number of drill rigs required for the efficient control of the roof and heading strata stability.

The system needs to be simple with adequate ventilation and must be designed so that management can introduce a positive sequenced operation.

The system will provide for the maximum utilisation of the underground workforce, especially in the main and tail gate heading.

The "Simultaneous Development System" was developed and designed to meet the above "criteria" and the S.D.S. will provide the operation with varying functions that will improve the overall safety environment which is the most important aspect of any underground mine operation and allow the industry's achievable goals to be met.

The main functions and equipment on which the overall S.D.S. has been designed are indicated below.

"A" — "B" — "C" — "D" — "E"

SIMULTANEOUS DEVELOPMENT SYSTEM

"E"

Pillar dimensions can be variable depending on mine planning e.g. 200m 300m 400m

The increase in the length of the pillar dimensions decreases the number of cut throughs by half. The cut throughs will be developed from the tail gate heading and 'break into' the main gate heading. The system provides greater stability in the M.G. heading especially in longwall retreat.

"D"

Comprehensive mono rail system M.G. - T.G. headings "multi use" system. System highly efficient.

The mono rail system is a very important aspect of the S.D.S. and will be used to:

1. Suspend the F.C.T.'s in the main and tail gate headings
2. Vent tubes will be suspended from mono rails
3. A pantechnicon mono rail will be installed in the M.G. heading
4. A centre mono rail in T.G. heading
5. A water barrier will be suspended from the mono rails in main gate heading

"C"

One only transfer conveyor to convey coal from the tail gate conveyor to main gate conveyor.

A transfer conveyor will transfer coal from the tail gate conveyor to the main gate conveyor and will be suspended from monorails. The capacity of the transfer conveyor will be 4 to 5 tonnes per minute.

"B"

Tail gate heading one only flexible conveyor train Bridge conveyor

The installation of the F.C.T. will allow continuous development - installation of cable bolts as primary strata support - cable bolts can be erected 8 metres from the face while the F.M.M. is cutting.

"A"

Main gate heading one only flexible conveyor train Bridge conveyor.

No back work in relation to strata control will be required. Clear working area of 5.2m x 1.5m. Clean floor area - efficient ventilation by eliminating the shuttle car - efficient belt extensions - D.C.B.'s suspended from mono rail. Reticulation cable - water air hoses suspended on mono rail.

SIMULTANEOUS DEVELOPMENT SYSTEM

All calculations and assessments were based on the following:

a. Length of main gate heading 2800m
b. Length of tail gate heading 2800m
c. Distance to mine in cut through 38m
d. Distance to mine establishment road 245m
e. Pillar dimensions — 200m x 40m
f. Number of cut throughs 14
g. Heading width 5.2m
h. Heading height 3m
i. Longwall block — 300m x 250m
j. Total distance to mine to complete panel development
   Pillar dimensions — 200m x 40m 6380m
   Distance to be mined with pillar dimensions of 100m x 40m (Existing System) . . 6910m
k. S.D.S. gain – 532m less to develop – 14 fewer cut throughs – 14 fewer stoppings – 532m less roadway to stone dust – roof bolts, cable bolts, cost and installation savings. ***Significant cost savings***. Significant gains in metre advance - 532m less to develop per 2800m long panel development.

BOLTING PATTERN

W.straps + mesh 4650mm in length
Spacing - 700mm
6 - 2.1m long roof bolts per strap in headings and between cut throughs
8 - 2.1m long roof bolts per strap
12m in cut through area
10m cable bolts as required by management

The "Simultaneous Development System" allows the following tasks and functions to be carried out efficiently and safely.

a. All break offs and cut throughs will be developed from the tail gate heading, reducing potential "hotspots" (areas of instability) in the main gate heading and a 50% decrease in the number of cut throughs also creates greater stability in the main gate heading.

b. All cable bolting - roof bolts and rib bolts will be installed as the main and tail faces advances and "No Back Work" will be required during the development of the panel headings and during the longwall extraction operation. The immediate roof and the lower strata must be controlled as soon as possible and no "Sag" should be allowed to occur and the S.D.S. allows ultimate control of the stability of the headings.

c. The F.C.T. installation suspended from the mono rail also adds efficiency to the operation as the D.C.B. and F.C.T. control box will be suspended from the mono rail - the reticulation cable will be suspended in a coiled configuration from the mono rail - the water and air hoses will be suspended from the mono rail in a coiled configuration.

d. The F.M.M. in the main gate heading will develop the heading in a straight line for a distance of 2800 metres. The F.M.M. in the tail gate heading will mine in a straight line for 2800m but every two hundred metres the F.M.M. will break off and develop the cut through.

The roof bolts supporting the mono rails will be erected as part of the primary roof bolt support system.

e. All supplies in the main gate heading will be via the pantechnicon mono rail.

All supplies in the tail gate heading will be via the centre mono rail.

f. A conveyor belt system will be installed in the tail gate heading and a new designed conveyor structure will be introduced with the S.D.S.

g. The main gate conveyor system will be efficiently installed during every 200 metre extension and will remain installed correctly as no shuttle cars will affect the conveyor installation. The installation of conveyor structure on a clean floor will ensure all stands are installed correctly. A new concept of a conveyor installation can be made available.

h. The ventilation of the main – tail gate headings will be efficient as the "equivalent orifice" of the headings will not be variable as no shuttle cars or supply vehicle will be operating in the heading.

I. The mono rail system will allow the transfer of all equipment used in the heading developments (not F.M.M. or transformers) from a completed heading to a new panel development.

The pantechnicon mono rail will transfer all the pantechnicon equipment and cable loops from the completed longwall extraction to the new longwall to be extracted and the input power cable are outgoing cables will have to be reconnected ready for operation.

j. A standard ventilation system using fibre glass ventilation tube with a design variation in the face area will be suspended from mono rail.

A new designed system of extending and retracting ventilation tubes is available that will ensure an uninterrupted ventilation circuit over a distance of 100m.

While the main gate conveyor loop take up is being recharged with 200m of conveyor belt – 11 ventilation modules will be installed in the extending and retracting ventilation tube installation and the ventilation circuit will be uninterrupted over a distance of 100m.

While the face mining machine is operating, the ventilation circuit will be uninterrupted over a distance of 200m.

The above are the main functions and equipment requirements to briefly explain the S.D.S. system and all details of the total system is available, supported by detailed plans and sketches.

The figures used for assessment resulted in the final analysis in relation to time scheduling.

6380m of panel development would be completed in 14 weeks and a 40% contingency has been included in this time schedule.

Three weeks has been allocated to transferring equipment from a completed panel development to a new panel development.

Total time - 17 Weeks

Detailed information is also available for these assessments.

After hearing my paper I hope that I have, perhaps, introduced some new ideas – design concepts and new methodology that will generate debate, criticism and general discussion of the Simultaneous Development System. All equipment introduced into the S.D.S. is generally available and no major design changes are necessary, only additions to existing equipment and some small modifications.

A new type of conveyor belt installation and extending and retracting ventilation tubes is available as components of the 'ultimate system' but is not included in the ***Initial S.D.S.***

I have no reservations whatsoever that the ACARP workshop will produce '***Great Ideas***' and the underground mining industry, in relation to panel development, will be promoted to the front of the production chain.

I am very confident of my opening comments and I hope I have created some interest that will generate future dialogue. Being persistent will remind you again.

COMMENTS

The S.D.S. provides 'time gains' that relate directly to 'metre gains' that relate to 'cash flow gains' that relate to higher profitability and finally and most importantly, 'contracted consumer commitment' is positive.

The S.D.S. also provides for positive operating savings that are directly attributed to the "Simultaneous Development System" and new methodology.

CONCLUSION

SIMULTANEOUS DEVELOPMENT PANEL

The S.D.S. will allow designed - planned management to be introduced into the 'crucial panel development' that has been seriously lacking in the past.

Management must develop strategies and initiatives in the panel development if the highly efficient longwall system is to achieve its productivity potential.

Management, manufacturers and the Queensland Mines Department must restructure, re think existing systems of panel development and introduce new methods that are simple and systematic to operate and are management efficient. Mining equipment and drill rigs must be designed to be very efficient.

Mining equipment should be simple in design, physically small in machine size with a very low non productive time level.

Drill rigs should be fitted to the F.M.M. that will be compatible to the primary roof support pattern.

Ventilation of the main and tail gate headings should be revised and improved to allow continuous ventilation of the face area.

A new coal transport system from the F.M.M. to the main gate conveyor must be introduced.

As most development headings are 5m in width, a mono rail system should be introduced into the new scheme of panel development.

If there is no change in the overall concept of developing main and tail gate headings in the longwall panel development, then the underground mining industry will not survive in the export world, as open cuts will not always be there as a 'back up'.

GARY ZAMEL & LLOYD ZENARI

Pacific Tunnelling Pty Limited

TBM USE IN ROADWAY DEVELOPMENT

Technological developments in modern Tunnel Boring Machines (TBM's) and associated back-up equipment are offering significant alternatives for mining operators to consider, in the selection of equipment for coal mine development drifts, roadways and longwall gate roads.

Greater flexibility in handling variable ground conditions, muck disposal and radius of turn as well as advance rates, are just some of the areas where modern TBM's can accomplish the flexibility that mine operators require to achieve cost effective roadway development.

Compact TBM's are achieving this acceptance due to the fundamental design of a shielded machine containing electrics and hydraulics within the forward shell, ability to achieve tight radius of curvature and grades, and ability to provide highly flexible support systems to meet all possible ground conditions.

These rapid excavation techniques common in the civil tunnelling industry need to be modified in order to fulfil the requirements of roadway development. Advance rates in excess of 200m per week are common in similar ground conditions tackled within the civil tunnelling industry, and it is this feature which most needs to be exploited within roadway development in coal mines.

The continual development of TBMs has led to a range of machines which are able to be customised to meet the ground conditions, and mine plans of individual operators. Whilst all Mine Developments will have common goals, the modern customised TBM will enable certain unique criteria at each mine to be fulfilled.

Back-up and production equipment — including the use of continuous conveyor systems — will also need to be modified to suit the unique requirements of the tunnel boring system selected, including the circular profile of the driven tunnel. This total system approach will ensure that improvements in safety, efficiency and cost-effectiveness are satisfied.

INTRODUCTION

The mining industry in general needs to investigate advances in modern technology to improve productivity in the underground domain. Traditional development methods using 'continuous mining' techniques have been favoured over rapid excavation methods developed in the civil tunnelling industry. The use of these systems has often led to delays in Longwall relocation due to incomplete panel development.

The ramifications on the cost of production coal in underground coal mines is often a direct result of the inefficient use of Longwalls and the capital costs of same.

In order to improve Mine Development techniques and machinery, inherent problems in roadway development in coal mines need to be addressed.

- Development rates,
- Cost-effectiveness,
- Ground support at the face — for both personnel safety and Equipment Safety,
- Ground support systems -- both short term and long term,
- Manoeuvrability - development should follow the coal seam,
- Turn-around times for equipment used,
- Labour requirements, and
- Ventilation requirements.

Although TBM's have had limited success in mines to date, most trials were undertaken before technological advancements in design and manufacture solved the primary concerns with TBM use in Mine Development.

The modern TBM offers the following advantages to the Mine Developer:

- Faster and more economical development rates - weekly advance rates in excess of 200m being common in TBM projects in similar strata. Weekly results of 350m now being regularly achieved,
- Mining systems and ground support systems to suit all possible ground conditions,
- Reduced Labour requirements,
- Faster turn-around times for TBM assembly and disassembly,

- Tight curvature radii — both vertically and horizontally,
- Better ventilation characteristics in the completed roadway,
- Improved long term ground stability in a circular profile,
- Improved safety for workers through continuous ground support during development — initially within the TBM, and immediately after, through traditional support systems,
- Better working environment for tunnelling personnel —modern TBM's are highly mechanised, and hence less labour intensive, and
- Ability to apply a total systems approach with a high level of automation.

However, additional refinement of rapid excavation equipment and techniques can be undertaken to further improve the use of this technology in underground mining. This development needs to be supported by both equipment manufacturers and the mining industry to bring Mine Development 'up to speed'.

The outstanding track record of TBM's in the civil tunnelling industry justifies the further customisation of TBM's for coal mine development, rather than ignoring this fundamental, proven and reliable approach. However, mining methods will need to be modified to adapt to the operational aspects when driving with a TBM.

EARLY TBM USE IN MINE DEVELOPMENT

The use of Tunnel Boring Machines in the development of the Donkin-Morien coal mine in Canada, and the Stillwater palladium mine in the United States has proved the viability of tunnel boring systems in the mining industry. Magma Copper are currently using a TBM to develop their ore-body in Arizona, however details of the results of this project have yet to be released.

The following benefits were identified from the two earlier projects :-

DONKIN–MORIEN PROJECT :

- The TBM was three to four times faster than Drill and Blast — which was used during the commencement of the adjacent drift,
- TBM required approximately one third of the manpower over drill and blast,
- There were minimal injuries associated with the TBM drive,
- Advance rates for the TBM were in the range 1.9m/hr, whilst Drill and Blast was 0.3m/hr,
- At October 1984 prices, Tunnelling costs for the Drill and Blast operation were CAN $12 000/m, whilst TBM costs were CAN $4800/m. The associated reduction in tunnelling costs of CAN $7200/m meant that the capital cost of the TBM was repaid after approximately 1000m of development,
- The machine driven tunnel is also expected to cost much less in long term maintenance, and provide significantly better ventilation characteristics, due to the circular profile, and smooth tunnel walls and the absence of overbreak and fracturing associated with blasting,
- Ability of the TBM to mine through mixed ground with compressive strengths varying from 10MPa to 100MPa,
- TBM capable of boring at gradients of -22% to +3%,
- TBM able to drill investigative bore holes through the cutterhead, during the regular maintenance cycle, and
- In order to create a flat working invert in the circular profile of the tunnel, the required amount of material was allowed to spill off the end of the TBM conveyor, and later compacted in to place. In the drift, prefabricated steel grid floor was used.

STILLWATER PROJECT :

- A reduction in development cost of about one-third over conventional methods,
- A reduction in Rock bolting of 80-90%,
- Significantly reduced re-bolting of tunnel walls in the long term, and
- A significant reduction in the development time of new areas for mining. As an example, the 5700W level was completed in 10 months, compared with over two years for conventional methods.

FURTHER DEVELOPMENT OF TBM TECHNOLOGY

Current development rates in underground mining identify the need for new development techniques, coupled with improvements in TBM design to satisfy the requirements of Mine Development.

Advances in Longwall technology, and their implementation in mines worldwide, have emphasised the need for improvements in Development technology to keep pace with Longwall production and the overall reduction in 'panel time'.

Certain aims of mine development, such as advance rates, safety and cost-effectiveness are common to all development programs. However, each mine will have certain criteria on their 'shopping list' for a TBM, which will need to be fulfilled in order for the use of the TBM to be successful.

The inherent design characteristics of shielded TBMs offer the flexibility and safety required in this application.

UNDERGROUND COAL MINE DEVELOPMENT

Improvements in modern Longwalls have allowed for larger panels to be mined, which require significantly longer gate roads. A common problem associated with most Longwall operations is the inability of development to keep abreast of Longwall production rates. This results in lost production and inefficient use of capital equipment, as the Longwall often has to wait for the completion of a new panel before being re-located.

The ongoing improvements in Continuous Miners and Roadheaders has only partially solved the problem because the actual cutting time of most continuous miners is only approximately 25-35% due to the following delays:

- Establishment of ground support,
- Delays due to removal of coal from the heading, typically by shuttle car,
- Delays incurred during the supply of materials to the face - such as bolts and straps, and
- Delays due to conveyor belt and infrastructure extensions.

Future Longwall Panels in Australian Mines will require approximately 7.5km of development - comprising 6.5km for Longwall Development and 1km of main headings.

This equates to about 30 weeks to drive the tunnels, and approximately 10 weeks of set-up time. Realistically however, development times per panel are closer to one year.

Such significant amounts of tunnelling call for rapid excavation procedures and equipment.

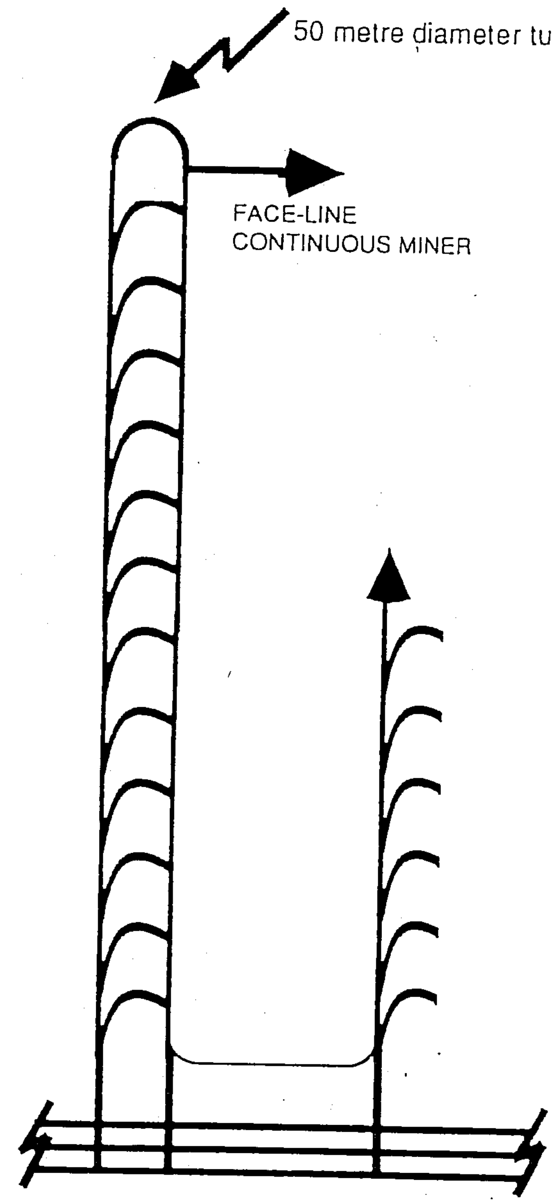


Figure 1.

Ground support and mine planning requirements, standard in most underground coal mines have identified changes which will need to be made to current TBM designs to fulfil the following requirements:

- Ground support at the face is mandatory — this may be achieved through angled bolting from behind the TBM Cutterhead,
- Breakaway capability of the TBM — for cut-through development, and
- Assembly and Disassembly of the TBM must be fast and easily achieved in a blind heading, with underground equipment — such as shield movers.

Using current development ideology, the TBM must be capable of gate road development (Figure 1) (personal communication — Professor Jim Galvin).

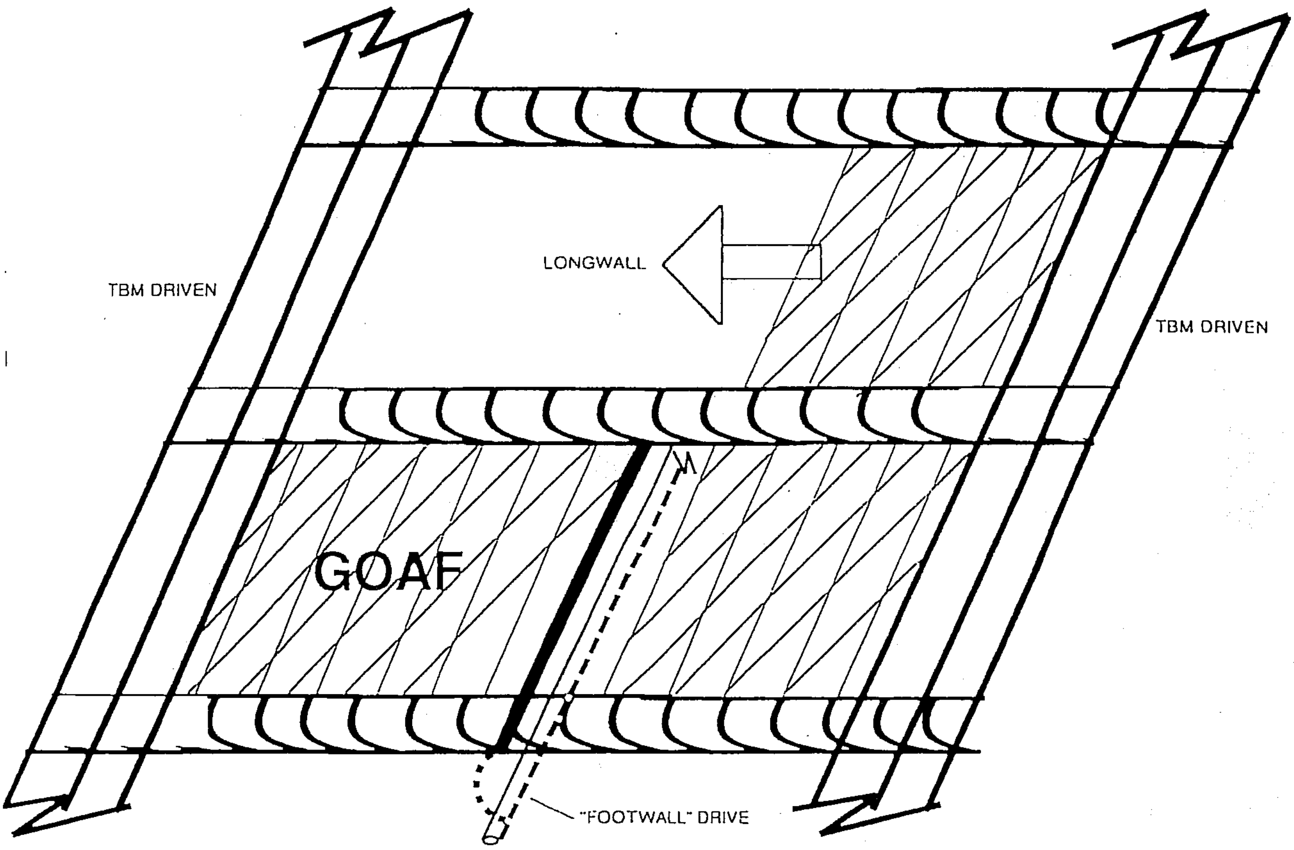


Figure 2.

During tail gate drivage, the TBM may need to be able to drive the cut-throughs, then reverse and continue with gate road development. At the end of the future Longwall panel, the TBM may be required to negotiate a 40 - 50m diameter turn and drive the headgate.

Manoeuvrability of the TBM is also essential for quick reaction to faults and rolls (horizon control) along the main development route, so as to minimise cutting of rock from the roof or floor of the tunnel. This feature may not be as essential in gate road development however, as it is unlikely that the Longwall would be able to negotiate a deep fault zone — which the TBM, being much more manoeuvrable than a Longwall would be able to negotiate.

This would require the gripper assembly of the TBM to work either vertically — during development of cut-throughs, where the grippers would act on the tunnel roof and floor — or horizontally during gate road development — where the gripper pads would act on the roadway ribs.

As previously mentioned, modern Longwall technology has enabled the introduction of larger production panels. The use of TBM's in mine development could be further expanded, and procedures commonly used in metalliferous mines — such as below seam drives (footwalls) — could be incorporated into the mining method, to further increase the size of Longwall panels and hence improve overall productivity (Figure 2) (personal communication — Professor Jim Galvin).

A separate TBM or Roadheader could be used below the seam,in the 'footwall', to divide the 4km panels. After the Longwall had advanced beyond this point, the 'footwall' tunnel could be elevated to the elevation of the Longwall to complete the return for the foul air. Service and maintenance on the Longwall would also be completed at this point prior to advancement of the Longwall in to the next panel.

This method would enable greater production rates to be achieved through the following:

- Increased use of Longwall equipment - since the need for relocation is halved,
- Faster development rates through the use of TBM's,
- More efficient use of the coal resource through minimisation of pillars which are later reduced to goaf, and
- Division of muck handling between TBM and Longwall, to compensate for the increased proportion of rock in the development material.

Another method of Longwall Development also justifies further investigation, especially in mines which may have spontaneous combustion concerns. This method involves

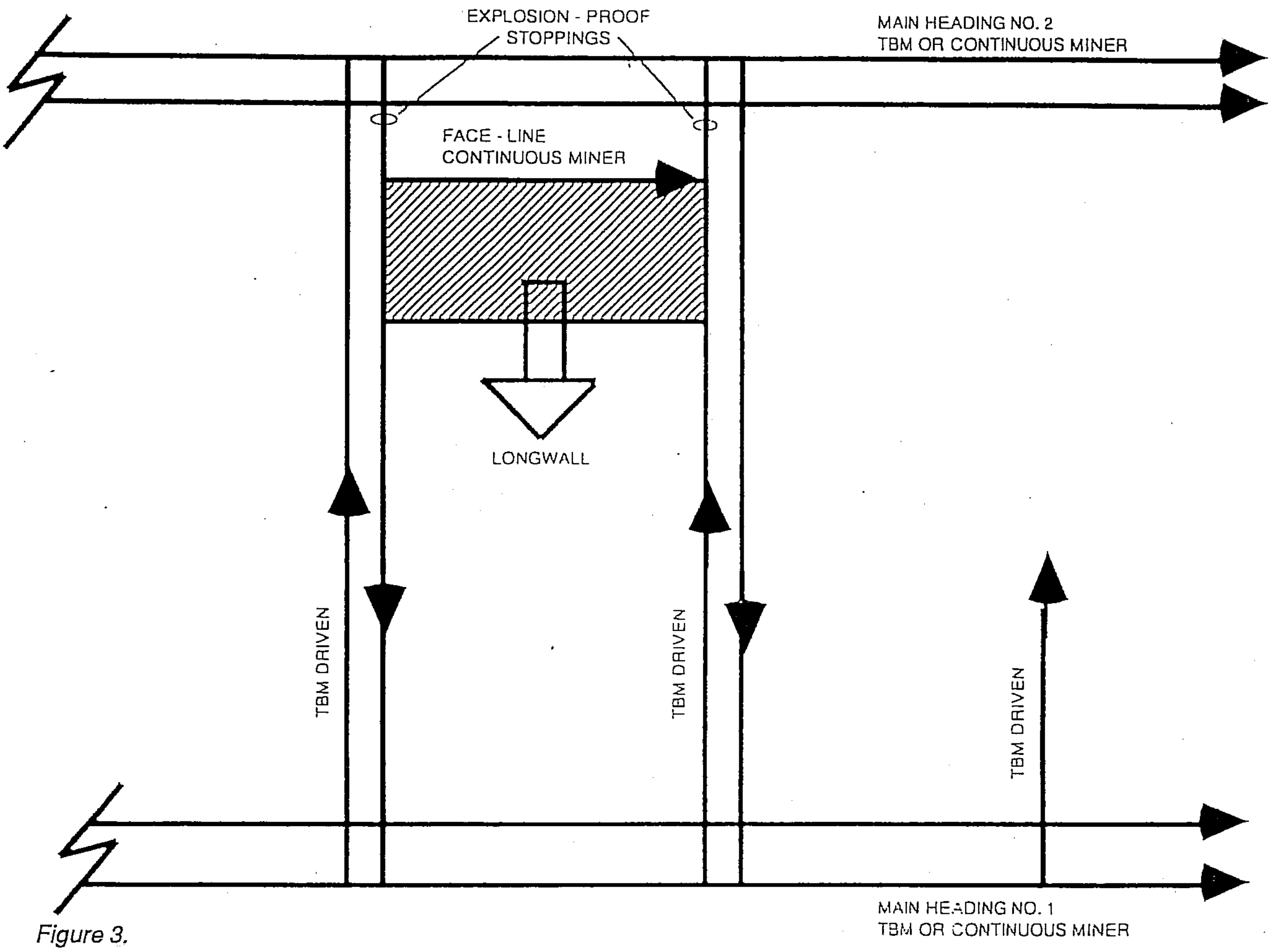


Figure 3.

single-entry headings, which due to their nature would be more effectively bored by a TBM (Figure 3).

Main Headings 1 and 2, and the associated Pillar development would be by Continuous Miner, TBM or a combination of both, depending on development rates of the gate roads.

Gate Road development would be via single-heading TBM drives with return air being channelled via ducting to the main drives, out-bye. The TBM would break-through into Heading number 2, where it can be disassembled - in the case of a shielded machine in to three major components. These may be rotated on steel plates, and the TBM re-assembled to bore the next development drive in the opposite direction.

The process would then be repeated upon break-through into main heading number 1.

The use of a TBM in coal mine development would involve the adoption of continuous conveyor technology and belt loop take-ups which would ensure continuous haulage during TBM advance. Therefore, the TBM in itself would only be 6-7m long, as no 'TBM train' would be required. This enables fast turn-around times of the TBM to be achieved, which are necessary at each hole-out.

To ensure that return-air during Longwall operation was channelled via the tail gate, and not across the Goaf, the Face Line would be driven via continuous miner, and explosion-proof stoppings inserted at the head and tail gates prior to the commencement of production. Development of the Face Line by continuous miner also ensures a flat roof, necessary for chock installation.

This method would also enable extremely fast Development rates to be achieved, as two continuous miners — one each in Headings 1 and 2 — would be able to keep pace with the TBM drive and enable ventilation requirements to be maintained.

Depending on the development rates of the gate roads, another TBM may be justified for single-heading main development drives. Cut-throughs at the gate road intersections would then be driven by continuous miner, for men and materials, and access to a second egress.

Assuming the Longwall Panel is 3000m long, and the TBM maintains an average of 300m per week, the Longwall Panel could now be developed in 6–7 months, which is a significant improvement on current rates.

CONCLUSION

Technological advancements in modern Tunnel Boring Machines and associated backup equipment offer significant advantages to all underground mine operators with regard to cost effective mine development. This has already been proven in numerous projects worldwide including the Stillwater and Donkin-Morien projects.

Further modifications to the design and capabilities of these compact TBM's need to be made however, for this method of development to appeal to the mining community at large.

In order for these further modifications and improvements to take place, the mining industry must support the equipment suppliers, to promote research and expenditure in this field.

Customised machines, and complete mining and production systems, to fulfil unique requirements of individual mines and development methods need to be designed to achieve the common aims of all Mine Development Programs - especially safety, speed of advance, and cost-effectiveness.

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MAINTENANCE — FUTURE CHALLENGES — PEOPLE, ASSETS AND TECHNOLOGY

"It is difficult to predict anything, especially about the future ..."

This observation can very easily sum up the futility of not planning for the future; it suggests that we should let happen what will happen.

Well, this may be a very comfortable strategy to adopt, but when it is applied to maintenance, chaos and disaster will surely follow.

I suggest that this philosophy of ***"what will happen, will happen"*** is the basis of many contemporary maintenance systems.

The 'fix and break' method of organising labour and resources is commonly practised in many underground coal mining operations.

The practical result is poor performance of mining equipment, major losses in tonnage output, high pressure on maintenance staff to fix breakdowns and no respite from crisis management environments.

The Production Department is dissatisfied with the Maintenance Department and the Maintenance Department is dissatisfied and frustrated by the lowly place they occupy on the corporate value-adding totem pole.

I further suggest that the rate of technological development is being slowed by the lack of a receptive and competent operational environment.

A latent barrier to extracting the benefits of new technology is the lack of fluidity of the operational and maintenance systems to readily adopt the changed circumstances and meet the demands for new skills.

The major challenge for maintenance in Roadway Development Systems is therefore to undertake a strategic review process to evaluate whether existing maintenance practices and performance is at a sufficient level of development to support the next generation of equipment and an even more challenging, future technologies.

It follows that if gaps in performance are identified, opportunities can be defined and action plans developed for implementation.

A snapshot strategy may be as summarised by:

- Where are we Now?
- Where do we Want to be?
- How do we get there?

TECHNOLOGICAL CHALLENGE

What challenges will technology present? For discussion purposes some ideas can be gained by integrating known and projected technologies (See Figure 1).

Conclusions that can be inferred by this discussion are:

1. Equipment productivity and safety will be improved by technology through the removal of the operator from the immediate development face.
2. Machine availability will be improved by advanced condition monitoring and real time stress level control.
3. Remote or autonomous operation of equipment will be assisted by smart sensors and smart actuators.

In an engineering design sense, electronic, mechanical and hydraulic systems will be more closely integrated requiring a greater depth of design knowledge to produce a workable product.

In an operational sense the performance of the mining process will be frozen at the time of design by virtue of the selection of technologies. It is likely that the effectiveness of the equipment will increase at the expense of flexibility of application.

The maintenance system will move towards a more pivotal role in the mining process. The real role of maintenance will emerge as a creator of production capacity by concentrating on technology support.

Fig 1	OPERATOR NEEDS - ROADWAY DEVELOPMENT	

1. Consistent development rate in variable geological conditions.
2. Increasing productivity performance:
    - People tonnes/man hour
    - Capital tonnes/investment dollar
    - Operational $/tonne
3. Reliability of the roadway development process.
4. Safe working environment - Occupational Health & Safety Issues.

SYSTEM AND SYSTEM CONCEPTS

CURRENT	NEXT GENERATION	FUTURE
Coal Cutting Continuous Miners Road Headers	Remote Simultaneous Bolting/Cutting	Automatic Miner/Support System Non-visual Contact Control
Roof Support Hand-held Bolters Machine-mounted Bolters Bolter/Miners Mobile Road Bolters	Remote Mobile Roofbolter with mechanised tool handling	Roof Support Automatic and integrated with automatic miner
Coal Haulage Shuttle Cars Mobile Boot-ends Scoop Trams	Relocatable conveyors/ Mobile Boot-ends	Self-guiding Vehicles Self-advancing flexible conveyor Capsulated Coal Transport
Mining Environment Control Hand-held	Routine Gas/Ventilation and Geological Stress/ Deformation Monitoring	Predictive Gas and road condition reporting

KEY TECHNOLOGIES

CURRENT	NEXT GENERATION	FUTURE
Analogue, hard-wired communication	Digital cellular network	Voice, data, vision by wireless network
Direct measurement of machine condition	Predictive remaining life monitoring	Expert root cause failure generator
Data display on board	Remote overview	Mobile and remote data displays
Manual laser guidance	Auto laser guidance	Spatial navigator
Visual roof and floor detection	Ground sensing radar Remote visual display and control	Spatial navigator
Machine control, PLC and relay logic radio remote	"On board" control networks Micro-processor C.A.N. technology	Smart sensors single card computers C.A.N.
Machine overload protection fuse or shear pins	Direct stress measurement	Real time life prediction

Figure 1.

GROWING EXPECTATIONS OF MAINTENANCE

First Generation:	*Second Generation:*	*Third Generation:*
• Fix it when it broke	• Higher plant availability • Longer equipment life • Lower costs	• Higher plant availability and reliability • Greater Safety • Better product quality • No damage to the environment • Longer equipment life • Greater Cost Effectiveness
BREAKDOWN	**PLANNED/PREVENTIVE**	**PREDICTIVE**

1940 **1950** **1970** **2000**

CHANGING MAINTENANCE TECHNIQUES

First Generation:	*Second Generation:*	*Third Generation:*
• Fix it when it broke	• Scheduled overhauls • Systems for planning and controlling work • Big, slow computers	• Condition Monitoring • Design for reliability and maintainability • Hazard Studies • Small, fast computers • Failure modes and effects analyses • Expert Systems • Multiskilling and teamwork

1940 **1950** **1970** **2000**

Figure 2.

MAINTENANCE SYSTEMS

If we accept that equipment will become more complex to satisfy the mining and operational needs, the capacity of the maintenance process will need to be checked. The *'break and fix'* approach will be out of its depth unless the operators and designers do a truly exceptional job in specifying, designing and operating future roadway equipment.

Maintenance technology, the understanding of what maintenance's role is, the underlying theory and the organisational processes are undergoing rapid development.

The technology of maintenance has emerged during the last 10 – 15 years as a definitive discipline of engineering. The needs of the nuclear, aviation and aerospace industries have driven the research and practical application of the new methodologies of maintenance are numerous.

Accepted maintenance principles now include:

- Maintenance is not a way of making up for the deficiency in design and application,
- A machine needs maintenance when it cannot reach its potential performance through reason of deterioration or degeneration of its systems,
- Maintenance can only ensure that a machine with inherent capability and reliability will achieve its production performance,
- No amount of maintenance can sustain a machine beyond its potential,
- The machine may operate and perform its functions, however if it does not achieve its desired *operational performance* standard it has failed, and

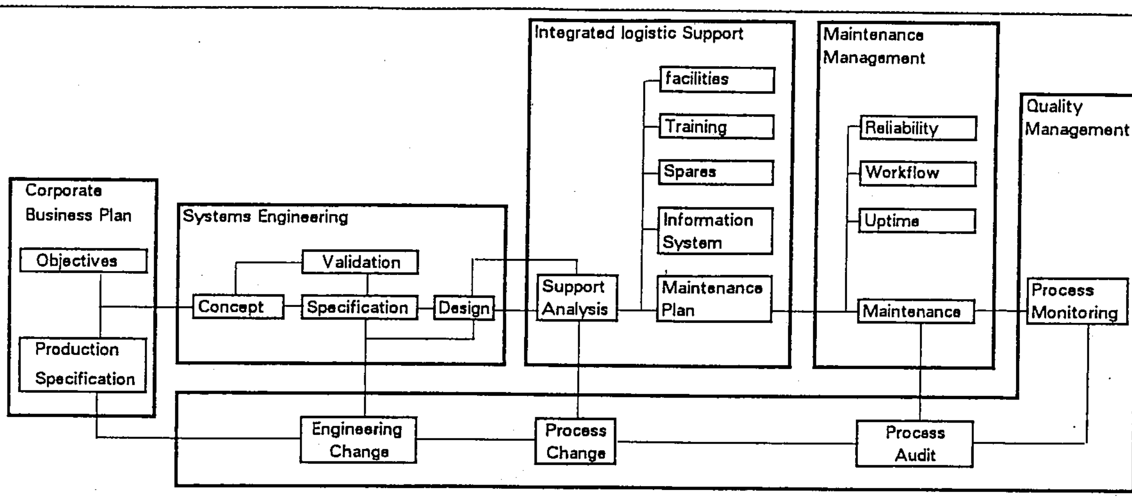


Figure 3. Maintenance support strategy model.

- Failure is defined when a machine will not meet its required performance. This does not rely on a breakage or a total shutdown to create a failure.

The mining process is a compatible arrangement between people and equipment to achieve a production objective. These two components must work in harmony, damage to either party leads to a failure situation.

To achieve this harmony the actions of people must be constrained by the limitation of themselves and the equipment in the specific operating environment.

Maintenance is moving rapidly from a *'fix and break'* approach, through a *'preventative/ scheduled overhaul work programme'* to a system based on condition monitoring, reliability analysis and teamwork.

Moubray (Moubray, 1991) in his book RCM II illustrates these points as shown in Figure 2.

The underground coal mining industry in Australia, at best estimate, could be judged to be in a transition from 'fix and break' to *'scheduled maintenance'*. Perhaps a few companies could be seen to be active in the third generation area.

ADVANCING MAINTENANCE DISCIPLINES

Maintenance theory and application is now moving to a level of maturity where the activities and techniques once used in isolation of one another are brought together to directly relate to the business objective.

Concepts such as:

- Life cycle costs
- Reliability and maintenance
- Overall Operational Effectiveness
- Failure Modes
- Maintenance Task Analysis

are merged by the Japanese manufacturing industry under the banner of TPM, Total productive Maintenance (Nakajima, 1988).

T.P.M. is a tightly integrated package of operators, maintainers and technical development groups working together in a continuous improvement environment.

For our purposes a simplified model of integrated maintenance support is represented in Figure 3.

This model is a disciplined and iterative approach to the management of assets to:

- Involve maintenance considerations into systems and equipment design.
- Develop maintenance requirements that relate to business objectives and system designs.
- Analytical base for maintenance tasks.
- Provide maintenance during operations at minimum cost.

This model illustrates that maintenance needs to be considered early in the technical development phase as a high percentage of maintenance costs are locked in at the conceptual stage (Blanchard, 1991).

MAINTENANCE AUDIT

While the challenges of future technology and recent views of maintenance technology have been discussed, the question remains as to how well contemporary maintenance systems in Australian underground mining rate.

To assist here the following is provided:

1. ACARP Study into Maintenance Practices was conducted during 1994 - the results are about to be published.

Innocence → Awareness → Understanding → Competence → Excellence

● Unpredictable Metres/Shift

- Poor equipment availability
- No cost control
- Breakdown maintenance
- High level of spares
- No preventive maintenance
- Little planned work <50%
- No Work Orders and schedules

- Average plant availability
- Good cost control
- Time based P.M. plan
- Inventory control of spares
- Systematic maintenance
- Planned work to 70%
- Supervisor/Trades 1:30

● Consistent Metres/Day

- Targeted plan availability
- Cost control by equipment and job
- Condition-based maintenance
- Integrated inventory J.I.T.
- Planned work >85%
- Fully integrated information system
- Supervisor/Trades 1:5

Figure 4.

2. Benchmarking services for maintenance organisation are now available on a national and international scale.

An immediate and instructive evaluation of the position of a maintenance organisation can be made by reference to a scale of performance characteristics illustrated by Figure 4.

This evaluation method is very useful in gaining a snapshot perception of how a maintenance system is operating. It also carries the notion that maintenance systems are capable of various stages of development with related organisational performance characteristics.

PROFILE OF SUCCESSFUL MAINTENANCE

DuPont (Larson, 1992) have developed more formal systems for auditing and promoting the growth of maintenance systems.

DuPont emphasises eight areas for focussing development activities and have rated the relative importance of each element:

■ Leadership	20%
■ Reliability Improvement	20%
■ Planned Maintenance	15%
■ Predictive/Planned Maintenance	15%
■ Human Resource Development and Training	10%
■ Contracted Maintenance	5%
■ Research and Technology	5%

The clarity of the profile of what they regard as successful maintenance organisations is interesting by itself. The real point suggested by the rating system is that successful organisations concentrate on leadership issues and concentrate on analysing and planning maintenance activities.

Leadership is the top priority for their development program and it encourages the maintenance function to step outside its normal corporate boundaries to encourage other disciplines, notably operations and supply, to participate in the process of maintenance.

This underlines the challenge for maintenance groups when future technologies are being evaluated to get in at the ground floor - not after the concepts are frozen.

CONCLUSION

To summarise, technology will bring forward productive changes only if maintenance functions provide a proper supportive environment.

The supportive environment can be generated by the development of a maintenance culture that enables maintenance to become a key value-adding component of the business and a core competency for leading edge roadway development groups.

WHERE WE ARE NOW?

- ■ Breakdowns
- ■ React to Problems
- ■ System Degeneration
- ■ Roadway Development delays interrupt Longwall Production
- ■ Technology creates as many problems as it solves

hand and right hand roof bolters and rib bolters. Operator platforms are provided for all three stations. Starting the machine is under the control of the 'machine' operator. All operators can shut down the machine independently.

Normal machine operation can be controlled from the remote radio control.

The system is designed to operate in 1m cycles. The cutter head, loading apron and conveyor are advanced in increments at the discretion of the machine operator until 1m of advance has been achieved. The chassis and roof bolter module are then advanced to re-start the cycle. A cutter head position indicator with digital display is provided at the machine operator station.

The planned method of operation for the KBII with its expected advance rates necessitated a detailed study of the environment in which the operators would work. The requirements were identified as:

- Adequate ventilation of the face and work areas,
- Dust free work area particularly for the bolter operators,
- Minimum noise levels, and
- Minimal maintenance

STABILITY AND STEERING FACILITY

To enable simultaneous cutting, loading and roof bolting it is imperative that the stability of the mainframe and drill module is maintained during sumping and shearing of the cutter head.

To ensure this stability the KBII incorporates an apron float system which utilises the bell crank lever mechanism, normally used to lift the loading apron for flitting, to apply a variable upward force on the loading apron. Since the reaction from the lifting cylinders is absorbed in the mainframe then adjusting the applied and reactive pressures in the float circuit allows the weight distribution between the mainframe and the apron to be adjusted to suit local conditions. It also follows that any adjustment in mainframe loading will affect the crawler ground pressure. The float system operates automatically when sumping and tramming.

Horizontal steering of the machine is affected by undercutting or overcutting the loading apron in maximum increments of ±30mm. The float circuit, active in sump mode, ensures that the apron follows the adjusted horizon.

GENERAL COMMENTS

Major advantages which evolved from the development are as follows:

- Stability and steering facility.
- Ability to cross steer the cutter head.
- Flat-top profile for ease of material storage and handling.
- Onboard scrubber system.
- Cutting reaction totally separate from the chassis.
- Optimum position of rib bolters.
- Good working range for a single machine, i.e. 1.8m - 3.2m.

THE FUTURE IN OUR SIGHTS

We now consider the KBII Continuous Miner to be a machine that is beyond the initial development stage, our experience with the machine whilst limited has assisted us greatly to identify areas of the machine which required attention or modification.

What we need now more than anything else is "runs on the board" to convince the coal mining industry both in Australia and overseas that the KBII is a viable or even preferable alternative to other roadway development machines that are currently available.

We have established our objectives for the KBII as follows:

1.0 Short Range

1.1 To establish one KBII continuous miner on a short term trial basis achieving good results in typical Australian mining conditions.

1.2 To make a number of offers to the marketplace for the trial of a machine on another commitment basis subject to performance.

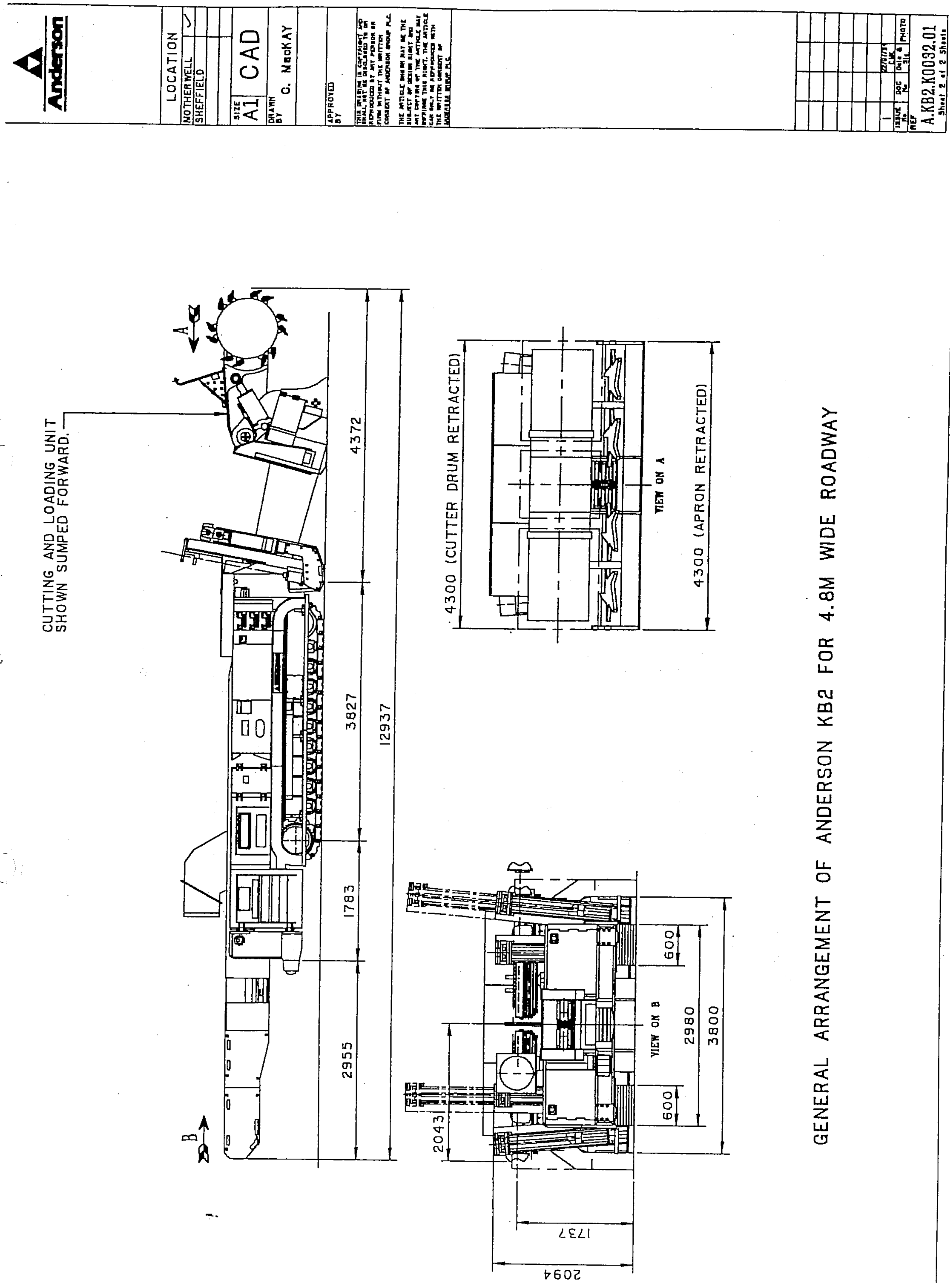
1.3 To apply all necessary resources to convince the Australian marketplace of our level of commitment to the KBII.

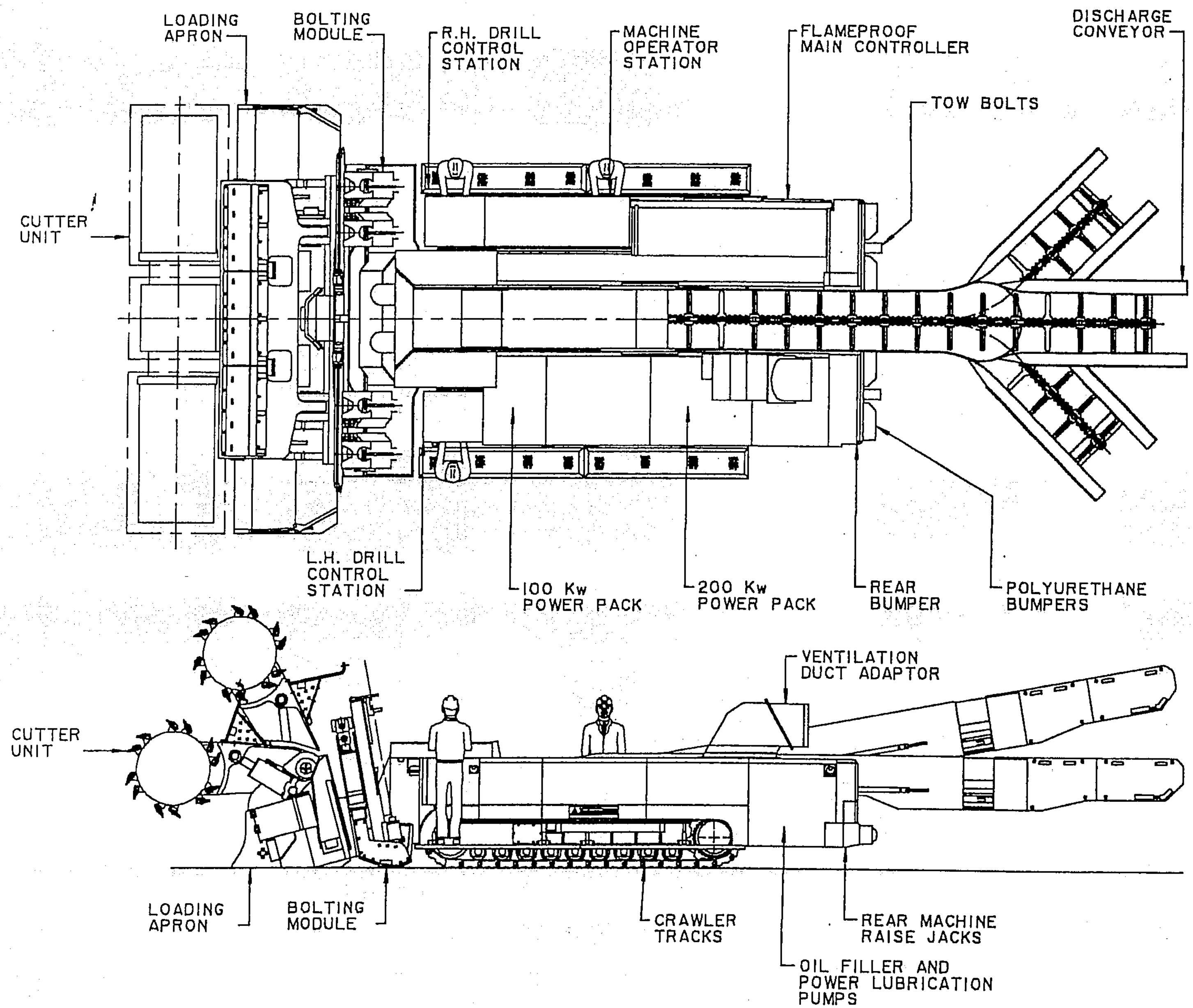
2.0 Intermediate Range

2.1 Apply all necessary resources to ensure that the first trial machine does achieve the required results during the trial period.

2.2 Achieve an order for the KBII at the mine where it is on trial.

2.3 Exploit all good performance achieved with the machine and ensure that the


Anderson
LOCATION
MOTHERWELL
SHEFFIELD
SIZE A1
CAD
DRAWN BY C. MacKAY
APPROVED BY
ISSUE
DOC No
REF A.KB2.K0032.01
Sheet 2 of 2 Sheets
CUTTING AND LOADING UNIT SHOWN SUMPED FORWARD.
A
B
4372
3827
1783
2955
12937
4300 (CUTTER DRUM RETRACTED)
4300 (APRON RETRACTED)
VIEW ON A
VIEW ON B
600
2980
3800
600
2043
1737
2094
GENERAL ARRANGEMENT OF ANDERSON KB2 FOR 4.8M WIDE ROADWAY



GENERAL ARRANGEMENT OF ANDERSON KB2 SHOWING MAJOR UNIT LOCATIONS

Anderson

LOCATION
MOTHERWELL ✓
SHEFFIELD

SIZE A1 CAD

DRAWN BY C.MacKAY

APPROVED BY

ISSUE No	DOC No	Date & Sig	PHOTO
1		[illegible]	

REF A.KB2.K0032.03

Australian market place is aware of the machine performance level.

2.4 Achieve additional orders for the KBII continuous miner at additional sites using limited trial offers to attract potential customers initially.

3.0 Long Range

3.1 To establish the KBII Continuous Miner as a primary product for Anderson Rea together with our Longwall Shearer.

3.2 To enable Anderson Rea to grow with an increased yet complementary product range.

3.3 To offer greater security for Anderson Rea Mining Customers based on an expanded product range offering on-going levels of work for the Company.

3.4 To achieve 50% plus market share of the available continuous miner market in Australia.

In order for the KBII to realise its full potential at any mine the infrastructure to support the machine in the way of coal clearance and materials handling must of course be compatible.

At Anderson we recognise these areas of vulnerability and in the area of materials handling we are working closely with other suppliers in the development of a material handling system to complement the KBII. We recognise that the most significant success will be achieved with a totally integrated roadway development ***system***.

CONCLUSIONS

The KBII has evolved from long experience with firstly continuous miners with hand held bolters, then continuous miners with machine mounted bolters which were still cyclic in operation to the current stage of development where cutting, loading and bolting simultaneously has been proven to be successful.

Experiences to date have shown that machines such as the KBII which are single pass machines, with fully mechanised bolting systems and capable of simultaneous cutting, loading and bolting are pointing the way ahead in achieving the advance rates required to match the performance of high production longwall faces. The working environment and safety of the machine operators is also greatly enhanced.

Further developments in coal clearance, materials handling, ventilation and roof bolting will evolve which will allow this new breed of machine to maximise its potential and we are confident that the KBII will prove successful in this critical area of mine operations.

SPECIFICATION

GENERAL

Weight	85t (approximately)
Total installed power	500kW
Length	12.94m
Height	1.35 (chassis) 1.45 (boom pivot)
Width	4.3m (apron & cutter retracted) 2.98m (crawler tracks)
Ground clearance	0.30m
Power supply	1050 Volts @ 50 Hz
Maximum cutting height	3.20m
Cutting width	4.8 to 5.2m
Minimum operating height	2.20m (approximately)
Minimum flitting height	2.2m
Tracking speed	0–16m/min
Cutting rate	Mean–5t/min
Loading rate	8t/min
Conveyor capacity	16t/min
Cutter drum diameter	1200mm
Cutter rpm	51.8
Cutter motor power	2 x 100kW
Conveyor speed	65m/min
Power to conveyor	42kW
Loading system	Spinner
Operating speed	79 rpm
Power to loading system	35kW (17.5kW each side)
Trace pressure	2.33kG/cm (cutting)
Power to track	166kW (83kW per side)

John Pointer

AUSMINCO

ROOF BOLTING AUTOMATION FOR UNDERGROUND COAL MINES

A fully automated single bolt cycle roofbolting drill has recently been successfully demonstrated in simulated surface trials.

The underground application of fully automated roofbolting drills in a production environment is less than a year away.

BACKGROUND

What seemed the most obvious starting point for an automatic bolting system for the coal industry was the adoption of a successful automatic hard rock bolting system. All the major tunnelling and rock bolting companies had such systems in operation through out the world. In 1985 NERD&DC funded a proposal from Clarence Colliery and Atlas Copco for development of an automatic bolter based on a successful Atlas Copco bolting system used in hard rock.

From a bolting operations point of view, the Atlas Copco system performed well as a manually operated unit. During its trials it set over 1000 bolts with only a few problems, mainly due to the resin cartridge insertion. However, from space utilisation point of view it was obvious that the bolter was too large for use on any of the existing or planned new generation miners. Even though the bolting system shared many parts with the production hard rock model, the project exhausted its funding before a fully automatic system could be put on trial.

This project provided a valuable lesson concerning the design of bolters for Australian coal mining conditions. It was not possible to simply adapt machines that were successfully used in the hard rock or tunnelling industries. The successful autobolter would have to be specially designed for the cramped and difficult conditions that exist in Australian coal mines. This process would not be a low cost or simple exercise.

Further projects to develop automated roofbolting drills have been initiatives (some with industry support) of equipment manufacturers or individuals such as Joy Manufacturing, Olsen, Fletcher and Ausminco (formally Swiss Industrial Co. Australia).

Apart from the Ausminco project, funded in part by ACARP, none of these systems have been trialled in Australia.

DEVELOPMENT FACTORS AFFECTING DESIGN

The two basic factors driving the development of the new bolters are Productivity and Safety.

For this purpose productivity can be measured in rate of roadway development. Most of the new generation machines use the continuous cut and bolt or similar arrangement. The efficiency of this arrangement is locked into the effective rate of bolt insertion and fixing. Therefore the bolting system must complete its cycle in the permitted time and perform at least as reliably as the mining machine. More than one bolter will be fitted to each mining machine, a situation that could prove a maintenance nightmare unless the bolter has been designed for simple and quick maintenance.

The other factor concerns the safety of the workers at the face. The new mining machines do provide immediate support and a measure of protection. However the automated bolter has the potential to deliver a consistent result and to reduce the reliance on the skill of the operator. The automation package should also reduce the number of workers at the face.

A significant consideration is the overall size of the package. The space available on the new generation road heading machines is not unlimited. An automatic bolter has to control a significant number of functions, and the resulting control package has significance in both size and cost.

The preferred mode of operation is automatic with little or no requirement for human intervention. For difficult working conditions the ability to use remote control is advantageous, while manual override would be a necessary part of maintenance and repair.

Additional factors influencing design were the requirement to use conventional 'off-the shelf' drill steel, chemical anchors, bolt and washer plates etc.

THE AUSMINCO AUTOBOLTER

(Refer Figure 1).

The development of the Ausminco Compact Autobolter commenced in late 1989 and progressed through a lengthy feasibility study including CAD modelling to find the most compact arrangement to suit the limited space provided on Continuous Miners.

A full size detailed model was then constructed which proved to be very useful in both demonstrating the concept behind the machine and understanding its complexities.

The project proceeded steadily through the design and detailing stage including the number of significant changes which were improvements on the original concept.

The final design of the Compact Autobolter consists of four broad groups, The Drill and Feed Group, the Carousel Group, the Base and Valving Group and the Electronic Control Group.

The drill and feed group

This consists of a modified tried and proven model LHU Telescopic Feed and HDR Rotary Drill.

Modifications to the Feed include;

- A new top plate with concealed sensors to detect washer plate installed and drill head at the roof.
- A slide detection assembly which contains a number of sensors to monitor the position of the drill head during its travel up and down the feed.
- A modified timber jack assembly to include an innovative cable management system for the top plate sensors and a roof height detection sensor.

Modifications to the Drill included an inbuilt Dolly, and although this feature was not in the original design its inclusion resulted in removing a complex Dolly handling subgroup.

The carousel group

This consists of large diameter hollow centre post about which the drill steel, the chemical resin tube and the roof bolt rotate.

A hydraulic motor drives the Carousel through a set of gears. This drive mechanism has a number of sensors to monitor the position of the Carousel and a locking pin to ensure exact radial location.

Hydraulic clamps with inbuilt sensors either retain, guide or release their respective drill steel, resin tube or roof bolt as dictated by the Electronic Control Module.

The Carousel Group also has sensors to detect the presence of the drill steel, the resin tube and the roof bolt.

The base and valving group

As the design progressed it became apparent that the number of interconnecting hoses between the actuators and the remote mounted control valves would become troublesome.

It was therefore decided to mount the valving on the base plate and internally port this base plate to the various actuators. This however did increase the length of the base.

A lot of ingenuity was needed to achieve this goal, however the end result of a compact arrangement with minimal exposed hoses was worth the effort.

The original intended supplier for these valves was ACIRL however due to the restructuring of that company the development of their valves was not continued and an alternate valve manufacturer was engaged to complete this portion of the project.

The solenoid valves used are the low wattage intrinsically safe type.

These solenoid valves pilot control the directional control valves and together they all mount on a steel block which is ported directly into the base plate.

The electronic control module group (ECM)

The ECM may be further divided into four major sections.

- Operator Console:

  On the prototype unit, a radio transmitter was used as the operator console. The transmitter console consists of eleven switches to allow individual control of each function of the autobolter as well as the autocycle.

- Input Section:

  This section interprets the various sensors fitted to the autobolter to determine the position of the cassette and the drill head, the state of the drill and bolt clamps, as well as detecting presence of a washer plate, drill steel, chemical and bolt steel.

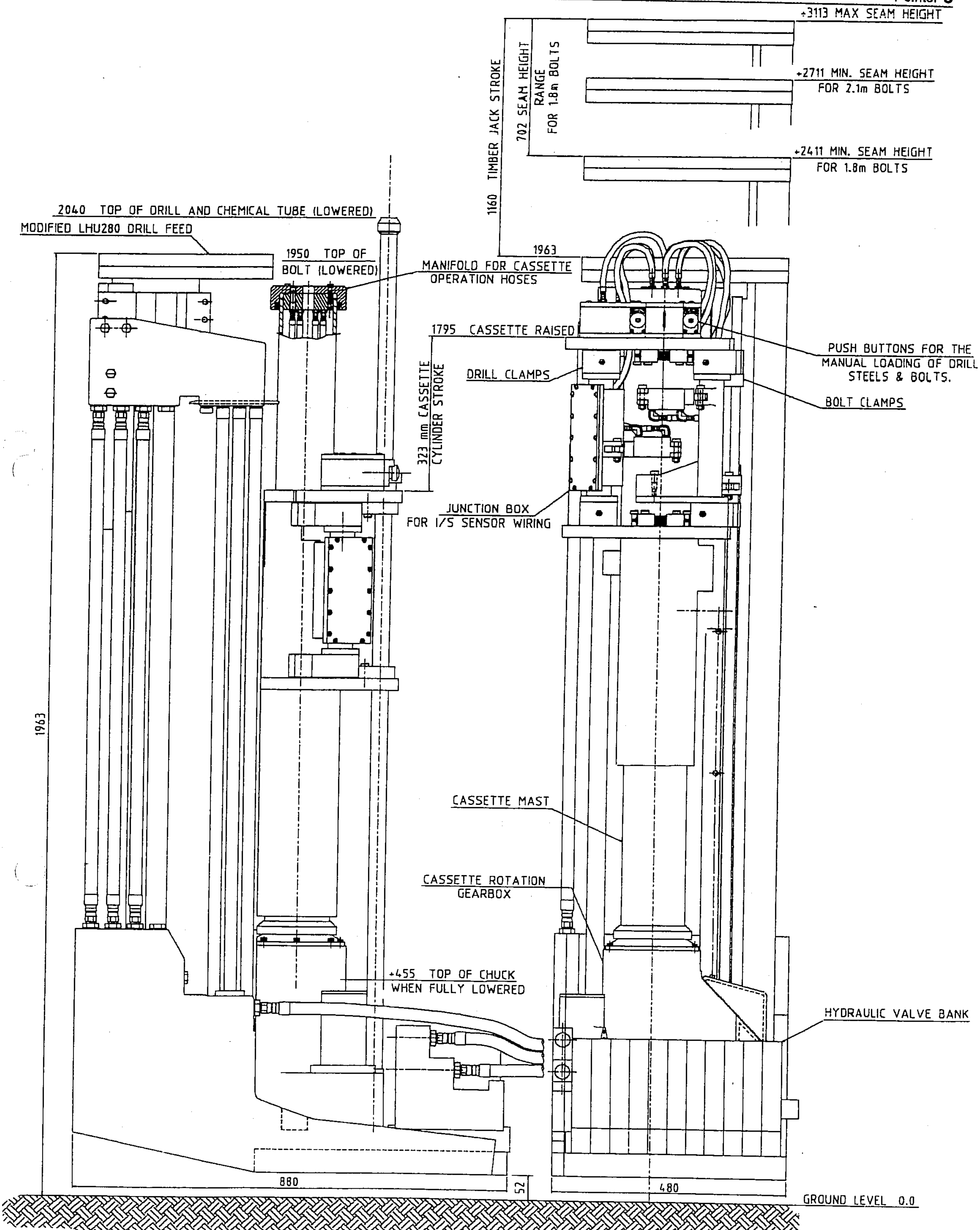

+3113 MAX SEAM HEIGHT
+2711 MIN. SEAM HEIGHT FOR 2.1m BOLTS
+2411 MIN. SEAM HEIGHT FOR 1.8m BOLTS
1160 TIMBER JACK STROKE
702 SEAM HEIGHT RANGE FOR 1.8m BOLTS
2040 TOP OF DRILL AND CHEMICAL TUBE (LOWERED)
MODIFIED LHU280 DRILL FEED
1950 TOP OF BOLT (LOWERED)
MANIFOLD FOR CASSETTE OPERATION HOSES
1963
1795 CASSETTE RAISED
PUSH BUTTONS FOR THE MANUAL LOADING OF DRILL STEELS & BOLTS.
DRILL CLAMPS
323 mm CASSETTE CYLINDER STROKE
BOLT CLAMPS
JUNCTION BOX FOR I/S SENSOR WIRING
1963
CASSETTE MAST
CASSETTE ROTATION GEARBOX
+455 TOP OF CHUCK WHEN FULLY LOWERED
HYDRAULIC VALVE BANK
880
52
480
GROUND LEVEL 0.0

Figure 1.

- Solenoid Driver:

  Contains circuitry to drive each of the nineteen solenoids fitted to the autobolter to perform the various functions.

- Microprocessor Control:

  The heart of the ECM is the microprocessor control section. This section acquires data from the operator console and the input section and then determines which of the solenoids should be operated to perform the requested function.

As previously mentioned, individual control of each of the bolters functions is provided by the operator console to allow the operator to adjust the cycle if required. PAUSE and RESUME functions are also provided to allow the operator to halt the autocycle at any time. The operator may then use the individual controls to, for example, withdraw the drill steel and replace it if it has broken. He may then resume the autocycle to complete the operation.

Certain parameters are user adjustable, for example, mix and set times may be adjusted to allow the use of different chemicals.

On the prototype unit, a four line, forty character LCD was fitted to show and/or alert the operator to the various fault conditions, such as "Drill Steel not in place", as well as displaying what stage in the autocycle the ECM was currently performing.

To simplify maintenance, a solenoid self test is included to isolate faulty solenoids. The ECM also tests the operator console, allowing the operator to determine faults such as faulty switches or low battery conditions.

Following is the autobolter operational sequence. The Preliminary and Reloading operations are manual. The Drilling, Chemical Insertion and Bolt Insertion Operations are completely automated.

During the surface trials a 1.8m bolt was fully installed in 105 seconds from pressing *"Auto Cycle Start"* (Step 3) to the machine returning to rest (Step 46).

AUTOBOLTER OPERATIONAL SEQUENCE

OPERATION NO:	OPERATION DESCRIPTION
PRELIMINARY	
1	Put washer plate onto pad of timberjack
2	Put W-strap onto pad of timberjack
3	Press "Auto Cycle Start"
DRILLING	
4a	Drive timberjack to roof (erect W-strap)
4b	Lift cassette from Level CI to CII (323mm)
4c	Withdraw indexing pin to unlock cassette
5	Slew in cassette to CL of drill same as CL of chuck (rotates from position 0° to position -90°)
6	Insert indexing pin to lock cassette
7	Drill gripper valve to grip position
8a	Turn on R.H. rotation of chuck at low RPM (to engage drill in chuck)
8b	Advance drillpot slowly from Level BI (+453) to Level BII (+583). This level BII allows 20mm overstroke to successfully engage the drill square drive.
9	Advance drill pot at high speed from Level BII (+583) to Level BV, which is a level that varies with seam height and results in the drill tip being 100mm from the mine roof.
10a	Turn on flushing water
10b	Wait 0.5 sec.
11a	Turn on high speed R.H. rotation
11b	Advance drill pot at high speed to level BIII (+1333) to collar drill tip.
12a	Open drill grippers
12b	Withdraw indexing pin to unlock cassette

13	Slew out cassette (to clear driveway of drillpot) (cassette rotates from position -90° to position of 0°)
14	Insert indexing pin to lock cassette
15	Advance drillpot to level BIV (this position is variable depending on seam height)
16a	Retract drillpot at high speed feed to level BIII
16b	Slow down chuck RPM
16c	Turn off flushing water
17	Withdraw indexing pin to unlock cassette
18	Slew in cassette so CL of drill grippers same as CL of chuck
19	Insert indexing pin to unlock cassette
20	Close drill steel grippers and maintain valve in grip position
21	Retreat drillpot at high speed feed to level BII
22	Turn off rotation of chuck
23a	Turn on LH rotation of chuck at reduced RPM
23b	Retreat drillpot slowly to level BI (to disengage chuck from drill steel)
24a	Turn off LH rotation of chuck
24b	Withdraw indexing pin to unlock cassette
24c	Drill grip valve to neutral position

CHEMICAL INSERTION

25	Slew in cassette to that CL of resin injection tube same as CL of chuck (from position -90° to -180°)
26	Insert indexing pin to lock cassette
27a	Turn on RH rotation of chuck at low RPM
27b	Advance drillpot at slow speed until top of resin tube is at bolt plate. (This position would vary with seam height)
28a	Turn off RH rotation
28b	Turn on flushing water (to inject resin cartridges) 5 sec.
29a	Turn off flushing water
29b	Turn on RH rotation of chuck at low RPM
29c	Retreat drillpot to level BII
30	Reverse chuck rotation from slow RH to slow LH
31	Retreat drillpot to level BI

BOLT INSERTION

32a	Withdraw indexing pin to unlock cassette
32b	Turn off chuck rotation
33	Slew cassette so that CL of bolt same as CL of chuck (from -180° to +90°)
34	Insert indexing pin to lock cassette
35	Bolt gripper valve to clamp position
36	Turn on RH rotation at low RPM (2 sec) to engage nut
37	Advance drillpot slowly (to engage drillpot with bolt nut) to level BIIA
38	Advance drillpot slowly to position BIIIA
39a	Open bolt grippers
39b	Withdraw indexing pin to unlock cassette

40	Slew out cassette (to clear driveway of drillpot)
41a	Insert indexing pin to lock cassette
41b	Speed up RH rotation to mixing rate, timer set to correct mix time
41c	Advance drillpot at high speed
42	Turn off advance of drillpot (wait until resin mixed)
43	Turn off rotation of the chuck (wait until resin hardened)
44	Turn on RH rotation at slow speed (to tighten nut) and slow feed
45a	Turn off chuck rotation
45b	Retreat drillpot to level BI
45c	Lower cassette to level CI
45d	Retreat timberjack
45e	Turn on water
46	Turn off water after 1 second

RELOADING

47	Replace resin injection tube with recharged one
48	Fit roofbolt to cassette
49	Check condition of drill bit
50	Reposition drill mast and align with the next bolting position

Whilst the Ausminco Autobolter has successfully demonstrated the automation of a single bolt cycle, further development was required to reduce the complexity inherent in the current prototype and facilitate the full automation of multiple bolt cycles.

These issues have been addressed, and a modified *"Mark II"* version will be trialled in the near future.

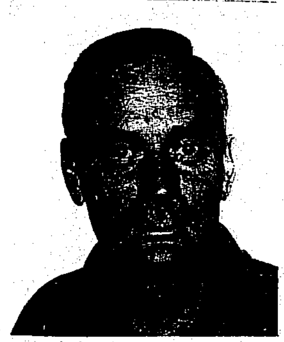
Space and weight restrictions with the installation of automated roofbolting drills onto new generation continuous miners have constrained the development of fully automated roof bolting drills to date. Such constraints do not exist to the same degree with the application of automated roof bolting drills on independent bolting machines.

Emerging trends in mining methods based on the use of independent machines for roofbolting operations will significantly improve the capacity for fully automating roofbolting drills.

New innovations such as the screw bolt make automation significantly easier by removing the complexity and size associated with automating the chemical insertion sequence in a conventional bolt installation. With the success of the screw bolt, (or a similar type of bolt), and the automation of roofbolting drills, bolt installation times consistently under 60 seconds becomes a real possibility.

REFERENCE

RICHARDS, O. & SHEPPARD, L., 1992: Automated Bolting Rig Developments. *(Underground Roadway Support Systems - Safety and Productivity).*



John Pointer

OUTBURST CONTROL

The paramount issue behind the need for outburst control in underground coal mining is that of safety. The importance of this issue has unfortunately been reinforced by a number of fatalities resulting from outburst events. These events have occurred during the roadway development process. The link has therefore been established between the urgent need for improved outburst control and the roadway development process.

At this time the outburst phenomenon is most prevalent in the New South Wales south coast coal mines, particularly in the Bulli Seam. However, other Australian coalfields have experienced the problem and no doubt as depth of mining increases, the phenomenon will become more widespread. Research into the outburst phenomenon and outburst control will undoubtedly benefit a significant proportion of the Australian underground coal mining industry.

A second issue connected with the need for outburst control is the management of the outburst risk from an economic standpoint. It must be stressed that this issue is very much a secondary one when compared to the safety of workmen. The means of outburst control must still permit the economic extraction of reserves whilst ensuring the safety of the workmen involved. In short, the risk presented by the outburst phenomenon must be managed effectively.

This outburst control session has been fortunate in attracting a number of excellent papers that I am sure will provide an overview of the state of play as regards the outburst phenomenon. The authors have ably presented their thoughts and experience on the subject and their presentations will no doubt assist us in focussing on future research priorities and directions.

The session will commence with an Inspectorate perspective on management of the outburst risk. This will be followed by a particular Mine Management Plan as an example of managing the risk.

There will follow a number of presentations incorporating current, proposed and suggested research. Themes will include fundamental aspects of the outburst phenomenon and detection of potential outburst zones by drilling and other means.

The final presentation will give an update on progress with one aspect of remote roadway development driveage.

I am confident that the session will provide delegates with a broad overview of the outburst phenomenon which will assist them to identify and have an input into future research directions and priorities.

PHILIP EADE
BHP Steel Collieries

TERRY ABBOTT

NSW Department of Mineral Resources

UNDERGROUND DEVELOPMENT WORKSHOP — An Inspectorate Perspective on Outbursts

There is increasing pressure on underground coal mines to improve productivity levels to combat the effects of reduced coal prices, dollar exchange values and competition from open cut coal mines.

Underground coal mines in the Southern District of New South Wales are becoming progressively deeper and to maintain viability need to operate efficient longwall mining systems. Annual production levels of 2.5 to 3 million tonnes per longwall are currently being achieved in less than ideal conditions but the industry thrust has to be towards production levels of 4 million tonnes per annum or greater. To achieve productivity levels of this magnitude there is continuing pressure on mine management to increase roadway development rates.

Mining coal at greater depths demands greater roof support density and an increased need for demonstrated management skills in the areas of gas drainage and outburst control. Effective pre-drainage of gas from the seam to be mined and adjacent seams are an essential pre-requisite to rapid roadway development and longwall extraction.

The majority of Southern District longwalls operate in the Bulli Seam, the uppermost seam in the Illawarra Coal Measures. The Bulli Seam provides a high quality coking coal product for export or domestic steel making purposes. This seam at depths of 400-550 metres has a gas content of up to 18 cubic metres per tonne and a gas composition ranging from 100% methane to 100% carbon dioxide.

Increased development rates dramatically increase the potential for outbursts because gas is unable to desorb naturally from the coal as the coal face advances.

THE OUTBURST PHENOMENA

An outburst is best described as the uncontrolled violent release of gas and associated coal material into the working place during mining.

All outbursts within the Southern District of New South Wales have occurred in the Bulli Seam. The majority of outbursts (250) have occurred at the Kembla Coal and Coke owned West Cliff Colliery but a total of nine Bulli Seam mines have been deemed as outburst prone. Each of these mines have an Outburst Management Plan in place, formulated to manage the perceived different levels of outburst risk identified at each mine.

Table 1 sets out the number, location and magnitude of outbursts.

A total of 12 outburst related fatalities have occurred since (1985) and 4 of those have occurred in the last 3 years. All of these fatalities have resulted from carbon dioxide effects.

Table 2 lists outburst related fatalities.

Outburst risk is directly linked to seam gas content. Effectively drain the gas and the risk is eliminated.

The outburst risk is significantly increased by the presence of geological structures and zones of crushed coal or mylonite.

THE OUTBURST MANAGEMENT PLAN

Currently, there is no legislation in place with respect to outburst control. It is planned however, to make provision in the Coal Mines Regulation Act, 1982 for appropriate recognition of the outburst hazard. A less prescriptive approach to outburst hazard management has been adopted since 1992 requiring each outburst prone coal mine to have in place an Outburst Management Plan (OMP). The OMP can effectively take account of the different levels of risk occurring at each coal mine.

The OMP is a procedural document that sets appropriate standards for prediction, prevention of outbursts and protection should an outburst occur.

The investigation of the most recent outburst fatality on 25 January 1994 demonstrated the need for improved standards and management control within each OMP. The Inspectorate are currently auditing and reviewing OMPs to avoid a recurrence of the events leading up to the fatality of 25 January 1994.

The Inspectorate are currently developing an Outburst Mining Code for implementation by

TABLE 1
BULLI SEAM OUTBURSTS

Colliery	N° of Outbursts	Size (tonnes)	Gas	Geological Structure
Appin	11	2-88	Mainly CH_4 CO_2 on dyke	Predominantly strike slip faults; mylonite zones
Brimstone	2	30	CO_2	Mainly dyke related structures with strike slip movement
Corrimal (Cordeaux)	4	12	CH_4 & CO_2	Sheer zone associated with minor faulting and dykes
Kemira (closed)	2	60-100	CO_2	Normal fault with mylonite
Metropolitan	40	1-150	Mainly CO_2 with minor amount of CH_4	Predominantly with dykes and faults that exhibit slicken sides and mylonite
South Bulli	7	1-300	Mainly CO_2	Strike slip faults with mylonite; dyke zone and thrust fault
Tahmoor	88	5-400	Mainly CO_2	Mainly strike slip faults; dykes (110°-135°) and thrust fault: mylonite usually present
Tower	19	1-80	Mainly CH_4	Mainly strike slip faults with dyke
West Cliff	250	4-320	Mainly CH_4 with CO_2 to the NE	Predominantly strike slip faults (100°-110°) with slicken sides and mylonite; dykes and thrust faults have been associated with outbursts

TABLE 2
FATAL OUTBURSTS IN THE BULLI SEAM

Colliery	Year	N° Killed	Size (tonnes)	Gas	Structure
Metropolitan	1896	3	Unknown	CO_2	Dyke and soft faults zones
Metropolitan	1926	2	140	CO_2	Fault with 5 m throw
Metropolitan	1954	2	90	CO_2	Normal fault, 0.3 m throw
Tahmoor	1985	1	400	CO_2 & CH_4	Behind a dyke associated with strike slip movement
South Bulli	1991	3	300	CO_2	Thrust fault associated with 35 m of mylonitic coal; very high gas pressures
West Cliff	1994	1	300	CO_2	Intersection of 2 strike slip fault structures associated with 30 cm of crushed coal; extending into roof; very high gas content

June 1995. The code should serve as a model for each OMP and the process of development of the code will provide for input from industry.

The less prescriptive management plan approach to outburst control has proven to be effective in the management of outburst risk but the process requires commitment from mine management towards continuous improvement.

OUTBURST MINING PROCEDURES

The OMPs make provision for mining under outburst mining procedures above a set gas threshhold, that is a seam gas content above which there is potential for outbursts. The gas threshold value takes into account gas composition.

Outburst mining procedures limit the number of persons at the working face, particularly during the cutting cycle and provide extra protection for the continuous miner driver.

Production levels are considerably less when operating under outburst mining procedures because of reduced manning, planned waiting periods after the cutting cycle and other safeguards. Typically, productivity levels achieved under outburst mining procedures are only twenty-five per cent of the productivity levels achieved from normal mining operation.

Prior to the outburst fatality of 25 January 1994 management were prepared to expose people to severe outbursts when operating under outburst mining procedures.

Measured seam gas levels after gas drainage were sufficient to generate high concentrations of carbon dioxide and large tonnages of ejected coal. The protection afforded to continuous miner drivers, however, was considered to be adequate.

Recent initiatives implemented by the Mines Inspectorate set new standards for gas thresholds and call for remote mining techniques only, above the levels shown on Figure 1.

Considerable effort is now given to geological structure identification and improved gas drainage to allow normal mining to occur. This should limit the incidence of outbursts and significantly reduce the exposure to risk of coal face operatives.

OUTBURST RESEARCH

Individual companies have invested money into research activities to improve their understanding of the outburst phenomena. Unfortunately, the results of research do not benefit the industry as a whole. Coal operators prefer to retain a competitive edge in the market place.

Current gas drainage techniques are practically based and do not take advantage of recent advances in drilling and drainage technology.

Considerable outburst research was conducted during the period 1979 to 1983 but current outburst mining techniques again, evolved on a practical basis, accelerated by social pressures resulting from recent fatalities.

There is considerable scope for research in the areas of gas drainage and outburst control.

Effective gas drainage is a pre-requisite to efficient deep mining. The industry both in New South Wales and Queensland must face this challenge if it is to remain viable.

Long term seam degasification programs are disregarded by management because they are unable to plan far enough ahead. Long term programs can be self funding, particularly where there is a market for the gas.

Short term drainage techniques are expensive and inefficient and seriously impact on development and extraction rates.

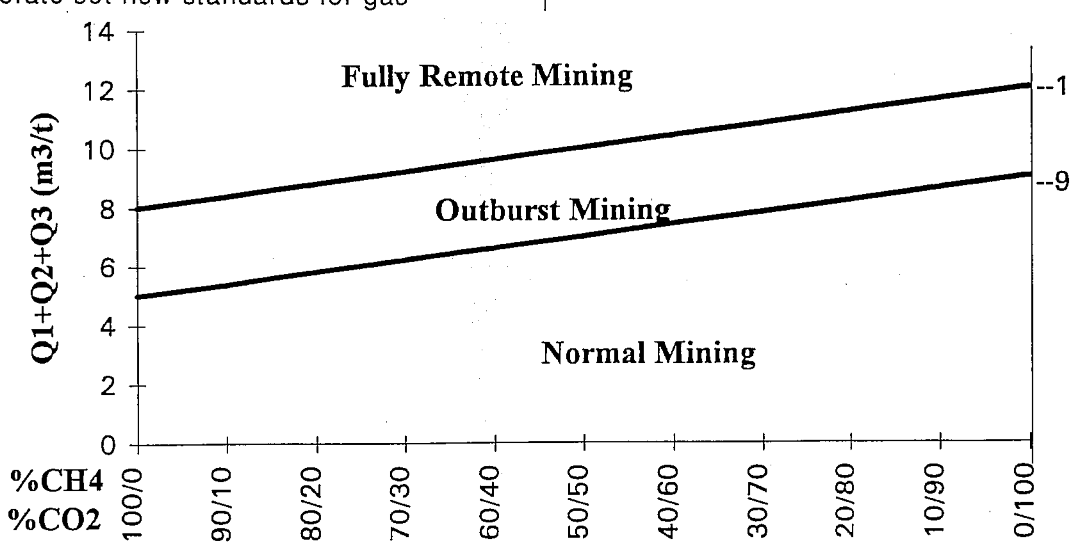


Figure 1. Gas Thresholds.

Research is needed to improve drainage efficiency by the utilisation of stimulation, dewatering and other techniques.

Drilling efficiency needs to be improved using contemporary borehole control techniques. Drilling and drainage must be effectively interfaced with longwall development and extraction.

Carbon dioxide gas is particularly difficult to drain in certain areas. We clearly do not have sufficient understanding of the gas storage and release mechanisms. Geological structures are difficult to drill and pre-drain.

What role does the residual gas component (Q_3) have in the outburst situation? Residual gas components can range from 0-7 cubic metres per tonne and drainage of this gas component is practically impossible.

In outburst prone mines, it is extremely important to be able to identify structures ahead of mining. Drilling has proven to be less than 50% effective. Other techniques such as in seam seismic and RIM techniques have had some success but more industry based research is required.

It is important to know the gas content of insitu coal immediately ahead of the face when rapidly developing roadway. New techniques are required to predict change in the seam gas regime.

Finally, if we are prepared to mine in a high gas environment, then we need to exclude people from the outburst risk. Remote mining techniques are being trialed on an individual mine basis, but an industry wide approach is required to allow sufficient funding for effective research. Remote mining should not be limited to coal cutting operations. It is just as important to mechanise roof bolting activities on a systematic basis because bolting crews traditionally work at the face without any protection being afforded. The consequences of not investing significant funds into research and not making effective use of statistical data are more frequent fatalities. Fatal accidents have the potential to close a mine, curtail an industry and sterilise a valuable coal resource.

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PETER ALLONBY

Appin Colliery

MANAGEMENT OF THE OUTBURST RISK AT APPIN COLLIERY

The Bulli seam in the Illawarra region of NSW is renowned for the risk of outburst during roadway development. The risk however should not be considered exclusive to the Bulli seam. Seams in Queensland are becoming deeper and are known to be as gassy and have similar structures to those commonly associated with Bulli seam outbursts. Outbursts have been recorded at Leichardt Colliery and at Collinsville. As mines in other areas become deeper they should also consider the risk. Bulli seam outbursts have been associated with both methane and carbon dioxide. Carbon dioxide poses a particular problem because of the toxic nature of the gas - a relatively non violent outburst can still emit fatal concentrations of gas.

Gas content threshold levels have recently been specified by the NSW Department of Mineral Resources for all Bulli seam development workings based on the perceived relative outburst risk of methane and carbon dioxide mixtures. The limit above which normal mining is not permissible and the limit above which only fully remote mining is permissible are defined. With current mining technology that upper limit is effectively the level above which mining shall cease. The threshold gas contents for Appin Colliery, shown in Figure 1, assume the balance of the sample is methane (generally methane and nitrogen).

Appin has worked exclusively in the Bulli seam throughout it's 32 year life and has experienced 21 reported outbursts, generally associated with methane gas and all associated with geological structures. These outbursts have generally been small but the largest recorded was 88t. The predominant gas at Appin is methane however the mine has for a number of years had a geologically disturbed zone running through it where the predominant gas is carbon dioxide. *In situ* gas levels at the mine of 10 - 15m^3/t are above those regarded as posing an outburst risk and mining could not take place without gas drainage.

Development under outburst mining procedures at Appin will only proceed when all feasible avenues for prevention have been exhausted. This not only exposes workers to a higher risk the development rates achieved whilst outburst mining are around a third of those achieved under normal mining. Appins Outburst Management Plan was introduced several years ago to compliment gas drainage operations in managing the outburst risk. The plan is essential to ensure that production pressures do not influence decisions relating to safety.

PRINCIPLES OF OUTBURST MANAGEMENT

Outburst management principles are categorised by the three Ps :

- **Prediction** - know what is ahead of you
- **Prevention** - take steps to mitigate the hazards
- **Protection** - provide worker protection in case the mitigation is inadequate

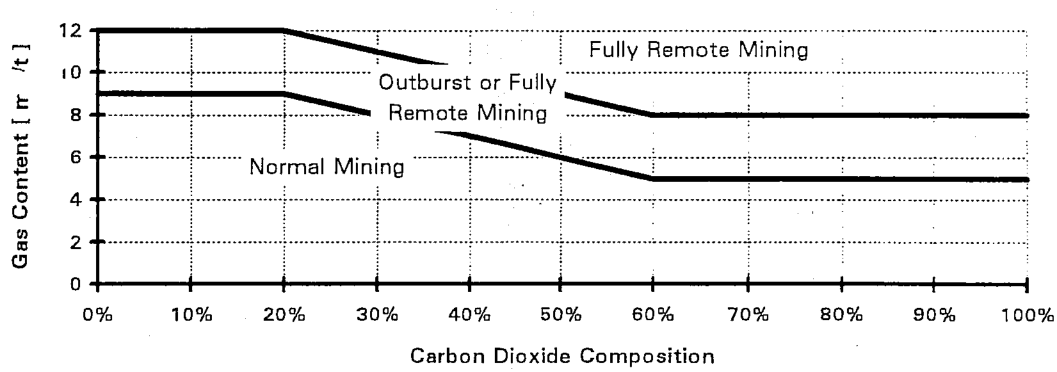

Figure 1. Appin Colliery Outburst Thresholds

THE OUTBURST MANAGEMENT PROCESS

The Ps of prediction and prevention are addressed at the weekly Outburst Risk Review meeting. The attendees use standard processes to review the available data in relation to the planned development advance for the forthcoming week.

The potential for outburst immediately around the working development face must be known. Information on the presence of structures and gas content and composition is gained from sources such as surface boreholes, extrapolation of structural mapping of nearby roadways, radio imaging, seismic etc. This information is unfortunately too regional to be of great value for outburst decision making. The information used for outburst decision making is gained from inseam boreholes which can intersect structures and can be used to take gas content/composition cores. These inseam holes may be gas drainage holes but in many cases holes are drilled specifically for outburst management.

Table 1. APPIN COLLIERY Outburst drilling June/July 1994.

Development metres advanced	2 230m
Length of inseam drainage holes	10 000m
Number of core holes	25
Length of specific core holes	800m
Number of cores	39

The outburst drilling for June and July 1994 is shown in Table 1.

Major considerations at the meeting are:

Is the data reliable and representative of the areas to be mined?

- have all structures been detected?
- do we know accurately enough where holes intersected structures?
- do we know accurately enough where cores were taken?
- are core analyses low due to proximity of gas drainage holes?
- is density of coring adequate to detect localised gas variations ie are we confident we know what is in our next cut?

Is there a need to drill for additional data?

- if data is unreliable or not representative will additional drilling provide the necessary confidence?

Are structures outburst prone?

- determined by following a standard decision making flow chart (Figure 2).

If the gas content in an area is close to outburst mining threshold can the mining operations be re-scheduled to allow further gas drainage?

- can development be re-sequenced and/or priorities changed to allow delay mining in marginal areas?
- are additional drainage holes advantageous?

Is there any other option other than to work under outburst mining procedures?

- can that driveage be significantly delayed to allow further mitigation?
- is there an alternative driveage which could eliminate the 'outburst' driveage?

Is there a need to reconvene before the next scheduled meeting?

- will information become available that should be considered before the next meeting ?

All decisions 'fail to safety' ie. no mining will take place unless it apparent that no further mitigation is feasible or further drilling will not provide meaningful data. Provided that gas contents are below the upper threshold then mine will only proceed under outburst mining procedures.

Attendees at the meeting typically include Mine Manager, Undermanager-in-Charge, Gas Drainage Engineer, Geologist, and Core Sampler (production worker). Crew members often attend and an open invitation is extended to the District Inspector of Coal Mines and Check Inspectors.

Following the meeting a mining plan and copies of the drilling information are then posted on a noticeboard for all employees to see and issued specifically to development deputies.

During development deputies look closely for outburst indicators. If they observe any untoward change they will terminate driveage until the circumstances have been reviewed, additional drilling carried out if applicable and instruction to recommence is issued by the manager.

OUTBURST MINING CYCLE

The third P is addressed by the mines standard procedure "Procedure for Mining under Outburst Conditions" which is designed to minimise operator exposure to the outburst hazard and to afford protection whilst in the

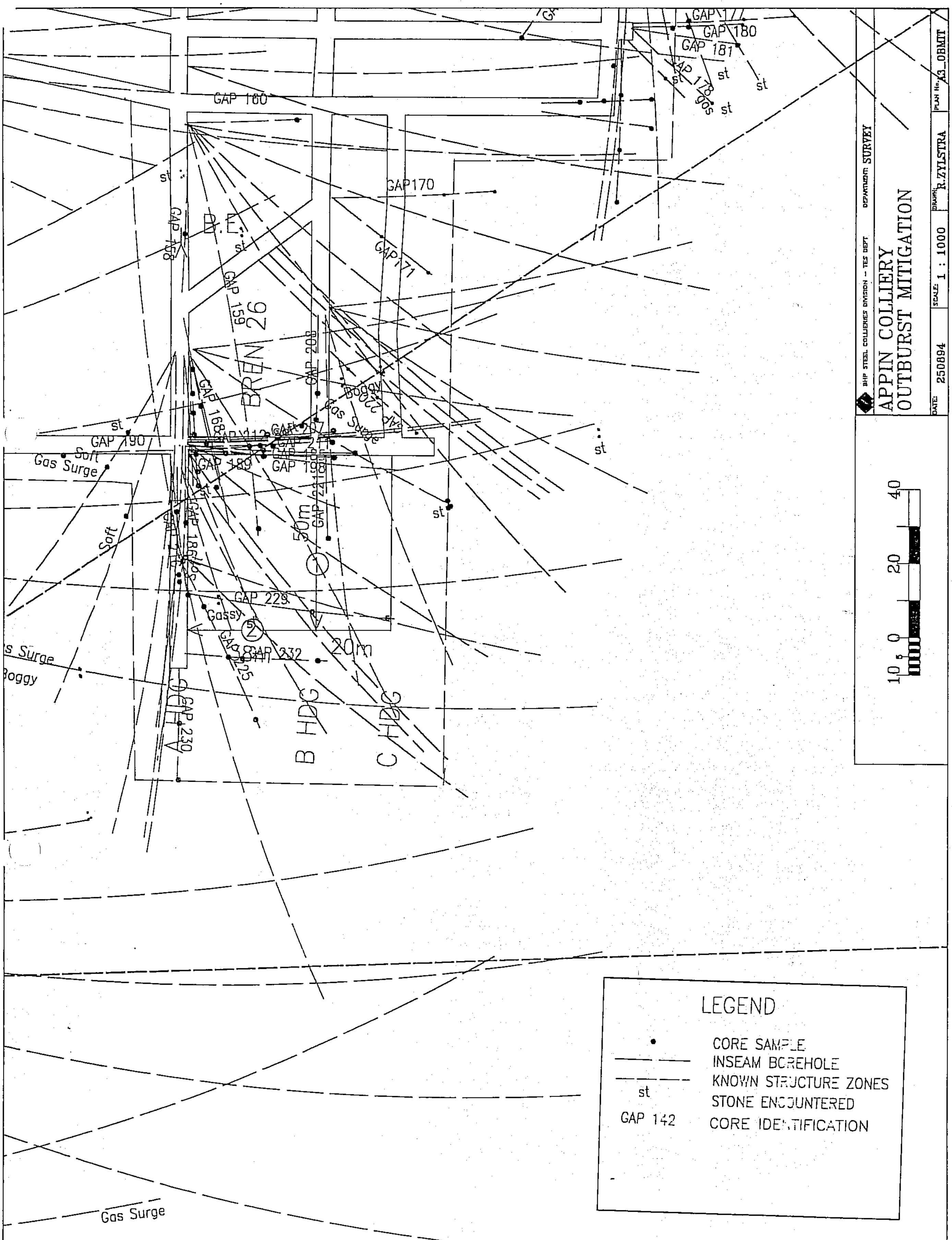

Figure 2. Extreme outburst mitigation.

face area. An Emergency Equipment Station equipped with breathing apparatus is manned in the panel throughout the cutting and bolting cycle to provide immediate response in the event of an outburst. All outburst workers are fully trained in the mining and emergency procedures.

Cutting and loading takes place with only one operator at the face. The shuttle car is driven to the face and the driver then retreats to the Emergency Equipment Station. The miner driver, in a specially designed cabin, cuts and fills the car then bumps the face with the head of the miner intermittently for 5 minutes before radioing for the car driver to return. The shuttle car chain is unable to be flighted during filling therefore the car is only about 3/4 filled.

The bolting cycle is carried out by a maximum of three operators wearing breathing apparatus. The miner driver remains in his cab throughout the bolting cycle to maintain radio communication with the Emergency Equipment Station. A deputys inspection of the face is carried out before bolting commences.

PRODUCTION AND COST IMPOSITIONS

The advance achieved whilst mining under outburst conditions is around one third of that achieved normally due to the inherent delays and inefficiencies in the cycle. Thus, in addition to the safety imperative, there is also a production imperative to remove the gas to enable normal mining to take place.

The mines overriding outburst management philosophy is simple - remove the gas and there cannot be an outburst. Gas drainage operations rarely interfere with production activities but, even after offsetting power generation revenue, account for a significant proportion of total mine costs. Drainage holes are usually drilled 6 - 9 months before mining and generally prove effective in reducing *in situ* gas levels from 10 - 15m^3/t to <3m^3/t.

Drilling for structure location and drilling for gas content/composition cores does interfere with production. To give a high level of confidence in the immediate mining area structure or core locations need to be known with reasonable accuracy and the density of cores needs to be sufficient to provide representative knowledge of potentially variable contents. Cores must also be maintained in advance of mining. This is achieved by frequent short holes being drilled without survey or longer surveyed holes. Either option is time consuming and generally cannot be achieved without loss of development shifts or redeployment to lower priority panels. Core analysis takes around 24 hours which obviously makes the potential production imposition greater as development rates increase.

An extreme case of outburst mitigation and investigation is shown in Figure 2.

CONCLUSIONS

The management of the outburst risk must be driven by the safety of the workers. To increase safety a greater understanding of the phenomenon is necessary combined with increased gas removal and improvements in the quality of data to enable better decision making.

As development rates improve the imposition of mitigation measures and data collection will increase and significant improvements must be sought in both areas to maintain the effectiveness and economic viability of development and gas drainage activities.

Fully remote development is desirable to remove workers from the outburst hazard as well as potentially reducing the extent of mitigation and predictive work.

REFERENCES

Appin Colliery Document QS-ACM-MA001, Outburst Management Plan.

Appin Colliery Document QS-ACM-SP023, Procedure for Mining Under Outburst Conditions.

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THE ROLE OF MIXED GASES (PARTICULARLY CARBON DIOXIDE) ON OUTBURST PROPENSITY

The incidence of outbursting is directly related to the gas parameters present in the coal at the time of the event. The most common and convenient operational parameter measured is the gas content of coal core recovered from boreholes drilled in advance of the developing panel. The measured gas content is related to the gas coal equilibrium by the gas composition, usually determined on the free gas collected during the desorption experiment. Operational decisions on the mining method, and in fact whether an area will be mined, are based on the relationship between representative measurements of the gas content, gas composition, drilling conditions and the geological extrapolation of zones containing outburst prone structures (or free from outburst prone structures) in the vicinity of the current development.

Outburst 'thresholds' are empirically derived, limiting values of the gas content and composition under which no outburst events are historically recorded in a similar mining environment. These 'threshold values are based entirely on measurement, backcalculation, basic observation, and experience. They are not a measure of the safety factor and do not give a quantitative measure of the perceived risk of outbursting.

The determination of the gas content / composition relationship for coal relies on a comprehensive sampling exercise (usually core) ahead of the face. The sampling program is usually accommodated in the mining cycle by utilising non production windows, such as belt moves.

The determination of the final gas content/ composition is a labour intensive process by skilled operators. The final analysis, if conducted under a modified AS3980 method, can rarely be available during a period of less than 18 hours after sampling.

OUTBURST MECHANISMS

An outburst is defined as the violent ejection of coal and gas into the working place. The mechanisms which cause this phenomenon are not well understood. Historical data rarely include quantitative measurements. Usually an estimation of the volume of gas expelled (from an airway measurement) and a measurement of the volume of ejected coal is available.

The outburst mechanism can be simplified to two distinct stages:

- The initiation stage consists of a balance of the material strength (resistance) against the gas pressure and the induced stress field in the coal mass.
- The propagation phase depends on the energy of the free gas constrained in pores and mining induced fractures (volume and pressure), the strain energy accumulated in the system and the destructive energy of a possible shock wave locally modifying the coal strength. The mass displaced, and the distance transported, depends on the size of the particles and the energy of the system.

The effects on the respirable gas mixture depends on the volume of gas expelled, the rate and duration of expulsion, the composition of the gas mixture expelled and the diluting volume of ventilation air. Carbon dioxide is significantly the most dangerous gas, both from the point of view of its toxicity and its difficulty to disperse in the normal ventilation stream.

The coal strength, stress field and strength / stress anisotropy are clearly site dependent variables.

ROLE OF MIXED GAS

The gas associated with the coal is held in a physico-chemical attraction somewhat similar to a 'solid solution'. The amount of gas held at a particular equilibrium pressure is dependent on the partial pressure of each individual gas species and the presence of moisture. The affinity of a particular coal for a particular species can be determined by means of the Langmuir isotherm at the end points (100% gas species). Experimental isotherms take the form shown in Figure 1. Coal has the capacity to adsorb approximately twice the volume of carbon dioxide when compared to methane, at any given pressure.

Experimental isotherms for gas mixtures are not easily determined, particularly at insitu moisture conditions. Isotherms for mixtures of carbon dioxide and methane on dry Bulli seam

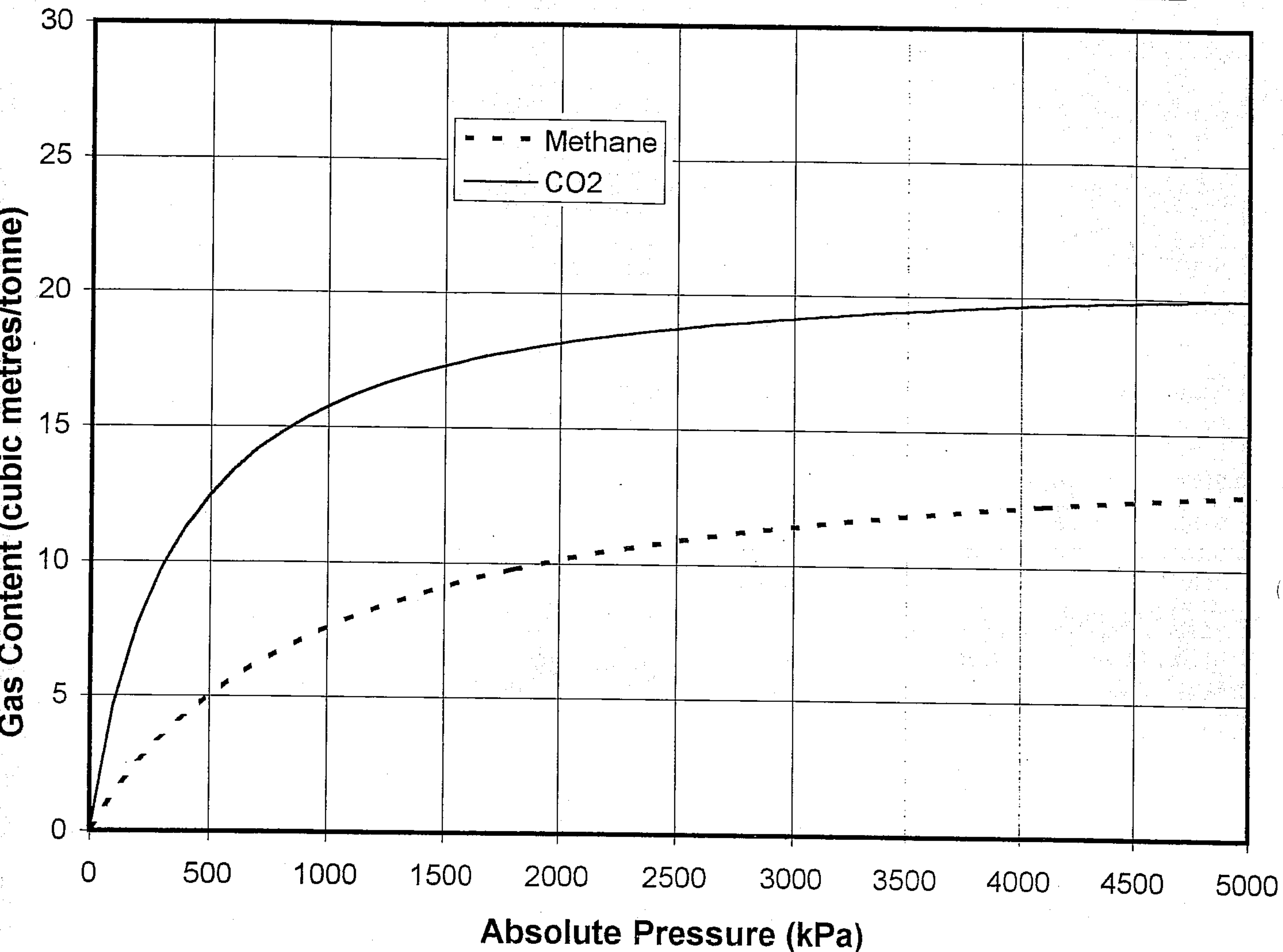

*Figure 1. Bulli coal — **in situ** moisture.*

coal take the form shown on Figure 2 (modified from Stevenson & others, 1992).

Isotherms do not give any indication of the kinetics of the adsorption / desorption phenomenon. Rates of desorption are dependent on the affinity of coal for a particular species, and the ease of removal of the desorbed phase during the desorption process. The ease of removal of the desorbed phase is, in turn, dependent on the permeability of the coal.

The equilibrium composition of gas mixtures in the solid phase relative to the adsorbed phase are reported to be similar at different pressures (Stevenson & others). We must remember that our measurements are from the desorbed phase, when desorbed against atmospheric pressure. Samples of the desorbed gas phase from core, or from drainage boreholes, are not representative of the composition of gas adsorbed in the coal matrix or in the pores. The relative proportion of carbon dioxide in the desorbed phase changes during the desorption process. In general, the ratio of carbon dioxide remains somewhat steady until some 50% of the gas is desorbed and then declines progressively to full desorption (Figure 3).

INTERPRETATION

Results of operational experience, research, and a comprehensive literature search, emphasise the complex site specific nature of outburst propensity, and highlight the lack of quantitative practical measurements useful in the assessment of outburst mechanisms. Gas pressure, and the steepness of the gas pressure gradient, are the common threads of all estimations of outburst propensity. Methods of measurement of these parameters vary from country to country and site to site and therefore make comparative assessment difficult.

The use of existing 'threshold values for the gas content / composition relationship is not conducive to the industry aims of increasing productivity, reducing costs, and do not adequately enable the quantification of risk and so promote safety.

The gas pressure, and gas content, is demonstrated to vary in undrained coal seams, while the reservoir pressure remains relatively constant. The effect is more pronounced when the coal is relatively undersaturated in gas.

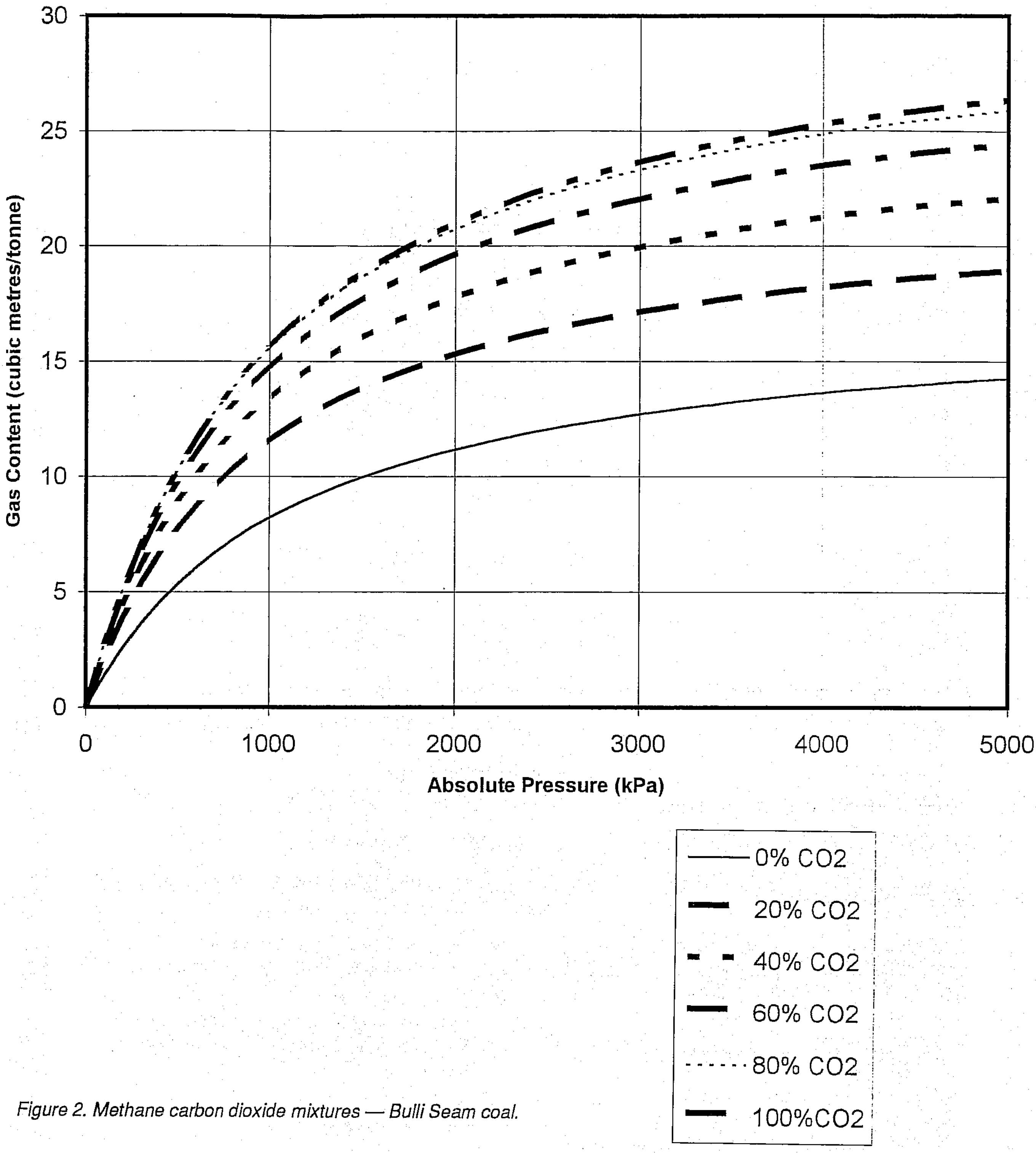

Figure 2. Methane carbon dioxide mixtures — Bulli Seam coal.

This variation can be adequately explained by the following equation.

Reservoir Pressure (virgin) = Gas equilibrium pressure + hydrostatic component

The gas equilibrium pressure is a function of the amount of gas available in the area. Higher gas contents (and gas pressures) associated with some 'structures may be related to secondary gas injected into the system from other seams / sources giving the impression of 'gas pockets.

This mechanism is easier to visualise with carbon dioxide because of its solubility in migrating water and the measured high bicarbonate species in Bulli seam water.

Gas associated with these 'structures' is more difficult to drain because of the anisotropic permeability, two phase permeability effects, presence of dissolved gas in the fluid being drained and the possibility of gas migration into the system.

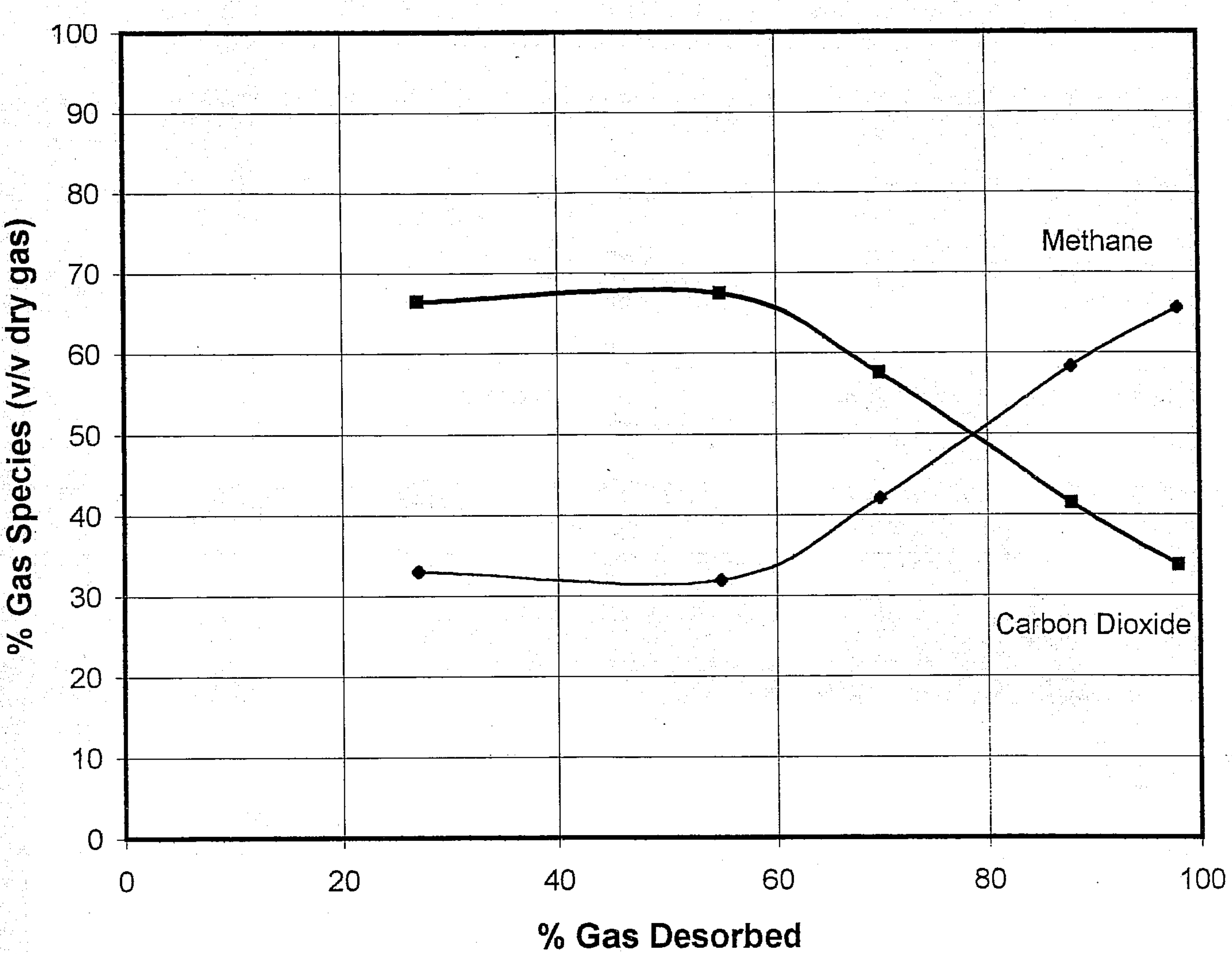

Figure 3. Gas compositional variation from core.

PRIORITY RESEARCH TOPICS

Fundamental Research

- The mechanism of mixed gas adsorption / desorption including the influence of coal type and preferential adsorption sites on the gas content / pressure.
- The mixed gas equilibrium at insitu moisture conditions, the influence of moisture on the competition for preferred sites, and the pressure / content equilibrium in the presence of moisture.
- Identification of the source of migrating gas by isotopic techniques
- The degree of gas saturation under particular conditions. The influence of water pressure on the gas / coal equilibrium and the relationship under conditions of varying permeability.
- Stress/ strain changes during the desorption process. The potential increase in free gas volume during the drainage process due to coal shrinkage.

Applied Research

- Outburst 'structure identification and location prior to mining.
- Sorption pressure measurements and their relationship to reservoir pressure.
- Influence of water (desorption, permeability) in areas of high carbon dioxide or mixtures of carbon dioxide with methane.
- Remedial drainage options in high carbon dioxide environments.

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STEVENSON, M.D., PINCZEWSKI, W.V, SOMERS, M.L. & BAGIO, S.E., 1992: Adsorption of Multicomponent Gas Mixtures on Coal at In-Seam Conditions: submitted to *Society of Petroleum Engineers (Revised SPE23026)* 1992.

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IN-SEAM DRILLING UPDATE

Outbursts can be controlled with proper application of in-seam drilling and successful gas drainage. There are many technical problems which obstruct the proper application of in-seam drilling to outburst detection and prevention.

This paper summarises the main problems faced by in-seam drilling operators and the research supported by ACARP, AMIRA and CMTE to address the problems.

The 1980's attitude to outbursts, ie that control and/or prevention was achieved simply by drilling holes into the seam, are not acceptable in the 90's. There is now an acceptance that in-seam drilling must be controlled to the extent that the trajectory of each hole must be accurately located within a narrow envelope and that the gas pressure or content of coal ahead of the face must be proven to be within acceptable limits before the coal can be mined. These requirements demand greater control of the drill bit and development of technical advances in in-seam drilling than ever before.

The ACA Workshop "Underground Coal Mining Exploration Techniques", 1991 determined that in-seam drilling was ACARP's top priority for underground coal exploration research. The preferred approach for research was to develop a comprehensive scope for an integrated research initiative, driven by industry needs and priorities. A scoping study was conducted in 1993. It defined short and longer term research needs (Hanes, 1993a).

The Exploration Task Force (of ACARP, Underground Subcommittee) designed a collaborative research program which included several complimentary and concurrent research initiatives (Hanes, 1993b). The program will be conducted over a three year period from January, 1994. Total funds of $1 500 000 to $2 000 000 are envisaged during the period. Other research agencies such as the Cooperative Research Centre for Mining Technology and Equipment (CMTE) and ACIRL are collaborating in the overall program.

DRILL SUPERVISOR SURVEY

Drilling operators and supervisors assigned the highest priority need to economically and efficiently locating the drill bit with respect to roof and floor and in the XY plane. Other problems were assigned lower priorities.

Problems facing longhole and rotary drilling are shown on Figures 1 and 2. Table 1 lists the needs for research and development.

DRILL BIT LOCATION IN XY PLANE (PLUS INCLINATION)

The single shot survey camera and the measure-while-drilling (MWD) survey tool are used to locate the bit in the XY plane.

and to assess bit inclination. Surveying with the single shot tool is very time consuming. The only operating "MWD" tool currently approved for use in Australian coal mines is the Dupont tool and its derivatives by AMT (Advanced Mining Technologies). The Drill Scout MWD tool, supplied by Surtron Technologies (WA) is currently undergoing testing for approval for use in coal mines.

For long exploration holes and gas drainage holes drilled parallel to proposed development panels, there is a need for a reliable real time, behind-the-bit monitor which can provide survey data at least at each rod change and

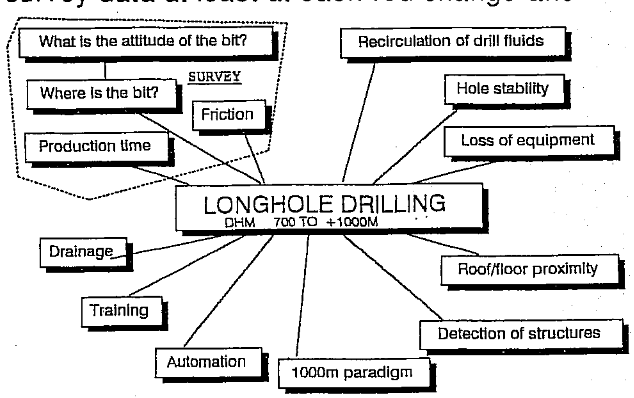

Figure 1. Longhole Drilling Problems.

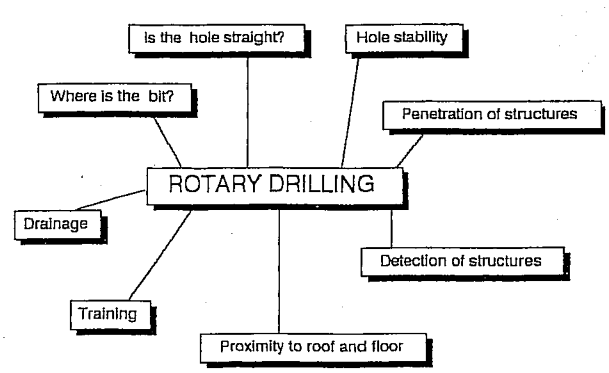

Figure 2. Rotary Drilling Problems.

TABLE 1
IN-SEAM DRILLING R&D NEEDS

SHORTER TERM

a) Inexpensive, reliable and fast survey,

b) Roof/floor proximity sensor,

c) Recognition of geological hazards

d) Maintenance of hole stability

e) Drilling fluid recycling

LONGER TERM

a) Reduced friction for 2000m+,

b) Behind the bit monitoring ,

c) Automated control and monitoring, and

d) Real time computer interpretation/control.

which can transmit data from the planned depth of hole. The above tools should satisfy this need. Therefore, no allowance was made for research for long hole survey tools in the 1994 ACARP round. A comparison of the tools is proposed as an AMIRA co-operative research project in 1994.

Prior to recent times, gas drainage holes and short (50m to 200m) face structure holes were seldom surveyed and their trajectories were estimated based on experience. This practice can fail to danger and is no longer acceptable. These drills with their current down-hole configuration are notorious for producing curved holes.

There is a need for surveying of rotary-drilled across-panel gas drainage holes and face probe holes. The main and minimum need is to know where the holes end. Two push-in tools approved for use are now available. These can be used on the rods for multishot surveying after drilling is completed.

The Peewee distributed by Warajay is a multishot photographic survey tool and Surtron Technologies distribute the CHAMP electronic multishot (magnetic) system. Comparisons of the efficiency of these tools are being conducted by the mining operators on a co-operative basis.

ACARP is funding a trial of in-seam seismic technology for bit location in 1994 (Table 2). Demonstration programs to trial various survey tools for improvement of hole straightness in long holes, and to improve straightness of shorter rotary holes are proposed under syndicated research by AMIRA for 1994.

PROXIMITY OF BIT TO ROOF AND FLOOR

Currently, long exploration holes intentionally intersect roof and/or floor at set intervals (often 50m) to ascertain spot levels. The procedure is time-consuming.

A real-time stone proximity sensor is required at the bit to allow the operator knowledge of where in the seam the bit is located.

AMT is currently developing a roof/floor profiler based on multi-frequency sonic pulsing. The CMTE is considering the application of natural gamma and/or radar to track the bit relative to roof and floor and microdensity for tracking coal seam ash or roof profile relative to the bit.

DRILLING THROUGH UNSTABLE GROUND

Unstable ground, typically associated with geological structures, causes hazards to in-seam drilling including failure to penetrate the zone, erosion behind the bit placing the expensive downhole motor and survey tool at great risk, lack of representivity of any cuttings, and prevention of drainage beyond the cave.

One ACARP project being supervised by Dr Ripu Lama of KCC, "Assessment of Techniques for Maintaining Integrity of Drill Holes For Gas Drainage" will provide an update of suitable methods for achieving this goal (Table 2). A second ACARP project by Dr Ian Gray of AGA (Table 2) is the development of a system for pressurising holes during drilling to maintain hole stability and to provide a suitable environment for use of in-hole logging during drilling.

DETECTION OF STRUCTURES

Detection of structures currently relies on the diligence and attentive observation of the drill operator.

A system of detection of structures in drill holes is required. The caliper tool being developed by ACIRL, (Table 2) offers promise of a short term solution. Geophysical logging is being researched by CMTE and others who are considering applications of natural gamma, radar, seismic or radiometric methods.

A device which monitors changes in drilling parameters associated with structural changes in the seam is required. Trials of BHP's automatically monitored Proram drill to record basic drilling data are being funded by ACARP in 1994 (Table 2). Development by AGA of sensors for bit torque, load and RPM as part of longer term research into monitoring

TABLE 2
CURRENT RESEARCH PROJECTS

ACARP 1993			
1	Maintaining Integrity of Gas Drainage Drill Holes	12 mths	KCC/BHP
2	Optimisation of Long Hole Drilling Equipment	12 mths	SIMTARS (Completed)
ACARP 1994			
1	In-seam drill monitoring and bit location (Stage 1)	12 mths	BHP Research
2	Equipment and technology research for an underground drilling fluid logging system	6 mths	Lunagas
3	Caliper probe for in-seam boreholes	12 mths	ACIRL
4	Borehole pressurisation system	14 mths	AGA
5	Bit torque, load and RPM sensors	9 mths	AGA
7	Co-ordination of in-seam drilling research	24 mths	J. Hanes
8	Supplementary: Specification preparation for common electrical and mechanical elements for implementation of in-hole data acquisition equipment	3 mths	AGA
OTHER			
1	Drill Position Sensing	3 yrs	CMTE
2	Water Jet Drilling	3 yrs	CMTE
3	Survey Technique Trial	12 mths	AMIRA
4	Rotary techniques	12 mths	AMIRA

behind the bit or motor is being funded by ACARP in 1994 (Table 2).

To complement in-hole detection of structures, especially outburst-prone structures, a research project funded by ACARP in 1994 (Table 2) to define equipment and technology to log return drill fluid for its contained gases, was conducted by Lunagas.

RECIRCULATION OF DRILLING FLUIDS

High capacity downhole motors can use in excess of 300 to 400 litres per minute of water. The major problem facing recirculation is adequate removal of abrasive drill cuttings (fines) from the fluid. Research for recirculation has not yet been funded.

DRILLING LONGER HOLES

In 1993, ACARP funded a one year project titled "Optimisation of Long Hole Drilling Equipment" conducted by Dr Ian Gray of SIMTARS. It involved an examination and analysis of existing in-seam drilling records to determine the current physical limitations on drilling. The report's findings are detailed by Gray, 1994 (this workshop). The drill rods currently in use are limited to around 1200 to 1600m depth before yield occurs. Thorough monitoring and recording of drilling parameters are required to enable design of more efficient bottom hole assemblies.

CURRENT RESEARCH

Table 2 lists research approved in the 1993 and 1994 ACARP funding rounds as well as research and development and trials covered by other funding or proposed for co-operative research. The listed projects represent the start of an integrated research program to commence to address the problems of the industry.

DRILLING TECHNOLOGY TRANSFER

Drill operators are forced by production constraints to optimise the equipment and techniques they are familiar with and find it difficult to include experimentation in production drilling. There is an agreed need for testing new drill bits, downhole motors, drilling methods, etc, as well as trials of various techniques for overcoming general drilling problems and communication of results and recommendations to the industry as a whole. There is also some need for replacement of current folklore in drilling with manuals and response curves etc to standardise methods and to improve training response. AMIRA cooperative research programs to support trials of drilling techniques for improvement of drill trajectory are proposed.

The author has instigated regular workshops to bring together the drill supervisors/operators, researchers and suppliers to the industry on a quarterly basis to further improve technology transfer.

CONCLUSIONS

The Coal Industry currently has need of research to help solve shorter term and longer term problems of in-seam drilling. To address these needs, the projects listed in Table 2 are current or planned through integrated research, practical testing and demonstrations funded and/or supported by ACARP, other research agencies, operators and some suppliers.

ACKNOWLEDGEMENTS

This paper was prepared as part of ACARP project C3075, In-seam Drilling Project Co-ordination. The contributions and co-operation of all my colleagues involved in in-seam drilling as operators, suppliers and researchers are greatly appreciated. Their willingness to share knowledge is commendable.

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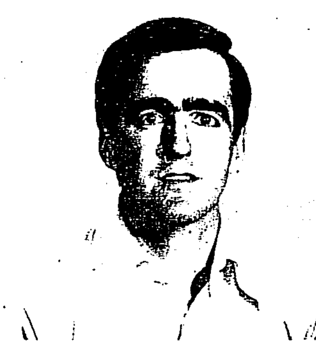


John Hanes

IAN GRAY

Sigra Pty Ltd, a member of the AGA Consortium

DRILLING DIRECTIONALLY CONTROLLED BOREHOLES IN-SEAM TO 2000m AND BEYOND

POTENTIAL

The value of being able to drill very long holes in-seam for exploration and gas drainage is easily recognised. Holes of the same order of length as longwall gate roads will greatly reduce the uncertainties associated with longwall panel development. They also can provide gas drainage protection against outbursts and high gas emissions.

CURRENT STATUS OF LONG HOLE DRILLING

In-seam boreholes are regularly drilled by rotary techniques to 250m. Directionally controlled boreholes are drilled using downhole positive displacement motors powered by the drilling fluid (currently water). These cover the range from 250m to about 800m with some holes reaching 900m and the occasional hole which goes beyond this length. At the time of writing the longest in-seam hole is 1535m, drilled by ACIRL at Westcliff Colliery.

The usual limitations on drilling longer holes are the time constraint of running single shot survey tools up and down the drill rods, and the loss of directional control associated with being unable to set the tool face angle (TFA). In addition rod jamming occurs occasionally and drill rods sometimes break.

The limitations associated with single shot survey tools are likely to be overcome in the near future by intrinsically safe electronic survey tools. These will transmit the survey and tool face angle information up the drill string.

In current drilling the trajectory of the hole is continually changed because of the bent configuration of the bottom hole assembly (BHA) which includes the bit, bent housing, positive displacement motor and survey tool housing. Stabilisers are usually not included because of the risk of becoming jammed in the hole. A typical BHA is shown in Figure 1. Such an assembly does not build angle in exactly the direction it is pointing or at a consistent rate. Examples of some of the variations in build up characteristics using the BHA in Figure 1 are shown in Figure 2. Significant differences are obvious as is the fact that in most orientations the BHA will build downwards.

Current drilling practice involves flopping the tool face angle between approximately 60 and 300° so as to maintain zero vertical build. As a consequence, the hole snakes in a horizontal plane. Vertical corrections are made upwards by pointing the TFA in the 60 to 300° zone and downwards outside of this zone. TFA's are corrected by rotating the drill string in the direction of tool joint tightening. TFA corrections are typically made every 3, 6 or 9m and surveys taken every two to five TFA changes.

Corrections to vertical trajectory are required to keep the drill hole in seam. Present practice effectively requires the hole to run out of seam and then be withdrawn and branched back into the coal to establish the drill bits location in the

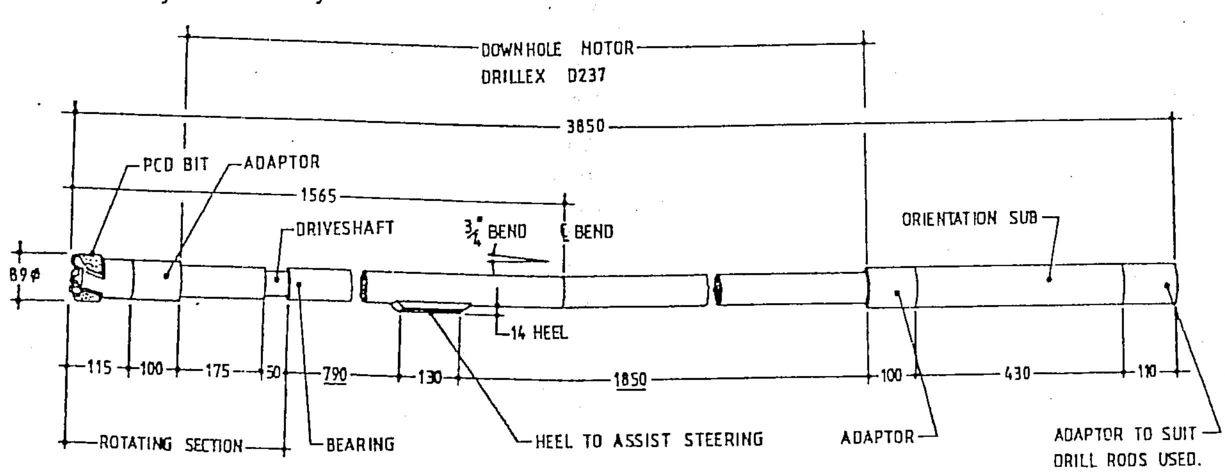
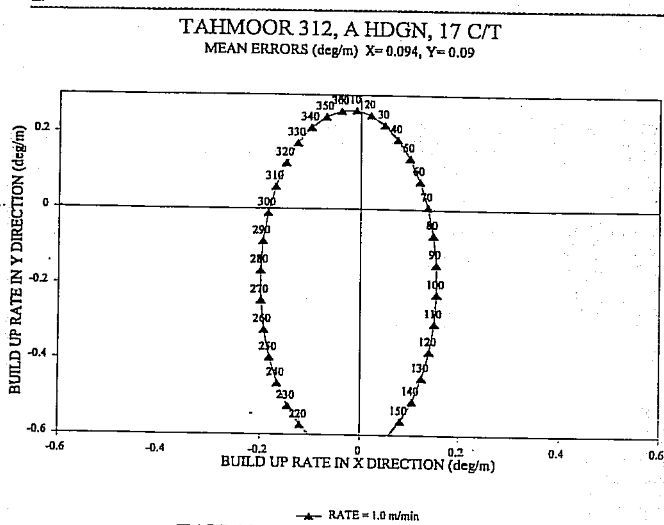


Figure 1. Bottomhole assembly as used at Tahmoor Colliery.



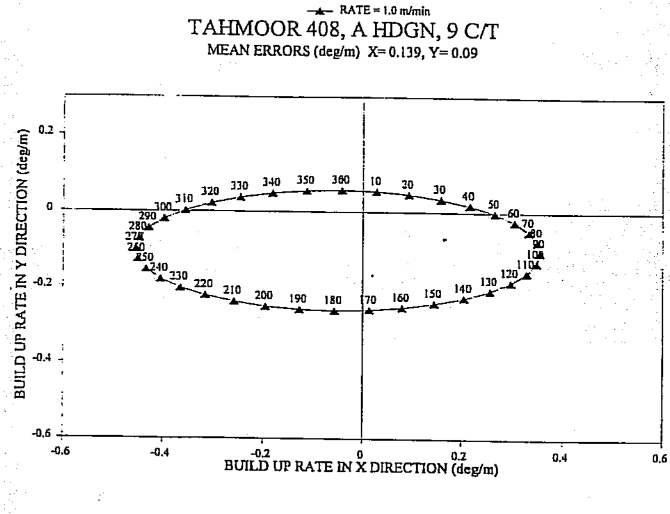


Figure 2. Build up characteristics of BHA in different holes at Tahmoor Colliery. Note drilling rate 0.5m/min.

seam. Sometimes several branches are made before drilling follows the seam. Most generally, an effort is made to drill near the top of the seam and to touch the roof periodically and branch downwards back into the seam. As can be seen from Figure 2 it is easier to build angle downwards than upwards using existing BHA's.

ANALYSIS OF EXISTING DRILLING RESULTS

In an endeavour to predict what distances might be drilled and to establish the techniques and equipment necessary to drill these holes,an analysis of long boreholes that had been drilled at Tahmoor and Westcliff (Northcliff) Collieries was conducted. This analysis was based on borehole survey and drill rig thrust information.

In the first instance the complete borehole trajectory had to be determined from incomplete survey information. This was achieved by developing the deviation vs TFA relations shown in Figure 2 and then using these to find the borehole angles between survey points.

Once the borehole angles could be estimated at each tool face angle change, this information could be used in a torque and drag drilling simulator. Given either bit or drill rig loading (thrust and torque), coefficient of friction between the drill rods and borehole wall, and fluid pressure and flow information this simulator solves the loading situation at any location along the drill string. The loads calculated include axial load, bending moment due to hole curvature or rod buckling, torsion, shear and fluid pressures. Unlike the simulators described in oil industry literature this is a finite element model that takes full account of the stiffness of the drill rods.

Having solved the loading situation the loads are converted in the simulator to stresses in the rod body. As multiple stresses exist they are described as a ratio of the yield state of the rod body material. This is called the von Mises ratio based on the yield criteria of the same name. A von Mises ratio of 1.0 represents the onset of yield, and numbers less than this indicate that yielding has not yet occurred.

In using the simulator to analyse existing drilling results the main unknown being sought is coefficient of friction. In the cases examined the drill thrust was known at various depths. In the case of one borehole the thrust required to slide the rods into the hole as opposed to thrust required to drill was recorded. In the latter case the friction could be readily calculated from the sliding analysis and the drill bit load calculated using that coefficient of friction and drill rig thrust as inputs to the simulator. In the cases where sliding measurements were not taken the bit load was estimated while the borehole was still short and assumed to be proportional to drilling rate thereafter. It is interesting to note that drill bit loads were apparently much greater in the hole NC93-5 drilled at Westcliff than those drilled at Tahmoor (20 to 28.7kN compared to 1.3 to 7.0kN). This does not appear to be a measurement inconsistency but rather a difference due to bit and coal type.

The coefficients of friction derived from this back analysis for five boreholes are shown in Figure 3. The values range from 0.105 to 0.26 with a mean of 0.17 and a sample mean plus two standard deviations of 0.25. What is notable is the consistency of the coefficient of friction value over a borehole length. This is particularly the case for boreholes 500A19, 408A9 and NC 93-5. There appears to be a slight drop in coefficient of friction after the start of drilling. This may be due to some borehole wall polishing. The values of coefficient of friction are low for a water based drilling fluid when compared with published results from the oil industry. This is good from the viewpoint of being able to drill longer holes.

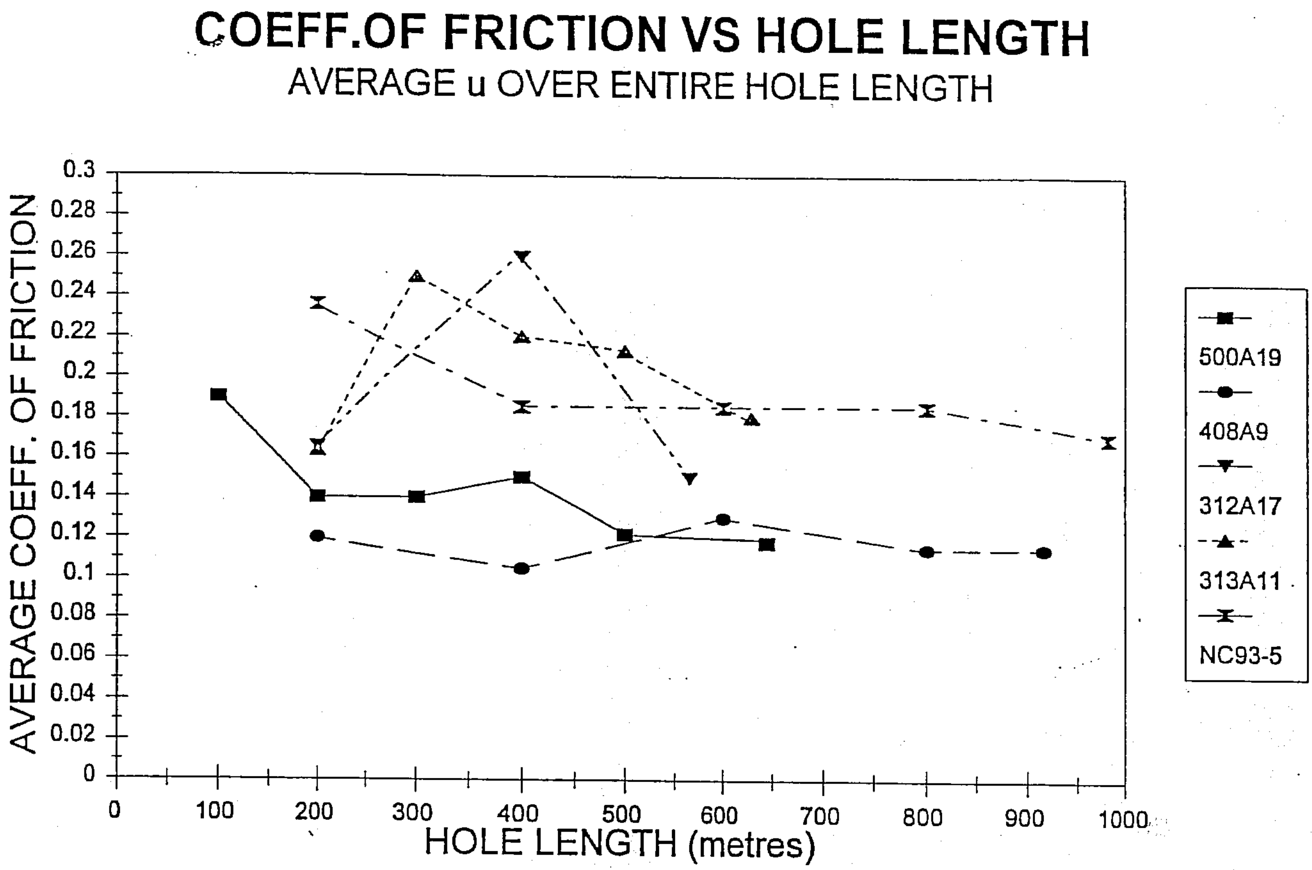


Figure 3. Coefficient of friction determination results.

The drill rod stress values calculated were not so encouraging as three of the five holes showed von Mises ratios for the drill rod body that exceeded 0.3. In four locations this ratio exceeded 0.6 and in one location reached 0.79. When the weakening effect of the rod joint is taken into account, it is quite likely that the rod joint yielded locally. This is not a satisfactory situation. These zones of high stress were all associated with sudden changes of direction, mostly with the creation of branches after the hole ran out of seam.

DRILLING LONGER HOLES USING CURRENT DRILLING PRACTICE

Borehole 312A17 drilled at Tahmoor Colliery was taken as an example of the best current drilling practice as it had no sudden direction changes and as a consequence the lowest rod stresses. A simulated hole with the same profile as 312A17 was created by joining the same survey data end to end with slope compatibility at the junction.

Simulation of drilling with lengths from 200 to 2000m using this profile showed some very interesting trends. These are that the thrust and torque required and the stress that the drill rods are subject to increase exponentially with drilling depth, bit load and coefficient of friction. The case was considered of a Longyear LM 75 drill rig with 135kN of thrust and pullout and 1.77kNm of torque with NQ rods in a borehole of 89mm (3.5") diameter and coefficient of friction of 0.17. The depth limitation was found to be 1700m pushing a bit load of 8.9kN, 1550m pulling the drill string and 1000m while setting the tool face angle. The latter is a drill rig torque limitation. If the make up torque of the rods is not to be exceeded then this limit reduces to 600m.

Two important points are made in this simulation. Firstly that pulling a drill string in a deviated hole requires more force than that needed to push it while drilling. Failing to take account of this may lead to drill string entrapment. Secondly that drill rod joint strength is in question, particularly in torsional loading.

DRILLING LONGER HOLES USING CHANGED TECHNIQUES AND EQUIPMENT

An indication of the limits of length that might be drilled can be gained by the onset of helical buckling in a straight, horizontal hole. This describes the condition when the drill rods form a helix inside the borehole. The consequence of this is that the loading and friction between the borehole wall and drill rod increase sharply. For the case of NQ/NT rods in a 95mm hole the limit is 6km with a coefficient of friction of 0.17

reducing to 4.1km with a coefficient of friction of 0.25. Using HQ/HT rods in a 108mm hole these distances increase to 8.8km and 6km respectively.

In practice the lengths reached before helical buckling occurs are far shorter because of deviations from the straight line of the actual borehole trajectory increase drag. Drilling straighter holes will increase the distance that can be reached. One of the prime enemies of reaching long hole lengths is in fact the BHA's as used in their current form. This is because the angle build capability of currently used BHA's are quite extreme, lying in the range of 0.2 to 0.5°/m in the horizontal plane, up to 0.74°/m downwards and up to 0.26°/m upwards for the cases examined.

Simulations to examine the effects of changing build-up rate and the interval between tool face angle changes were undertaken using the high limit of coefficient of friction (0.25) and NT/NQ rods and HT/HQ drill rods. The conclusion from this is that the loads and stresses decrease dramatically with reduced build-up rates and with extended intervals between tool face angle changes. In the latter case this is despite the larger deviations from the planned trajectory that occur with less frequent angular correction.

Using a BHA with a build-up rate of 0.15°/m or less, and a tool face angle change interval of 18m or greater, will enable current drilling equipment to reach 2000m. The problem is that the torque capacity of currently used drill rods is likely to be exceeded. The use of larger drill rods absolutely requires the use of BHA's with lesser build rates and fewer changes of tool face angle than is current so as to avoid problems with high bending stresses and unacceptable friction. The use of HQ/HT size rods would also require an increase in drill rig capacity of at least 2/3 and shortening of the rods from 3m to permit man handling underground.

FUTURE LONG HOLE DRILLING PRACTICE

Long hole drilling from underground has the potential to go to 2km without great changes to equipment save the improvement of drill rod torsional capacity. What is principally needed is a change in BHA so that straighter holes can be drilled with either fewer TFA changes or with rotation of the entire string. Current BHA's do not permit the latter as the BHA on average builds downwards.

With the adoption of straighter drilling practices the problem of keeping in-seam becomes more difficult as branching will not be as easy. This will force drillers to rely on geophysics in the drill string to detect the location of borehole in the seam. The term coined by the oil industry for such techniques is geo-steering. The presence of such geophysical tools will also assist the location of geological anomalies such as might predispose the seam to outbursts or cause roof stability problems. Likely to be of use for geo-steering are sonic, resistivity, density, gamma and radar probes. Virtually all these tools require a homogenous medium in which to operate. This is not presently available as the hole annulus runs a mixture of water and gas. The current ACARP project C3072 "Borehole Pressurisation System" will endeavour to address this by raising the borehole pressure above sorption pressure, thus preventing the formation of gas bubbles.

Drilling will need to be undertaken with automated monitoring of load and torque at the bit and drill rig so as to permit real time simulation of the loading and stress situation along the drill string. This will help prevent drill string breakage that will become particularly costly with the inclusion of geophysical tools of between $100 000 and $750 000 value.

Drilling beyond 2km is possible but will require a step up in drill string size to reduce buckling problems. It will also require much more gradual change of trajectory than currently used. Rotatable BHA's that drill straight segments will become essential.

CONCLUSION

To drill longer holes there is a need to drill with less deviation. This will require the development of suitable BHA's that will build angle far less severely than those currently used. BHA's will need to be specifically developed for coal as it is a very different medium from most oil or gas reservoirs where existing larger, rotatable, directional BHA's are used. There is also a need to reduce the number of changes of direction and ultimately to drill virtually straight segments by rotating the drill string. The reduced capability to change direction while drilling will require better survey tools and the introduction of geophysical tools to provide guidance on steering. Drill rods with higher joint strengths than the NT/NQ wireline drill rods currently being used will also be needed.

ACKNOWLEDGMENTS

This paper is based on the results of ACARP Project C3023 "Optimisation of Long Hole Drilling Equipment" completed by the author while working with SIMTARS. ACIRL, Westcliff and Tahmoor Collieries who are thanked for providing the drilling information on which this report is based and for permission to present it.

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CSIRO, Centre for Mining Technology and Equipment

GEOPHYSICAL LOGGING FOR THE DETECTION OF STRUCTURES FROM IN-SEAM BOREHOLES

ACARP's in-seam drilling research

Horizontal in-seam drilling is undertaken in many Australian coal mines for gas drainage and exploration purposes. The need to improve in-seam drilling technology was one of the key recommendations of the Australian Coal Association's Underground Coal Mining Exploration Techniques Workshop held in 1991 (ACA, 1991). This led to an in-seam drilling scoping study on the specific problems and requirements of mining operators (Hanes, 1993a, 1993b). The following short terms needs were identified and are now being addressed through a co-operative research strategy involving ACARP, AMIRA and the Centre for Mining Technology and Equipment (CMTE):

- inexpensive and reliable survey data with minimal delays for data acquisition,
- a roof/floor proximity sensor,
- accurate recognition of geological hazards preferably while drilling, or secondly by reliable hole logging after drilling,
- methods for maintaining holes in unstable ground,
- drilling fluid recycling.

Of these, the third relates to the requirement to identify outburst prone structures intersected or in the vicinity of in-seam boreholes. Some of the relevant research which is now underway concerns the development of borehole radar and sonic tools by the CMTE and an in-seam borehole caliper by ACIRL. These are all discussed in this paper.

The CMTE

The CMTE is a co-operative research centre formed in 1991 as an unincorporated joint venture between AMIRA, CSIRO and the University of Queensland. Initial activities mainly concerned metalliferous mining and processing but in 1993, a coal program commenced with significant support from BHP Australia Coal. Funding for the Centre currently amounts to approximately $10 million per annum.

Within the coal program, project CM6 concerns the development of geophysical tools for sensing and logging in in-seam boreholes. The project is developing a roof/floor proximity sensor based on radiometric measurements, and sonic and radar tools for the accurate recognition of geological hazards.

Outburst prone zones

At a meeting of South Coast mine geologists (Technical Group of the Bulli Seam Outburst Committee, 1991) held following the fatal South Bulli outburst, overriding evidence was given that outbursts mostly occur on dykes and strike slip and reverse faults containing a zone of shearing (the 'mylonite' zone).

Given the minor nature of many of these structures, their detection by conventional surface exploration methods (eg drilling and seismic reflection surveying) is difficult. Of the underground exploration methods, RIM (Thomson, this issue) offers some potential. Geophysical logging, if it can be conducted within in-seam boreholes can provide the opportunity to locate structures in or near the boreholes.

SEISMIC LOGGING

The seismic/acoustic approach to these problems targets the changes in elastic properties associated with changes in rock strength and rock type. Observations of seismic waves reflected and refracted at rock interfaces can provide a basis for imaging an interface - the coal seam roof or floor, or perhaps a fault or shear zone in the vicinity of the hole. Measures of the speed of the seismic waves indicate rock strength. A shear zone intersected by a hole would be characterised by lower velocities.

One of the difficulties in developing a seismic tool lies with the logistics of its operation. In particular, it cannot be assumed that borehole fluids will be present to allow acoustic coupling between the transducers and the walls of the hole. Unless fluid coupling can be guaranteed, the seismic source and receivers need to be physically clamped against the walls of the hole. The mechanics of the tool become a major issue.

An interesting possibility is that the noise generated by the drill bit can be used as the seismic source (Rector, 1992). Cross-correlation of the bit noise recorded, say, at the drill collar with that observed at a sensing position on the wall of the hole allows identification of reflections etc from geological interfaces. In this project, we have access to a borehole seismic tool developed by the University of Queensland (Moore & others, 1991). This tool was developed with the prime intention of locating soft shear zones which may be outburst prone. Detailed investigation of the tool's performance and the capabilities were not carried out and we are conducting this evaluation before committing to any further development. At the time of writing, in-seam boreholes were being drilled for test purposes in a highwall at the Swanbank Mine operated by New Hope Collieries near Ipswich.

RADAR LOGGING

When compared with the seismic technique, radar offers significantly improved resolution without the need for physical clamping of antennas. The range of investigation might be reduced to perhaps 5m, but for the detection of outburst prone structures from gas drainage holes, this should not be a major concern.

There are several examples of the use of radar in mining. Yelf & others (1990) conducted a study of underground applications in Australian coal mines. Hainsworth (1994) reports on work at the US Bureau of Mines concerning the measurement of pillar thickness in highwall mining and the detection of roof defects. An example of the use of borehole radar for geological mapping is by Olsson & others (1987). The principles and application of radar to this type of mining problem are therefore well demonstrated, the task we face is one of developing a radar system which is compatible with the requirements and the physical constraints of in-seam drilling.

To date, our efforts have concentrated on designing and building prototype bow-tie antennas with centre frequencies in the range 380 MHz to 1.2 GHz. Design considerations include incorporation of an annulus to allow the flow of drilling fluids in the situation of measurement while drilling. Preliminary trials of these antennas were undertaken in an in-seam drill hole drilled from a highwall at New Hope Collieries Swanbank Mine during August 1994. The hole angled up through a number of stone partings. Figure 1 shows the results from separate scans up and down into the coal seam. Reflections from the partings which continue across the hole are clearly present.

CALIPER LOGGING

The seismic and radar tools have the potential to image outburst prone structures directly. Another possibility for their detection involves

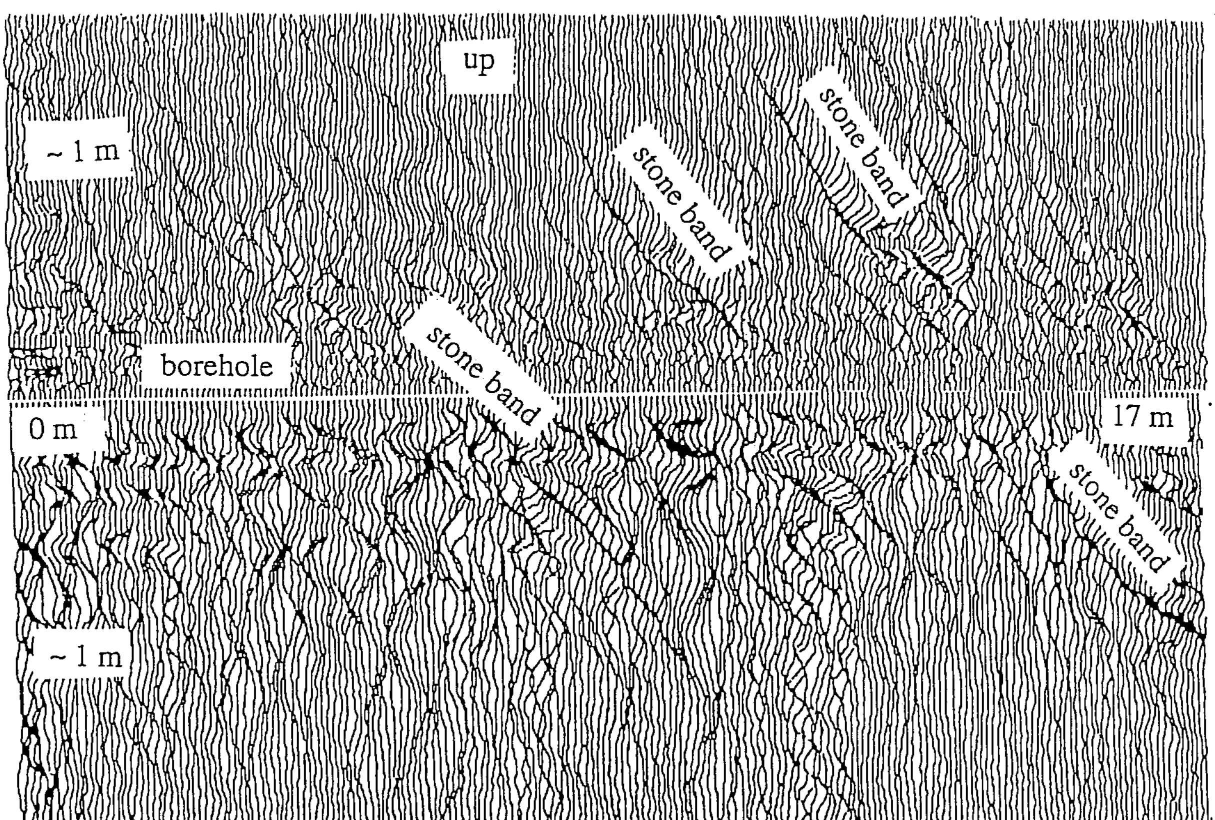

Figure 1. Radar measurements (up and down) in an in-seam borehole.

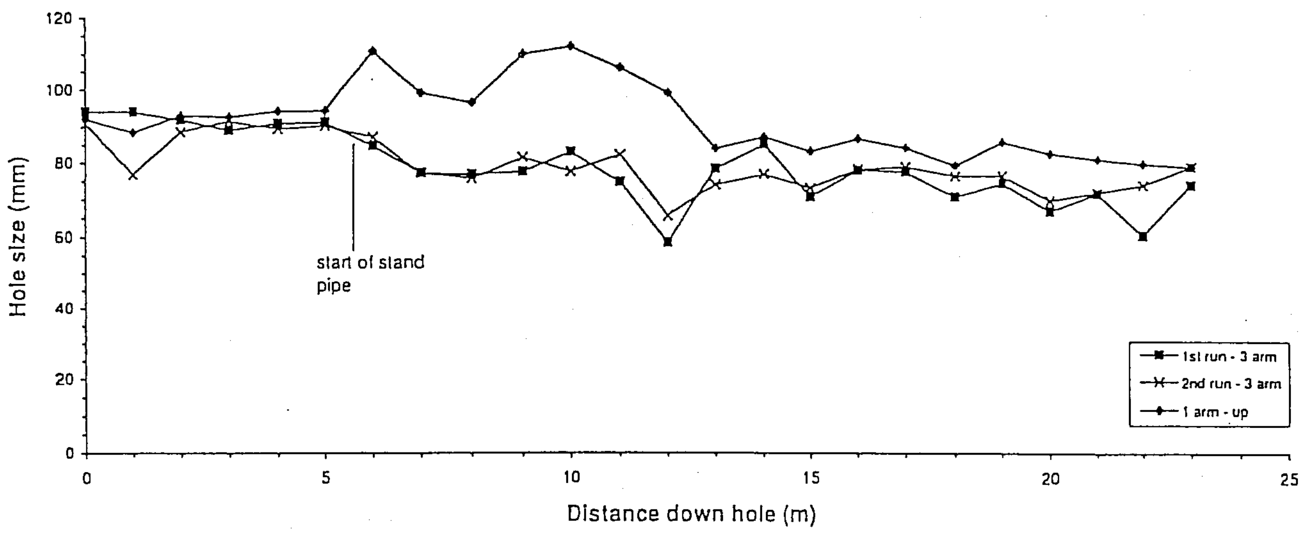

Figure 2. Results of borehole caliper trials showing the difference between hole measurements using 3 locked arms and 1 arm pointing straight up.

the use of a borehole caliper. Any shear zone intersecting the hole can be expected to produce a zone of caving which would be detectable by a caliper.

To test this hypothesis, a prototype caliper tool has been developed by ACIRL and is currently undergoing testing through ACARP funding. The current tool is intrinsically safe and involves three spring loaded arms linked to a vibrating wire strain gauge. The tool is designed to be pumped down drill rods with the bit removed and to lock in place extending from the final rod. A log of hole diameter can then be obtained by removing the rods. Figure 2, shows an example of a test run at West Cliff Colliery.

While the caliper shows great promise, it is not suitable for routine application in its current form with the three arms locked together. A minimum hole diameter rather than the maximum is currently being measured. The arms need to be made independent. Alternatively, the tool could be redesigned so that a single arm can be operated in a vertical orientation.

FUTURE PLANS

The development of the radar and sonic tools will continue through CMTE and possible ACARP funding for the next two to three years. By that time, a production radar tool which can be pumped down the drill string and used to produce a continuous log should be available. Full measurement while drilling will require further effort. Continued development of the seismic tool will depend upon the results of the current series of trials. In the case of the caliper tool, a final prototype should be undergoing testing by the end of 1994.

ACKNOWLEDGMENTS

The work described in this paper represents the efforts of many others. Current work on the sonic sonde is being undertaken by Robert Dixon, Zac Jecny and Michael Barry of CSIRO Exploration and Mining. The radar developments are due to Tony Siggins, Lee Hunt and Greg Turner of CSIRO Exploration and Mining as well as Wayne Murray of CSIRO Applied Physics. The caliper tool was constructed by John Lakeland of Geotechnical Systems Australia. Frank Hungerford of ACIRL and Richard Walsh of KCC were involved in the testing. Caliper results are presented with the permission of ACIRL. New Hope Collieries and West Cliff Colliery are thanked for hosting trials.

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ESTABLISHING AND VALIDATING LOW GAS MINING CONDITIONS

A safety target for mining in potentially outburst prone conditions is that there be no increased risk to personnel. The required level of safety is arguably achievable if the seam gas is reduced to a low enough level.

The main approach to date has been to remove the hazard posed by gas, through the application of in-seam predrainage.

The target for gas drainage in a rapid development mine, is to define and implement a system that achieves consistent and predictable low gas conditions. The high level of certainty required for both safety and high productivity, means that the "drill a hole and forget about it" approach is completely inadequate. It is all about being in control, and this means establishing a workable gas drainage plan and being completely aware of its performance and effect at any point in time - from drilling to mining.

This paper outlines the main technical aspects for a gas management system that has the potential to provide consistent, low gas conditions for development mining, and importantly, validates that these conditions have indeed been established during mining. Aspects of the approach may seem unwarranted, but the target is uncompromising safety, predictability and reliability — day in and day out.

CONTROL AND VALIDATION

Drainage Design

The process begins with gas drainage design - defining a borehole spacing that will meet a target gas content in the time available. In most cases, the drilling and drainage time is dictated by the development/extraction schedule. Normally, this is adequate, but in circumstances where time is too short, the mine schedule needs to take this into account.

Drilling and Hole Completion

While drilling the hole is obviously important, completing the hole (rendering it most capable to drain gas) is of great importance and generally receives inadequate attention.

The main detractors of reliability and predictability in gas drainage are:

- Lack of time or inadequate borehole spacing.
- Boreholes failing to reach target depth and/or location.
- Borehole blockages.
- Holes filling with water.
- Water/debris accumulation in hoses, fittings and pipe ranges.

Bore holes must be drilled to achieve the intended spacing. Borehole wandering using rotary drilling methods can easily result in a targeted 20m spacing being 30m in places. Such a change can more than double the required drainage time to achieve the same effect. Directionally drilled boreholes are required to readily achieve an accuracy of ±3m at 200m depth.

The borehole needs to be thoroughly flushed. Some thing as simple as coal fines building up and clogging at the stand-pipe needs to be avoided.

Outburst coal characteristically has a low permeability and high gas content. Gas drainage in such areas is especially difficult in holes drilled down dip. The gas desorption rate and hence gas velocity is often too low to expel water from the borehole. The water in turn retards gas desorption, the overall effect greatly slowing the rate of gas drainage. As well, outburst zones are usually unstable, presenting problems during drilling, and hole collapse after drilling.

While areas (hopefully very limited) will exist where drilling is too difficult to reach target depth, the approach to drilling and drainage needs to assume that water accumulation - for holes drilled down dip, and hole collapse, are inevitable. As a general procedure, the full length of the borehole should be lined using dewatering tube (if drilled down dip) or screening.

With dewatering tubing, gas pressure is used to force gas and water to flow into the bottom (say 8m) of the tube, by creating a seal between the stand-pipe and the tubing at the borehole collar. Screening (larger diameter tube not used for dewatering) should be configured the same way — to allow gas to only pass up the inbye end of the pipe. *The advantage of this configuration, is that **you know** that gas is flowing over the full length of the borehole. Together with borehole direction, this is one of the most important means of creating the targeted certainty and predictability.*

While the dewatering tube attends to water accumulation in the borehole, the hoses and pipe range need to be maintained free of water accumulation. Hoses need to be hung to prevent water accumulation, and be of a size that maintains gas velocity to either the pipe range or to a water trap. Pipe ranges need to have effective water traps at low areas.

Hole Flow Monitoring

Hole flow measurements are an important part of the process of maintaining the drainage system and defining the course of gas drainage. Every borehole needs to be monitored, either manually, using continuous recorders, or as part of a real time system. Each set of measurements needs to be processed on demand to progressively indicate the uniformity and extent of gas drainage, and to flag problem areas.

Profiles of the total gas drained are an important aid to identifying areas where drainage is less efficient. These are then used as targets for validation by gas content coring.

Software is available that will quickly provide reports on the total gas drained at any point in time, and flag boreholes that behave anomalously.

Gas Content Testing

In-seam coring for gas content measurement is now widely used on the South Coast as a means of defining the seam gassiness prior to mining. It is likely to remain the main method for gas drainage validation for the next three years.

The range of influence of a gas content test cannot be generalised. Variability is primarily created by changes in the inherent gas content and by changes resulting from partial gas drainage.

Trending of past values is a useful a guide to the sample frequency. Mines that do not gas drain, but who need to take cores to ensure mining within safe working limits, experience more uniform values.

For mines that drain the gas, the variability from sample to sample will depend upon the difference between the virgin gas content and the target result, in combination with the drainage time. With short term drainage, there is greater potential for rapid changes in gas levels, from highest between boreholes, to lowest closer to the boreholes. In taking gas content cores, it is important to core in the area of highest gas content — where the hole spacing is at its widest, or where the lower gas flows were identified during gas drainage.

Informed decisions on gas content sample locations, can be made if gas drainage boreholes are both surveyed, and monitored for gas flow. For mines that gas drain, hole position and gas flow criteria should dictate sample location, rather than blind adherence to a fixed sampling pattern.

Real Time Monitoring

Taking cores for gas content testing can be disruptive to an advancing panel, where drilling and mining equipment compete for the same space. Either the frequency of the disruption increases with the rate of panel advance, or cores are taken at greater depths, to cover the next cycle. Apart from this inconvenience, there is some uncertainty surrounding the interpolation of gas content values between sample points.

Real time return gas emission monitoring has the potential to quantify the face gassiness during mining, and provide continuous sampling of all cut coal with complete transparency to mining. It's prime role in a rapid development scenario, is seen as validating during mining, the low gas conditions established by gas drainage.

It is relevant to mention ACARP project 3076, "Real time return gas monitoring for outburst and gas drainage assessment" being conducted through GeoGAS. The project has a duration of three years, and commenced in January 1994.

The prime objective is to create a turn key, prototype, stand alone, real time, gas monitoring system capable of providing quantitative assessments of gassiness levels on at least a shift by shift basis and enabling unusual gas emission patterns to be readily flagged.

Sub objectives are to:

- To define and document the process of using this data to back analyse gas drainage effectiveness.
- To assess the potential for using the technique to quantitatively define seam gassiness on a sub shift period basis.
- To define indices relating the gas emission response (rate of emission, peak emission rate, quantity, composition) to outburst proneness in terms significant to outbursting.

The system involves real time monitoring of the return gas emission during mining using high accuracy CO_2 and CH_4 gas analysers and an air velocity meter. Coal production is measured using a belt weigher.

The system is being assembled and is due to be installed at West Cliff Colliery in late September 1994.

RESEARCH REQUIREMENTS

Proceeding on the basis of assumed consensus, that greatly increased predictability and reliability is required in gas drainage, and that the approach outlined is reasonable, it remains to define the impediments to implementation.

Performing these activities to a large extent requires little more than an operational approach and commitment. Some areas are easier to implement than others and this is where research is needed.

The status of areas requiring research (to the author at least) are:

» Directional drilling

The hardware exists and is being used to accurately drill boreholes to a pre-determined pattern. Drilling to target depth in poor ground conditions is probably the main area requiring attention.

Ultra long hole drilling (>500m) makes sense from a drilling economics point of view, but is seen as being at the expense of the level of predictability and certainty achieved from shorter (<300m) directionally drilled boreholes.

» Hole flow measurement

Manual measurements are tedious, and are one of the main reasons why routine gas flow measurement is not universally carried out. There is scope to significantly improve the methods of manual flow measurement.

Real time monitoring of gas flows would be the best solution. The task is considerable, with monitoring of, say over 100 boreholes for flow pressure, static pressure, purity and gas composition - all in a harsh environment.

Until something better comes along, manual measurements are well worth doing.

» Screening and Dewatering

Unless CO_2 concentrations are very high (85%) or there is little or no vacuum, the material used for dewatering needs to be fire resistant and anti-static. It is very expensive (say \$8 to \$22 per metre). What is the real propensity for static charge build up on PVC or polyethylene in a borehole? How do explosibility limits change under vacuum? The industry stands to save a lot of money from research in this area.

» Real Time Return Gas Monitoring

Real time monitoring of gas emission is being addressed through projects to BHP/KCC and GeoGAS.

CONCLUSIONS

To realise a consistent high level of gas drainage reliability and predictability, all the factors mentioned in this paper need to work together — drill targeted gas drainage boreholes, establish optimum conditions for gas drainage, chart the course and uniformity of gas drainage using hole flow measurements, validate the effect by targeting areas for gas content coring, followed up by real time, return gas monitoring.

Different mines have different degrees of risk, requiring a different mix of measures to be adopted. A mine with $15m^3/t$ CO_2 is a different proposition to a colliery with say $7m^3/t$ CO_2 as its virgin gas content. If you know the gas content is not going to rise beyond say $7m^3/t$, perhaps a completely different approach is warranted — concentration on structure defining techniques (at this stage, only RIM has the potential to do this to the level of certainty required, but it needs to be proven).

Either way, data redundancy is required to ensure that safety targets are achieved. For example, the level of certainty is greatly enhanced if say gas content test results are corroborated by real time return gas emission results.

The secret to introducing an approach as outlined, is to build it in to a system that delivers the safety, reliability and predictability — day in and day out. Knowing the direction you have to go, it remains to make sure that impediments to making the system work are over come — and this is where focused research can help.

Ray Williams

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PREDICTION AND CONTROL OF GASSY CONDITIONS FOR UNDERGROUND ROADWAYS

Roadway development systems and achievements are limited for those underground mines where gas hazards require utilisation of special technology due to restrictions in advance rates and ventilation requirements. Typical Australian underground mines utilise multi-headings for the 'first workings' and double and/or combination double and single entry headings for longwall block extraction practises. Currently, an average of 5 to 12m per shift continuous miner advance rates is achievable in gassy mines, with a maximum of 18m per shift when an efficient predrainage system is used.

For example, in selected gassy mines, continuous miner advance rate requirements for longwall continuity are 5 to 40m per shift (Table 1), and in some cases are not achievable due to hazardous gassy and outburst prone conditions.

For planning purposes, optimum advance rates can be predicted for various gassy and mining conditions, using the "Lunagas prediction simulation program".

UNDERGROUND ROADWAY GASSINESS PHENOMENA

Gas emissions to the driven underground roadways, when using continuous miners, depend substantially on the factors listed below:

- *in situ* gas content and composition,
- predrainage efficiency,
- heading geometry (width and height),
- continuous miner advance rate,
- number of continuous miners in panel,
- heading (panel) drivage length,
- duration-time factor ("T" coefficient),
- sum of coal edges in heading perimeter,
- rib, floor and/or roof gas emission factor ("E" coefficient),
- heading face coal output and coal cutting time
- panel geometry including pillar width and length (number of cut-throughs),
- local geological and mining conditions,
- ventilation system.

Most of these factors can be established or calculated for each heading and/or panel, however, the "E" and "T" coefficients represent specific parameters which depend on local gassy and mining conditions.

LUNAGAS MODEL FOR UNDERGROUND ROADWAY GAS MAKE PREDICTION

The method of gassiness prediction for nominated underground roadways in Australia has been based on long-term research and empirical results regarding intensity of gas release per square metre of exposed coal ("E" coefficient), and decay of gas release with time ("T" coefficient).

Table 1. Continuous miner development rates for selected gassy mines.

Mine - Longwall No	A - I	A - II	C - I	C - II	T - I	T - II
	Continuous miner advance rate (metres per shift)					
Current Development Rate	15.0	17.6	13.3	5.8	12.0	6.0
Required Development Rate	19.2	17.6	38.7	19.6	12.0	5.3

The total quantity of gas release to the underground roadways include:

- emission from freshly cut face coal and,
- emission from the ribs, roof and/or floor gas sources

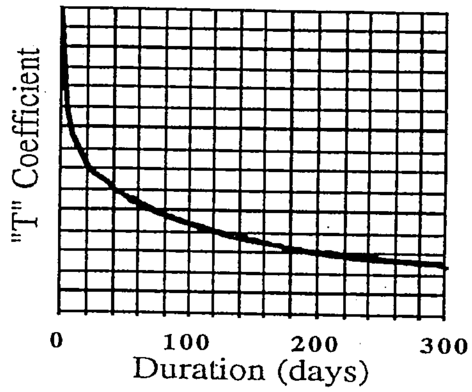

Figure 1. Intensity of gas release - "E" coefficient.

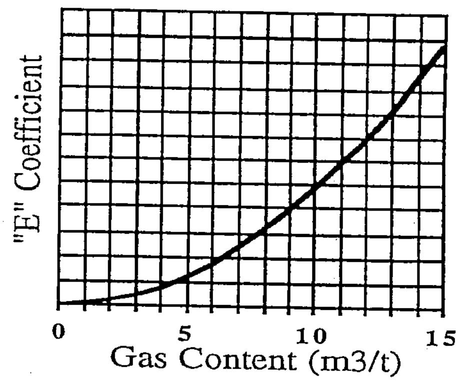

Figure 2. Gas release decay with time - "T" coefficient.

The Lunagas mathematical model incorporates all thirteen above mentioned factors as well as mine intake contamination and ventilation irregularity coefficients.

The intensity of gas release – rib gas emission coefficient (Figure 1), has been developed using Australian roadway gas balance results and typical advance rates for coal seams with virgin and remaining *in situ* gas contents of 1.0 to 15.0 cubic metres of gas per tonne of coal. Similar mining and gassy conditions have been used to establish a coefficient for gas release decay with time (Figure 2).

The "Lunagas simulation program" outputs include predicted gas makes and ventilation requirements for the heading intake and return sides (Table 2).

- gas make from cut coal,
- intake and return emissions,
- total emission (gas make),
- intake ventilation requirements,
- return ventilation requirements,

Figure 3 shows an underground workings gassiness prediction flow chart and outlines the logical steps of inputting and utilising various factors effecting the final output figures.

Figure 4 shows a graphical illustration of gas make prediction figures for a panel length of 250 to 1250m for a continuous miner advance rate of 12m per shift, various in situ gas contents and intensities of gas release ("E" coefficients).

CONTROL OF GASSY CONDITIONS AHEAD OF THE HEADING FACE.

There is a constant need for a reliable, accurate and fast method of determining gas and gas-rock outburst prone conditions ahead of the development face. The methods used to date do not cover all safety aspects and provide only limited information on the problem.

An Underground Drilling Fluid Logging System can be used for these purposes, and represents a complementary method for

Table 2. Lunagas simulation program outputs for an advance rate of 12m per shift.

Length Panel Drivage metres		**Predicted gas make** Cut coal Intake Return Total litres CH_4 per second				Ventilation Requirements Intake Return m^3 air per second	
200	505	35	28	30	93	11	9
400	975	35	34	45	114	14	11
600	1444	35	38	54	127	15	13
800	1914	35	41	60	136	16	14

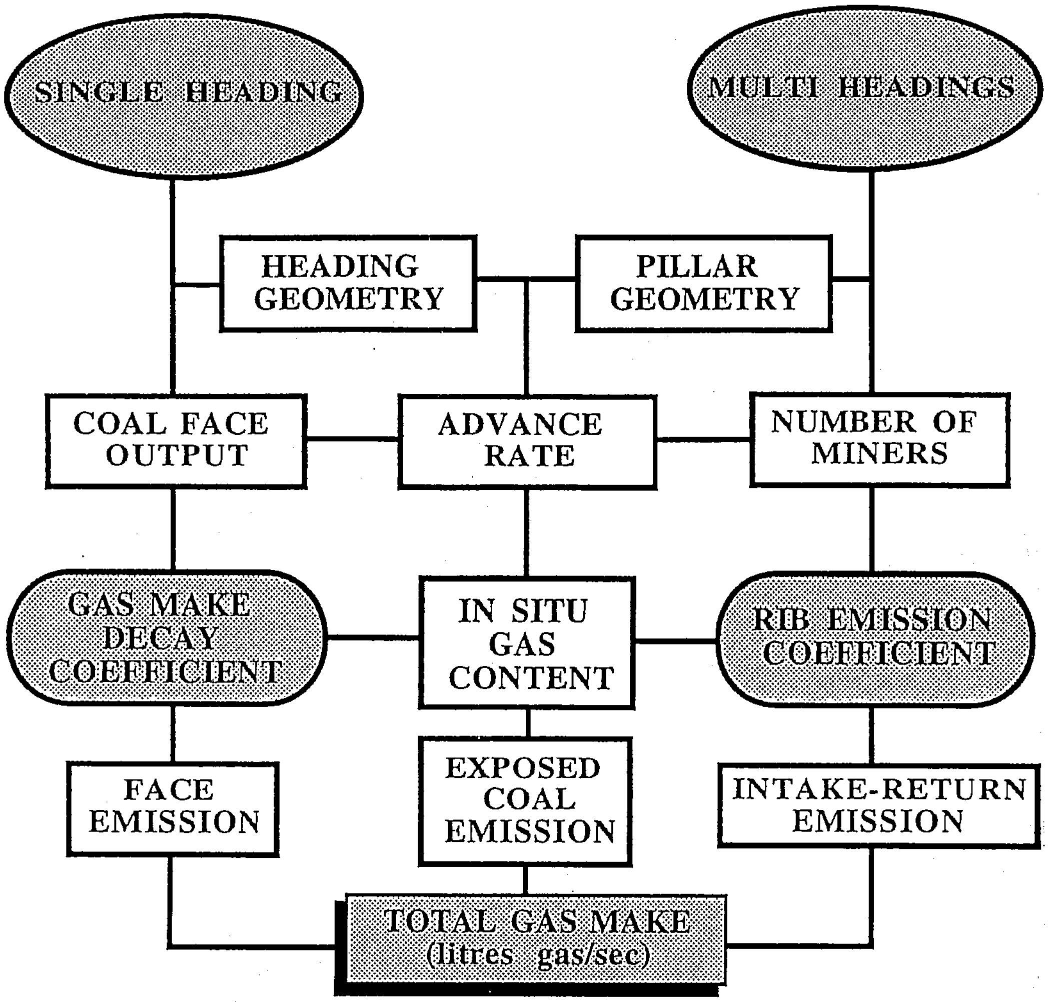

Figure 3. Underground workings gassiness prediction flow-chart

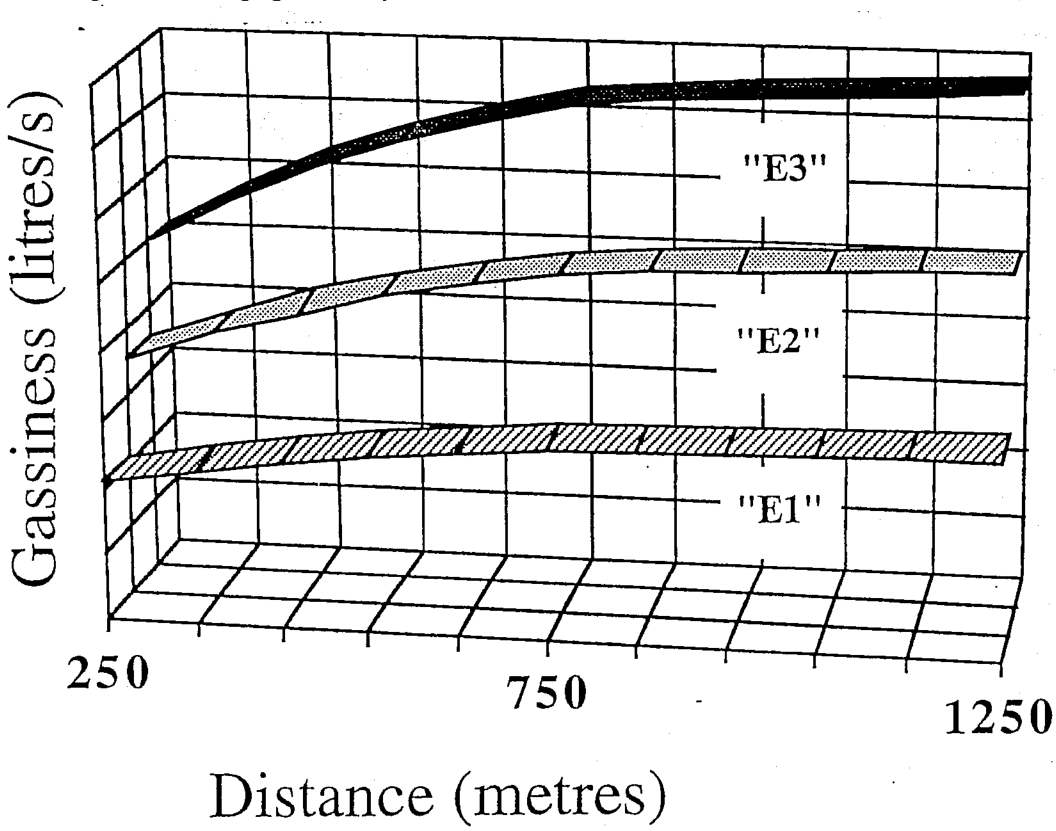

Figure 4. Predicted gassiness for various "E" coefficients.

recognition of gassy conditions ahead of a driven heading face.

Drilling fluid contains a certain quantity and quality of gas. This data can be transferred and used by the system to obtain information about both the gas content and permeability of a coal seam along a drilled inseam hole.

A similar technique, Mud Logging Technology, has been used in the oil related industry – vertical surface boreholes, with a great deal of success. The underground version of the system promises to deliver valuable data, however, alterations to the technology and equipment are necessary due to extreme underground conditions and statutory mining requirements.

Lunagas Pty Limited has been involved in the recognition and evaluation of technology and equipment currently available for the system to be designed and introduced to Australian underground coal mine conditions. Details of the selected equipment and proposed technology are presented in ACARP Project No. C3077.

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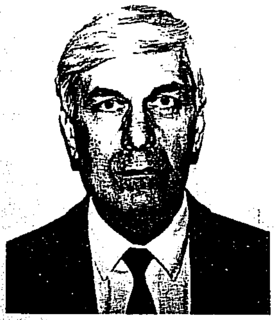

Les Lunarzewski

SCOTT THOMSON & MARK NEIL

METS Pty. Ltd.

THE APPLICATION OF THE RADIO IMAGING METHOD TO OUTBURST HAZARD MINIMISATION

Radio waves are differentially attenuated in rocks according to the electrical conductivity of the medium between transmitter and receiver. Coal is a particularly resistive rock type and provides a natural waveguide for the passage of radio waves between the confining layers of conductive roof and floor material. In areas of homogeneous coal radio waves are attenuated at a predictable, repeatable rate and variations to the norm can inevitably be attributed to changing geological conditions. Seam thinning, faulting, mylonite zones, igneous intrusions, and palaeochannels all increase the rate of signal decay and are therefore able to be detected remotely by radio wave analysis.

Radio imaging is best known in Australia as RIM (the Radio Imaging Method). RIM has been applied worldwide to over 130 longwalls for the delineation of geological hazards. The technique has proved to be the most sensitive high resolution geophysical tool currently available to longwall mine planners and recent technical developments have enabled the method to be applied with confidence ahead of the development face.

Recent work in the NSW Southern Coalfields has established the potential of the technique as an outburst hazard detection tool. The method can be integrated with drilling and gas content sampling to provide a high confidence predictive methodology that minimises the risk of development roadways intersecting an unexpected potentially lethal outburst.

CURRENT STATUS OF OUTBURST HAZARD ASSESSMENT

The greatest single difficulty in assessing the risk of dangerous outbursts is in defining with near absolute certainty the presence or absence of outburst prone geological structures and their likely magnitude.

Exploratory boreholes routinely drilled in advance of development provide important structural information but it has been clearly demonstrated that such boreholes do not provide the required level of certainty for mine operations. Unexpected, dangerous outbursts have occurred in spite of in-seam drilling. This is due to a combination of the very limited area of influence of drilling, and in the subjective nature of the interpretations.

Geophysical exploration methods such as in-seam and surface reflection seismic provide regional information which is lacking in the detail required to greatly assist the outburst detection process.

Because of this, recent developments in assessing outburst prone coal place major reliance on gas content testing, with the tendency to fix limits for mining that reflect the inability to be certain that outburst prone geological structures are absent. In many cases the uncertainty implies that too much caution is exercised and mining is slowed unnecessarily. Too little caution puts personnel unacceptably at risk.

Other issues affecting the confidence of prediction of outbursts include the uncertainty of gas drainage hole direction, and the representative nature of spot gas content samples. Gas drainage holes are rarely surveyed, and it is therefore never certain exactly what areas are being drained (or whether holes are draining from the full inbye extent of the borehole). In addition, gas samples from poorly draining areas have been found to exhibit a wide range of gas content values between holes.

The objective then should be to achieve a predictive methodology which establishes with near absolute certainty that outbursts will or will not occur during development drivage. It is suggested that radio imaging can play an important role in the process because RIM is:

- sensitive to relatively minor geological structures,
- gives near absolute certainty in assigning an area to be free of geological structures,
- takes measurements that are non-subjective,
- gives advance coverage of the total area to be mined (as opposed to a single borehole), and
- can fit in with the operational cycle.

The compelling empirical evidence to support the technique's potential role in outburst detection suggests that integration into the mining cycle should be a high priority for those mines operating in areas of significant outburst risk. Currently, trial survey programs are ongoing or planned at the majority of NSW South Coast underground longwall operations.

AN OVERVIEW OF RIM IN AUSTRALIAN COAL MINING

RIM has been utilised at most Australian longwalling operations over the past four years on an 'as needed' basis. In that time it has been used mainly to investigate known structure, and to answer specific questions such as:

- what is the magnitude of the structure?
- what is the trend of the structure?
- does the fault in the maingate link up with the one in the tailgate?

In only rare cases have mines used RIM as part of a risk reduction strategy, geared to establishing clear ground as well as learning more about known structure. In the investigation of outbursts, it is an important distinction to make. RIM is not only important for its capacity to identify outburst hazards, but also for its ability to *establish clear coal.*

RIM has consistently proved its sensitivity to geological structures using loop antenna systems from roadway to roadway in applications associated with existing longwall panels. Results have always been predictable and reliable in those coal seams where bulk seam conductivity is not too high and cultural effects associated with mine conductors have been minimised.

RIM Generation I (RIM I), as most longwall operators are familiar with, is currently being replaced by improved in-mine equipment. RIM I has helped establish the technology in operational mines but frequency inflexibility has limited its application in conductive coals (particularly in the Bowen Basin) and reduced its effectiveness in some cases. The next generation of RIM instrumentation (RIM 20/20)

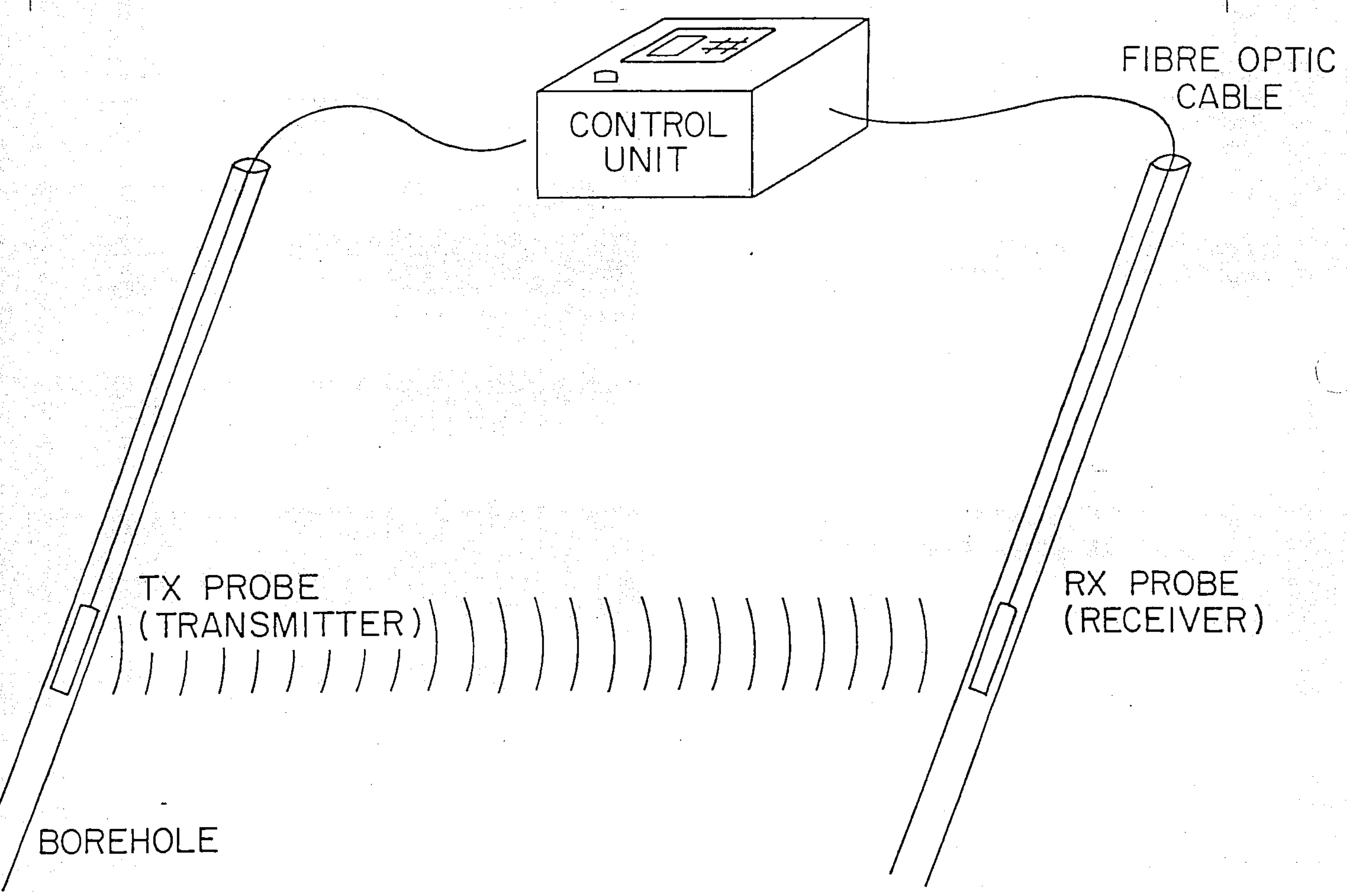

Figure 1. RIM II borehole to borehole configuration.

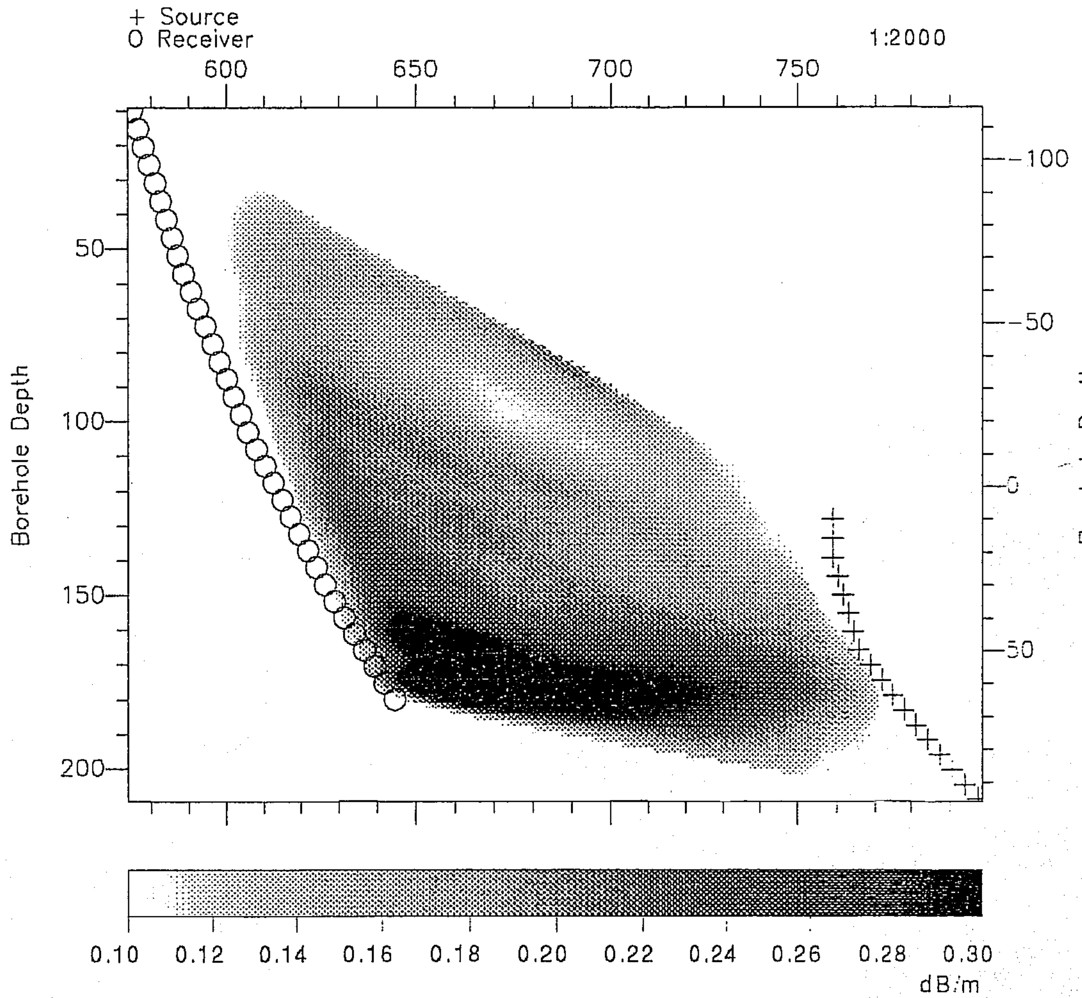

Figure 2. Image showing low-risk zone and zone affected by mylonite structure.

will increase the sophistication of RIM surveying and analysis by offering multi-frequency analysis and result in an improved delivered product to the mine.

The borehole equivalent, RIM Generation II (RIM II), when applied in horizontal holes (Figure 1) offers the opportunity to get ahead of the face and establish a geological model prior to the commitment to costly development. However, it requires open boreholes and these are only readily available in mines with gas drainage programs in place. It has been applied most successfully on the NSW South Coast, although hole caving has limited its application in some mines.

Efforts to establish the sensitivity of RIM ahead of the face from borehole to borehole have until recently proved unrewarding due to the logistical difficulties of inserting a probe reasonable distances up a horizontal borehole. Working ahead of the face offers technical benefits currently not available in roadway to roadway RIM. Firstly, borehole RIM can be undertaken at many frequencies, thereby ensuring the range and resolution is appropriate for the job in hand. Secondly, transmitter and receiver probes are isolated from mine cultural effects, so the results are free of 'noise'.

METS have developed a downhole insertion methodology that has enabled cross-borehole measurements up to 300m ahead of the face. The method is time consuming and manually difficult but practically effective. Typically, cross-hole RIM has utilised boreholes spaced 200m apart although closer spacings are feasible with higher radio wave frequencies, and likely to result in improved resolution.

Recent commercial surveys at Appin Colliery have provided promising results but more work is required to establish the confidence in the methodology and to ascertain the technique's rightful place in the production cycle. Aspects of logistics require attention. Areas where drainage has been ineffective have been highlighted and further work is required to quantify these findings.

RESULTS FROM APPIN COLLIERY

A series of cross-borehole RIM measurements were made in areas where potential outburst

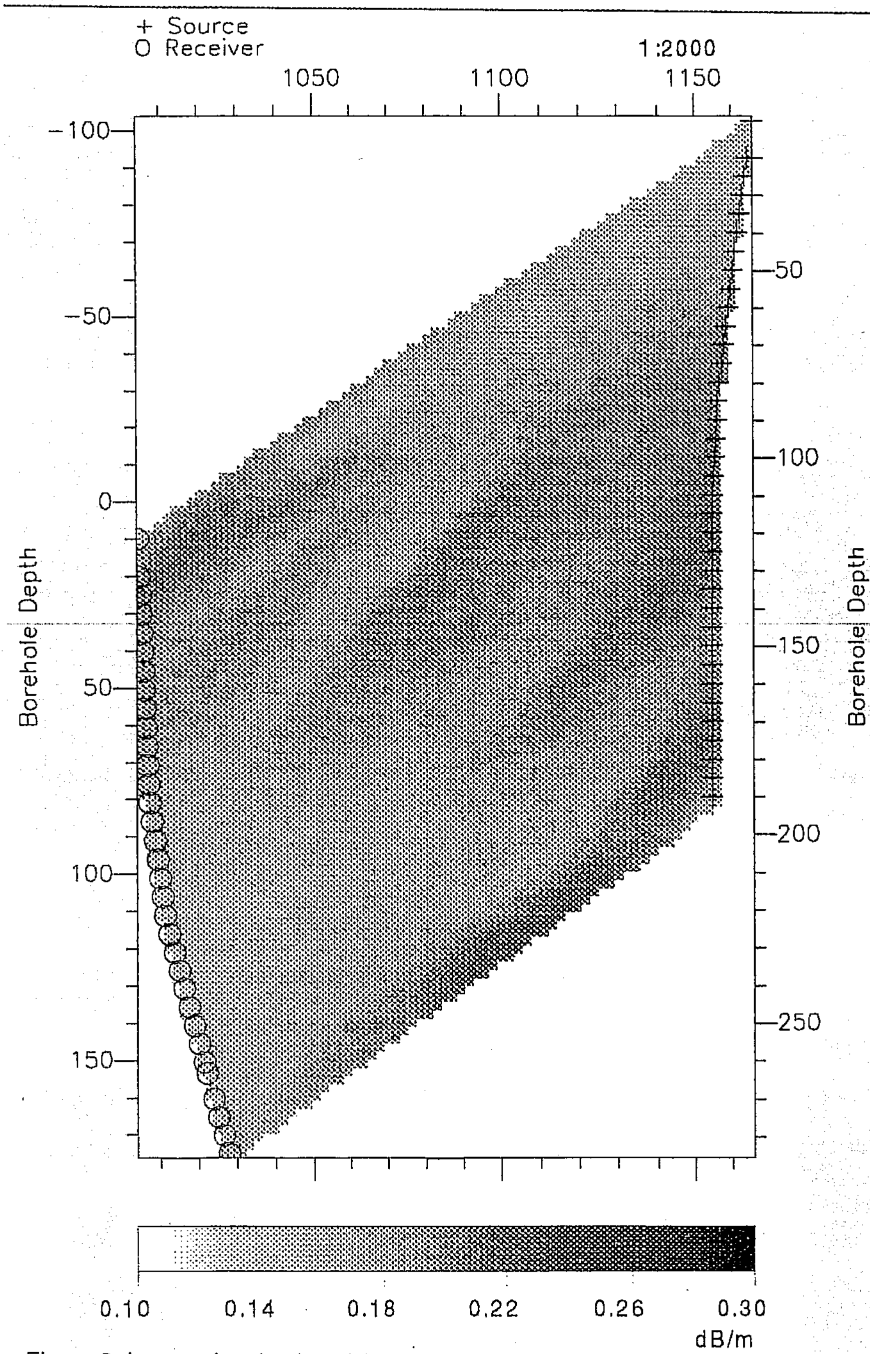

Figure 3. Image showing low-risk zone (clear coal).

structures were expected, and in other areas where no hazards were known to exist.

The RIM probes were inserted in gas drainage boreholes, usually up to 200m inbye. Cross-hole tomographic procedures were used to produce images of radio wave attenuation over the interval in question. The frequency employed was 300 kHz and cross-borehole spacing varied from 140-250m. Holes were surveyed using an Eastman single shot camera.

In Figure 2 a cross-hole RIM image shows an area of high attenuation inbye which can be directly related to a known outburst prone strike-slip fault structure. The outbye zone represents an area of low attenuation and minimal risk.

Using a constant attenuation scale, Figure 3 illustrates a zone which is hazard free. The consistently low attenuation rates, suggest the area has been effectively drained and is not affected by geological structures.

In contrast, Figure 4 is from an immediately adjacent area (note common boreholes) and clearly shows high rates of signal attenuation. This can be attributed to the presence of a potentially outburst prone strike-slip fault orientated sub-parallel to the boreholes. Later investigation revealed a high number of blocked gas drainage boreholes in the area. It

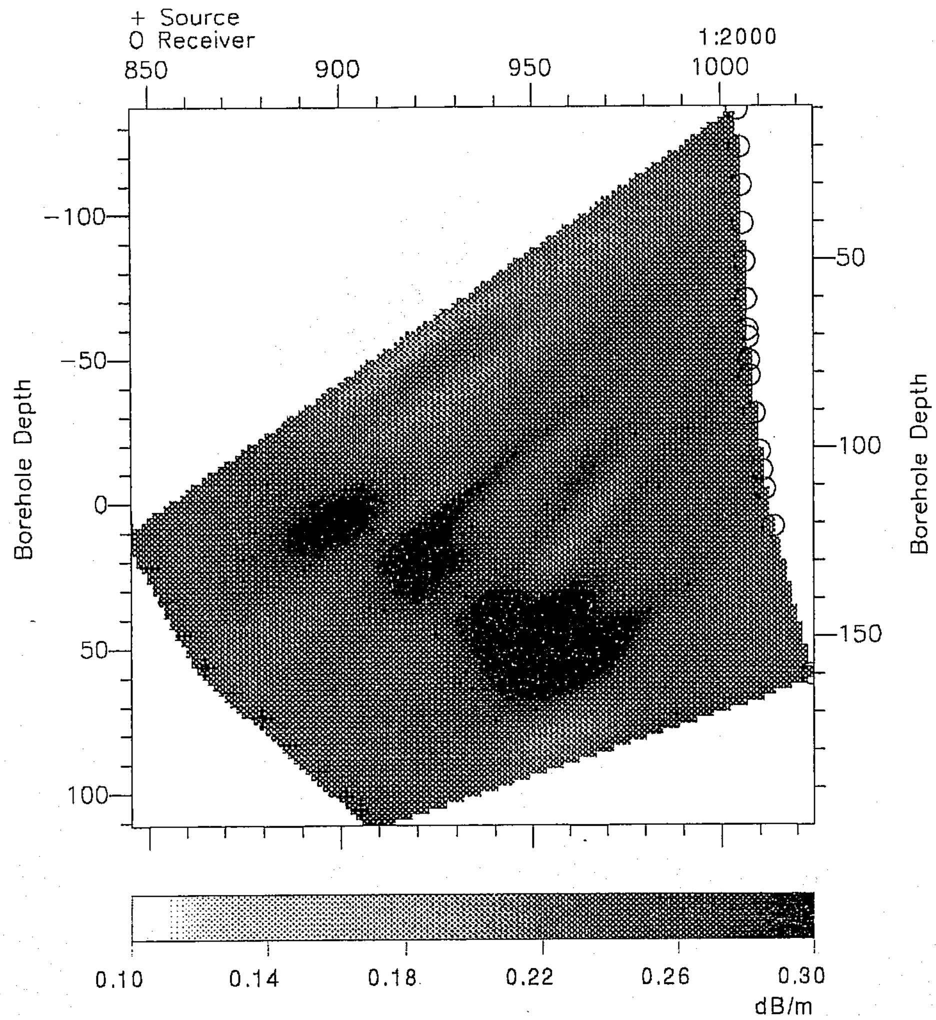

Figure 4. Image showing high-risk zone.

is suggested that development would need to be undertaken with caution in such an area.

CONCLUSIONS

Recent work in the NSW Southern Coalfields on the application of radio imaging to outburst detection shows promise for improving the confidence of geological structure prediction ahead of development roadways. A strong body of empirical evidence supports the association between high RIM attenuation rates (high bulk seam conductivity) and the presence of potential outburst structures. Equally important is RIM's capacity to establish low risk, non-hazardous clear coal ahead of the face.

In combination with in-seam drilling and gas content sampling, radio imaging can play a critical role in an improved predictive methodology which will improve safety in mining, and lower the overall cost to an operation of evaluating outburst hazards.

The application of the technique at Appin Colliery has established the clear difference in RIM attenuation between zones affected by outburst hazards and those which are not. The method remains to be fully integrated into the mining cycle and regularly applied, in order to improve operator confidence and ensure the potential of a valuable diagnostic tool is fully realised.

ACKNOWLEDGMENTS

The authors would like to thank David Reece and Jason Thomas of Appin Colliery for their help in organising and implementing the RIM survey program discussed in this paper. Thanks are also due to Ray Williams of GeoGAS for his comments on the application of gas content sampling to outburst prediction.

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Extent of Installation

The extent of installation and monitoring range from local mine face monitoring to overall mine-wide surveillance and source location. The level of human resources employed has been from automatic monitoring without human interaction, to dedicated team of research staff monitoring and analysing the signals on a round the clock basis.

Sensor Types

The sensors used are one of the following types:

- Accelerometers, high frequency sensors commonly used for acoustic emission (AE) monitoring.
- Geophones, medium frequency sensors principally used in seismic exploration surveys.
- Displacement meters or seismometers used by seismologists for earthquake detection.

These sensors may be single component or multi-component, usually cemented to competent coal or rock.

Sensor Locations

For high frequency accelerometers, the sensors need to be close to the source of the microseismic activities. As such they are usually installed near to the coal face, into the rib or roof. When the sensors are more than 15-20m from the face, they have to be reinstalled to a position closer to the face.

Most accelerometers need some form of preamplifiers to enhance the signals before transmitting them via long cables. These preamplifiers require electrical power (of low DC voltage) to function, hence a DC power source or battery is required.

For geophone-based sensors, the detection distance often ranges to several hundred metres without preamplification. They can be installed at the face, in an exploration borehole at seam level ahead of mining, or on the surface.

Displacement metres can be installed in mine or on the surface. They are much larger in size than accelerometers and geophones, and usually remain at one location for a long period of time after installation.

Microseismic Wave Type

As most of the researchers in these studies come from a mining engineering background, the wave types detected have been treated rather broadly. Most people only consider only one wave type, that is compressional (P) wave.

It was only in more recent investigations that researcher considered other wave types, such as shear waves, channel waves, and converted waves. These various wave types make data analysis rather complicated and source location inaccurate, as different wave types travel at different velocities.

Also, most researchers assume straight wave paths, which in realty are quite inadequate, as waves could also be refracted or reflected. This in an area that a lot of fundamental work is still required to achieve accurate location of the microseismic source.

Signal Frequency

The frequency of the microseismic signals has been the most debated issue in microseismic monitoring research. Some suggested that as outbursts are related to stress and strain redistribution, the frequency of the precursive acoustic signals should be very high, that is above 10kHz. However, signals of such high frequency are also very attenuative, and sensors have to be placed close the sources in order to detect them.

There are others who considered that signal frequency should be low, and characteristic of individual mining area. The frequency of signals used has been as low as 20 to 30Hz. Signals at these low frequencies can transmit long distances.

Other workers have opted for medium frequency signals, from 200 to 5000 Hz. These seem to agree with microseismic signals captured during small outbursts. However, a lot of other mining and environmental noises also have these frequencies. This makes filtering of real microseismic signals from noise difficult.

Signal Analysis

Earlier monitoring systems use analogue signal conditioning and filtering, and produce a warning light or sound if the signals exceed a certain threshold. The acoustic signals can also be listened to using a pair of headphones.

Modern systems first digitise the signals, and all subsequent processsings are performed digitally, using a special microcomputer, or the mine computing system. Some form of source location software is also used to confirm that the signals are related to real microseismic activities.

Data Storage

Most older systems store the data on high quality audio tapes in an analogue form. Newer systems store data in a digital form, on magnetic tapes, floppy disks, or removable hard disks.

Data storage, handling and archiving can be rather time consuming for mining personnel, and an efficient system is desirable to facilitate routine use.

Prediction Criteria

Most prediction criteria are based on an event count (number of microseismic events above a threshold value during a set period of time), or an energy count (accumulative energy content of the signals during a set period of time). Some rejection criteria of the signals may also be included, such as the hypocentre of the source from mining activity, or rise-time of the signals.

An outburst is considered as imminent if the event or energy count increases progressively for several periods, and suddenly reduces for a short duration.

Constraints and Complication

The most important constraint for an in-mine microseismic detecting system is contamination of signals by other mining induced acoustic noises. These noises include coal cutting, shuttle car moving, roof support drilling, or mine workers talking. The frequency spectra of some of these noises are shown in Figure 2, together with some real microseismic (AE) noises. It has been difficult to separate these events based on frequency discrimination.

It has also been time consuming and requiring special attention to move the sensors closer to the face as mining advances. This has discouraged some mining management to adopt microseismic monitoring on a routine basis.

For systems using surface sensors, while the requirement of moving sensors is reduced and easier, correlation of recorded signals and mining condition and activities underground is difficult. This is due to the fact that dedicated personnel are not underground or close to the face to observe and report activities of importance to interpretation.

FUTURE OPPORTUNITIES

In the past, most developers of microseismic monitoring systems aimed at producing a stand alone system for outburst prediction. This has been very difficult, and it is generally accepted that the chance of success in this approach is low.

Current thoughts have been to use microseismic as a tool to monitor the effectiveness of other preventative measures, such as gas pre-drainage, reduced mining rate, or stress relief by shot firing. Should microseismic indicates that, even adopting these preventative measures, there are still a lot of AE activities, then additional caution may be undertaken, or the mining slows down or stops for a while.

Sensors

To reduce the time consuming task of installing and moving sensors, major redesign of the sensors is required. Two sensors, both three-component, will be required to facilitate source location and directional discrimination.

The sensors should be of 'smart' sensor type, with build-in preamplification, filtering, and conditioning intelligence.

Cabling between sensors and base-station should be eliminated using a radio link. A form of this communication technique is currently being developed by an ACARP project at BHP Research (Project Number 3062).

IS Data Logger

In order for the monitoring personnel to attend to the sensors and be aware of the current mining activities, an intrinsically safe (IS) data logger/ processor is desirable. This allows the base-station be placed anywhere convenient, close to the face if required.

The above data logger would probably be a high power portable personal computer designed with IS specifications. A system that would suit such requirements is currently being developed by another ACARP project at BHP Research (Project Number 3070).

Signal Processing Techniques

Most of the signal processing techniques adopted so far use analytical digital processing algorithms. Current advances in artificial neural network (ANN) processing have shown that a combination of analytical and ANN would be more suitable for signal discrimination of AE data. This is worthy of consideration to reduce false alarms.

Correlation with other Remote Sensors

As the role of microseismic would be monitoring the effectiveness of outburst prevention processes, it is important that microseismic results be correlated with other active sensors during mining. Such sensors include gas contents, ground movement, cutting rate, vibration in cutting machines, etc.

CONCLUSIONS

The above sections have presented a summary of current practices in microseismic monitoring for outburst control. With the advances of electronics circuitry and computing hardware and software technologies, there is potential for microseismic monitoring as a tool to confirm that preventative measures are effectively implemented. However, more research is needed, and dedicated personnel is still the key factor to achieve success in this area.

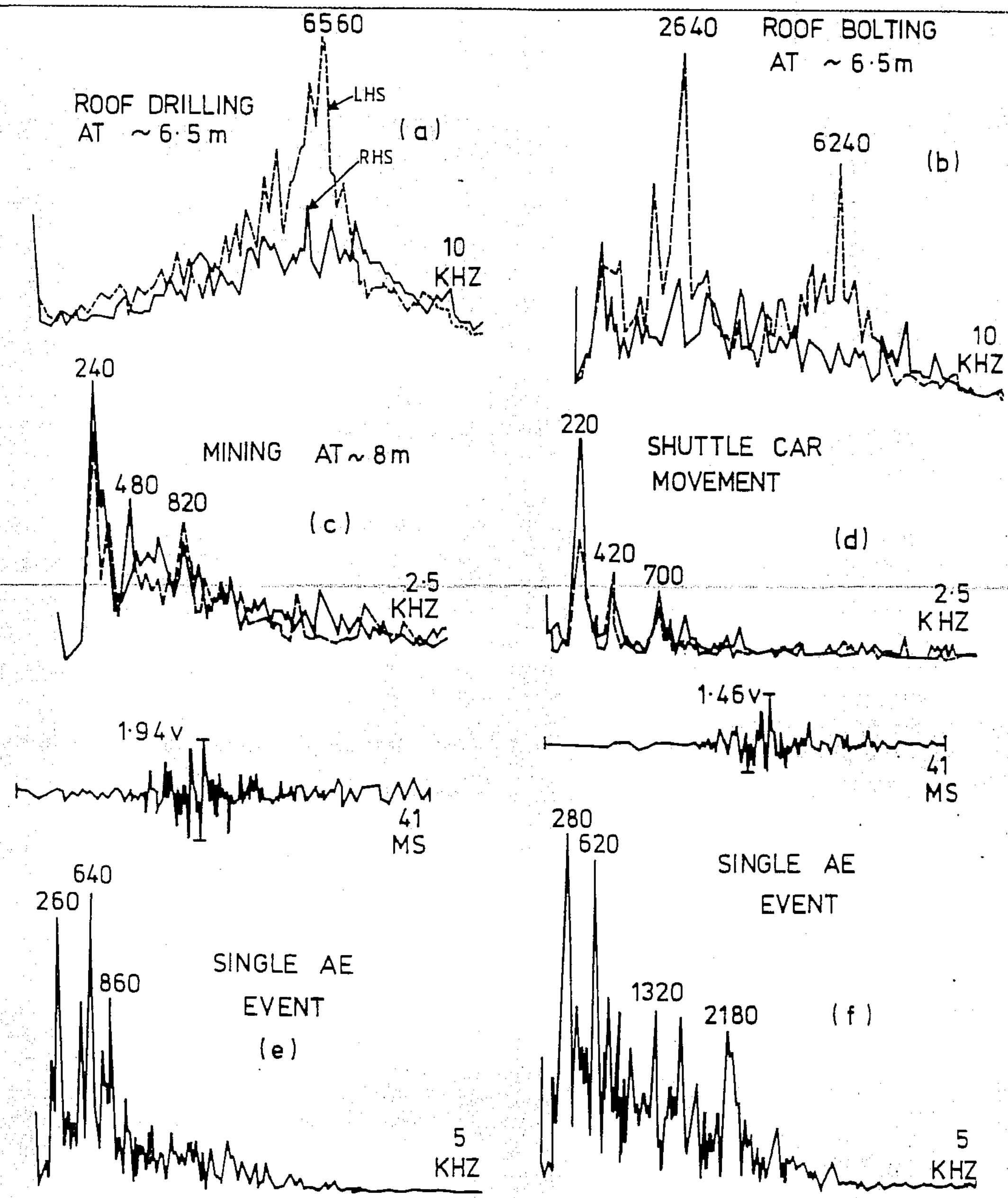

Figure 2. Typical signal spectra found at Metropolitan Colliery (Hewlett Packard 3582A Spectrum Analyser-Peak Mode, Hanning Window, Linear scales; to (c) inclusive 256 averages; (d) 32 averages.

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OUTBURST REMOTE CONTROL MINING AT TAHMOOR COLLIERY

Over ninety outbursts have occurred underground to Tahmoor Colliery since production commenced in 1978. These outbursts have invariably associated with geological anomalies, particularly dykes and faults. In the early 1980's techniques of prediction and protection gradually evolved with experience. However, existing systems were proven to be sadly inadequate in 1985 when a continuous miner driver was killed in a development heading. Consequently the concept of the encapsulated outburst continuous miner was developed by Tahmoor personnel. To go with this equipment, Outburst Mining Procedures were developed and crews given appropriate training. This system remained in operation for some six years and was successful in enabling safe mining of many outburst zones. The Tahmoor encapsulated miner concept remains the 'front line' defence against outbursts in the Bulli seam and forms the basis of the recent Department of Mineral Resources requirements for outburst miners.

Whilst reasonably successful of achieving its original aim of eliminating the risk of fatalities to miner drivers, the system proved to have several shortcomings, in particular:-

- Roof bolting was hand- held, and as only two operators were allowed under the Outburst Mining Procedure, was arduous for those involved.
- The procedures restrained advanced rates, typically to about 2m/shift, and as the mine encountered increasingly adverse geological conditions, lack of development was putting the mines viability at risk.

Efforts continued, with varying success, to improve prediction techniques. The ABM20 miner was introduced in 1991 in order to improve (normal mining) advance rates by the introduction of simultaneous cutting and bolting. Prior to the introduction of the ABM20, concerns were raised regarding the outburst risk to operators on its bolting platforms. Mine management, after considerable investigation, determined that the risk due to an unpredicted outburst was too high and implemented a policy of scout drilling and pre-drainage of suspect areas. Within a year, this policy had evolved tp pre-drainage in advance of all development roadways, (12cm panels as well as ABM panels). This policy has proved to be highly successful because Tahmoor has now gone over 30 months without an outburst, whilst developing in areas on average the most outburst-likely mined to date. Obviously this success has been costly, as some thirty men are now employed full time on gas drainage. However, no other known technique could have enabled the elimination of outbursts to this extent. Tahmoor now believes effective gas drainage will, in time, be proven to be a totally adequate counter to the outburst risk. However, several years experience/data will be necessary to prove that the outburst risk has been controlled. In the meantime, Outburst Mining Procedures must remain. During the early months of operation of the ABM20, the crews involved started thinking about the machine's potential to improve Outburst Mining Procedures and gradually a concept of remote mining using the ABM20 was evolved.

EXPERIENCES TO DATE WITH REMOTE MINING

The crews recognised that the ABM20 had several features which lent themselves to a superior outburst miner. Firstly, the drill rigs took most of the hard work out of bolting, thereby eliminating the problem of only having two operators allowed to bolt. Secondly, the machine had a sophisticated radio control giving some feedback (in the form of a LCD display) at the radio control unit. Thirdly, being a full-width machine, there was no need to change sides, which would be additional difficulty for a conventional machine in remote mining. Initially thoughts were given to driving the ABM from a 'phone booth' type shelter somewhat back from the face. However, it was soon concluded that it was best to remove the risk totally by having no one near the face whilst cutting (N.B. all Tahmoor outbursts had occurred during the cutting phase of the mining cycle). To enable remote mining all that was required was sufficient visual information to enable the miner driver to accurately cut coal whilst seated 150m from the face at the fresh air base. Essentially all that was required was a flame proof video camera, a flameproof video monitor, a communications system to link the two and a system to link the radio control to the miner, as the original radio control system was essentially 'line-of-sight'. A camera and

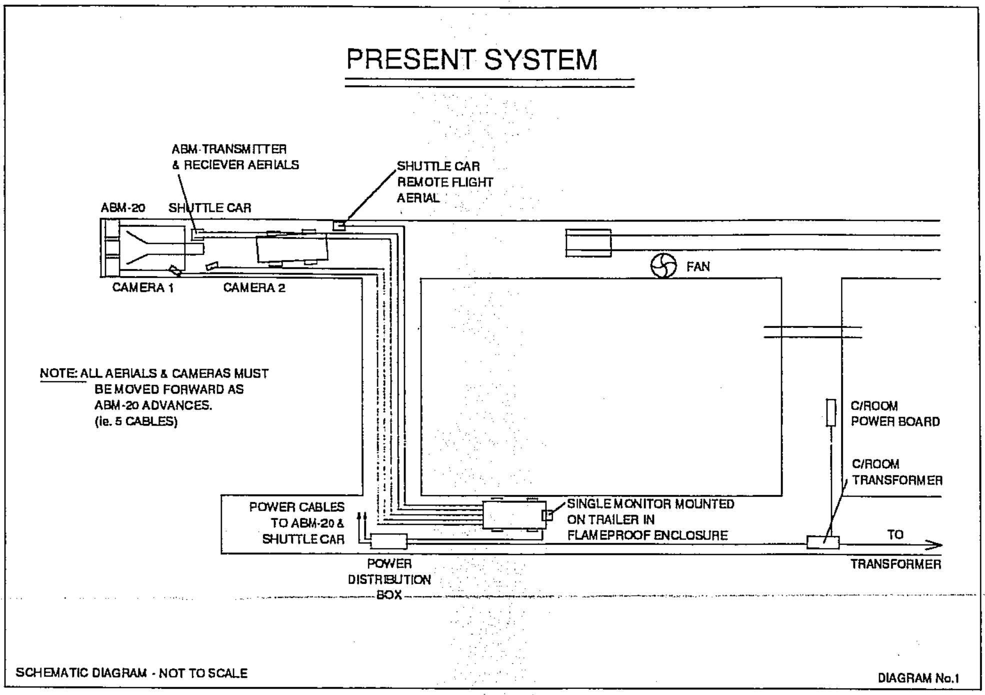

Figure 1.

monitor (black and white) were sourced and trialled underground. After some adjustments were made, reasonable quality pictures were consistently being received at the fresh air base. Linking of the radio control was achieved by the use of a simple aerial extension using coax cable strung along the roof bolts. A system was already in operation enabling remote control of the shuttle car flights. Appropriate risk reviews were done and a remote system was developed by the crews, management and inspectorate working together. The remote aspects of the system were focussed on the cutting phase of the cycle. The tramming of the shuttle car and the bolting were essentially as per the previous system except that the bolting was potentially easier and more efficient because of the hydraulic rigs.

At this time the panel where the ABM was located was approaching a known outburst zone (a 3 - 3.5m fault). Preparations were completed in time and the crews were keen to give remote mining a try. The fault was first mined in the left hand heading using the encapsulated 12CM. As suspected, a significant outburst occurred on the fault. The miner driver was adequately protected by the system but clearly of the opinion that it was not an experience he would knowingly go through again.

The ABM was then advanced in the right hand heading towards the fault. Various teething problems were encountered, particularly with effectiveness of radio controls due to aerial shortcomings. However, normal outburst rates of about 2m/shift were achieved. When the fault was reached and the machine started to cut up through it, a major (80t +) outburst occurred. This was witnessed on the monitor by the crew at the fresh air base. We believe this was the world's first televised outburst. The crew was most disappointed that it was not possible to have a video recorder to share this unique experience. Unfortunately the outburst did some damage to the ABM canopy, which meant it had to be withdrawn for repairs. The mining of the zone was then completed with an encapsulated Joy machine. However, the remote mining of outburst zones had been proven to be feasible and worthy of further development. The main problems encountered were:

- Lack of reliability of the radio control system, due to problems with aerials, and
- Lack of picture quality – whilst adequate views were possible, there was much scope for improvement.

FURTHER DEVELOPMENT OF REMOTE MINING

In June 1993 ACARP Project No. C3035 commenced. The objective of this project is to reduce the existing barriers associated with remote control outburst mining at Tahmoor and to set up a system which is expandable and safer with improved control and improved monitoring. The improvements in real time monitoring will enable historical data to be evaluated which eventually may assist in future outburst prediction.

The development of the separately ventilated control room for use in the hazardous zone means that the face miner machine condition plus CH_4 and CO_2 real time information is immediately available to the outburst team. The environment inside the control room is continually monitored for methane and flushed

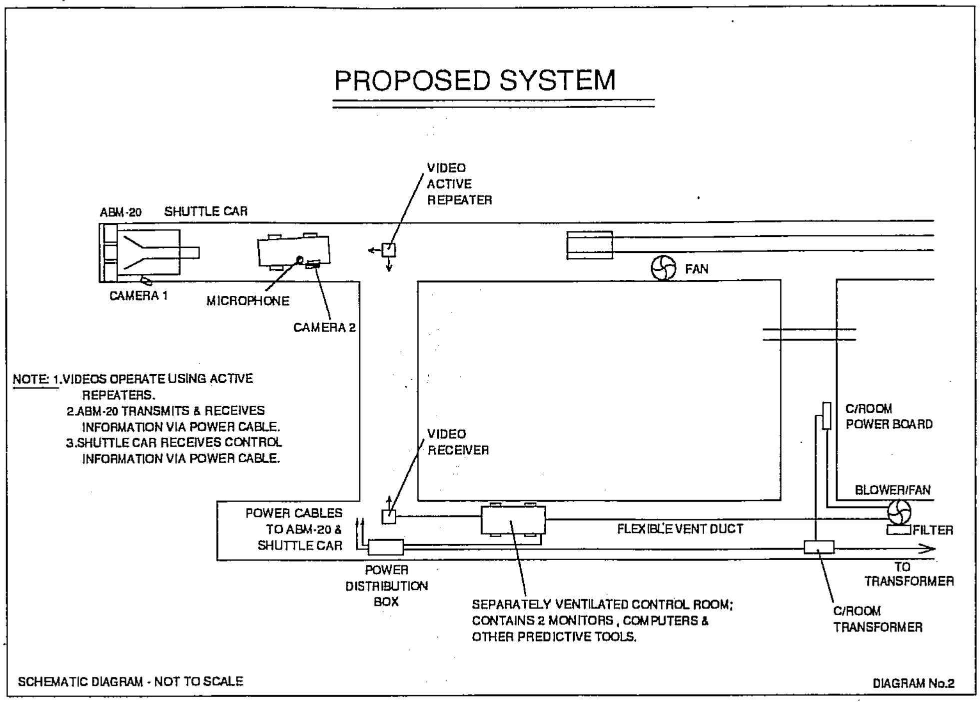

Figure 2.

with filtered air from outbye the hazardous zone. The electrical equipment used inside the separately ventilated control room does not have to be intrinsically safe or flameproof.

Spread spectrum power line carrier, used for control as well as monitoring, will provide more flexibility. No additional signal cable will have to be installed for monitoring and control. The monitoring and control can be carried out from any point along the panel power supply system and the driver will be able to adjust the miner cutter head and conveyor control hysteresis from inside the control room.

The computer screen inside the separately ventilated control room will display real time information for cutter head position, cutter motor current, pump current and conveyor current. Real time trend information on face % methane and % carbon dioxide will also be available on the same screen.

The miner driver will sit in front of two video monitors which will receive signals from cameras at the face. The method of video signal transmission will be by low power microwave. There will be one intrinsically safe (I.F.) battery powered camera mounted to the mine roof near the rib and adjacent to the ABM20 mine conveyor. This camera will be focussed on the cutter head and will also cover the area where the operators stand whilst they are bolting. The I.S. battery used to power the camera will also power the microwave video transmitter. This battery will have more than enough capacity to run the camera and transmitter for one shift. Video transmission will be assisted by passive repeaters which will be mounted to the mine roof at the intersections. These repeaters will enable the video signal to negotiate the corners between the camera and the control room.

The second camera, to be positioned near the face, is not intrinsically safe. We were impressed with the performance of this camera during selection trials so decided to mount this camera in a flameproof enclosure which will be mounted near the hungry board of the shuttle car. The miner driver will know when to start the shuttle car conveyor flights and when to stop the miner conveyor by viewing the monitor screen for this camera.

During the cutting cycle the miner driver will control the ABM20 miner from his seat which is inside the separately ventilated control room. He will use the same radio control transmitter to control the miner from this position as he would use when at the face during the bolting cycle. This transmitter has an outburst mode which will be selected during outburst mining. When the transmitter is in outburst mode the driver can remote control:

- The lowering and raising of the miner canopy whilst at the face during the bolting cycle, and
- When operated from inside the control room will have two additional select functions which control the shuttle car flights and provide a means of disconnecting power to the shuttle car and mine by opening the shuttle car and miner pilot circuits.

When the driver is using the radio control transmitter inside the control room there is a receiver directly in front of him which connects to a modem and power line carrier interface unit. All control commands from the control

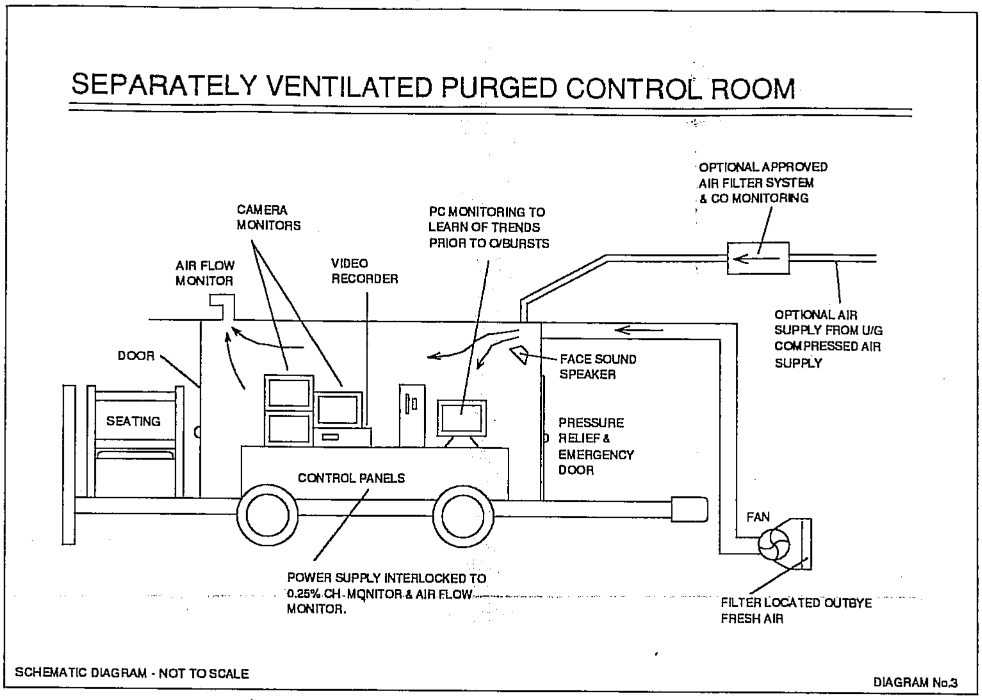

Figure 3.

room are via power line carrier to the miner and to the shuttle car.

Audio from the face will be transmitted to the control room via the video link.

A video recorder will be part of the separately ventilated control room furniture and will record the signal from the face I.S. camera. The recorder is a good quality surveillance recorder with time, date and 24 hour playback mode.

The separately ventilated control room has an enclosed area for the miner driver and monitoring control equipment. At the inbye end there is an open area with seating and rescue equipment. The open area is where the rescue operators can sit whilst other operators are at the face during the bolting cycle. The rescue operators can view the camera monitors through the room inbye windows whilst bolting is in progress at the face.

The separately ventilated control room is on wheels and has a draw bar. When the room is towed into position the wheels can be raised which lowers the floor and the roof of the room can be raised to a maximum height of 2.3m. This provides the miner driver with maximum available height.

The ventilation ducting from the outbye blower to the room made up from 100mm I.D., 20m lengths of spiral wire supported FRAS hose. Each 20m length has camlock fittings and attached dust covers for use when stored as individual lengths.

The outbye blower is a small 2.2KW unit manufactured to meet the Mines Department MDG3 guidelines. It is fitted with a fine filter on the inlet and will be positioned in service with its filter facing the centre of the roadway. The fan will be supported by its frame which will be supported from the roof.

Figure 1 shows the present setup for remote control outburst mining without the improvement mentioned above. Note the cables from the cameras to the video FLP small screen monitor and the coax cables used to extend the aerials from the outbye control transmitters on the open outburst trailer to the face area. There is also one other coax cable and inbye aerial which is used to feedback machine and methane level information. This information is displayed on a small LCD which is part of the miner control transmitter.

Figure 2 shows the new system using the separately ventilated control room, microwave video transmission and power line carrier for monitoring and control. A much tidier and easier system to set up with the additional advantages of the refinements in control and monitoring. The outbye blower is positioned outbye the hazardous zone.

Figure 3 shows a layout of the separately ventilated room.

CURRENT STATUS OF PROJECT

As of August, 1994, the design and manufacture of the room, communications and control systems is completed. All the relevant approvals have been received. It is intended to mine through a simulated outburst zone in September, 1994, as 'unfortunately' there are no known outburst zones to be mined at Tahmoor in the near future.