Development of in-seam directional drilling in the Australian coal industry

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AFFIRMATION

I, Frank Hungerford, declare that this thesis, submitted in fulfilment of the requirements for the award of Doctor of Philosophy, in the Department of Civil, Mining and Environmental Engineering, University of Wollongong, is wholly my own work unless otherwise referenced or acknowledged. The document has not been submitted for qualifications at any other academic institution.

Frank Hungerford
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ABSTRACT

The presence of seam gas in the form of methane and carbon dioxide presents a hazard to underground coal mining operations. Mining depths have progressively increased with the resultant increases in seam gas content and pressure while improvements in technology have allowed increases in mining advance rates. This has resulted in an increased risk of outburst during mining development and higher gas volumes released during mining. The drilling of in-seam boreholes to drain gas from the coal seam before mining was identified as the key means to mitigate the risk of outburst. In-seam drilling has been undertaken for the past thirty six years to reduce the seam gas content below defined safe threshold levels. This gas drainage was designed to mitigate the risk of gas outburst and lower the concentrations of seam gas in the underground ventilation.

The drilling practices have reflected the standards of the times and have evolved with the development of technology and equipment and the needs to provide a safe mining environment underground. Early practice was to adapt equipment from other fields, with rotary drilling being the only form of drilling available. This drilling was utilised successfully but with limitations due to the inability to steer the boreholes. This led to areas not having adequate drainage coverage.

Directional drilling was introduced through a research and development but limited by the standard of the technology available. It was not readily accepted as a viable alternative to rotary drilling due to the high cost and relatively low drilling rates compared to that of rotary drilling. Drill rigs from hard rock drilling industries were modified for use underground and combined with high pressure pumps to allow directional drilling with proto-type Polycrystalline Diamond (PCD) drill bits, low powered Down-hole Motors (DHMs) and Eastman single-shot surveying.

With a changing emphasis on gas drainage, research and development within the coal industry has created specific equipment, technology and practices to accurately place in-seam boreholes to provide economic and effective gas drainage.

Drill rigs were developed to specifications required for directional drilling in underground coal mines. Key elements of these drill rigs included flameproof electrics, mobility, self-contained anchorage systems, a high pressure water pump, a rotation lock for DHM drilling, 135 kN thrust for long-hole capacity and capacity to handle HQ sized
rods. CHD 76 rods have since been established as the preferred rod type for underground directional drilling.

The design of drill bits has evolved with improved and reliable performance with current drill bits fitted with PCD cutters capable of penetrating both coal and stone. The drill bit diameter was increased to a diameter of 96.1 mm and matched to DHMs fitted with 1.25 degree bends to provide directional control while reducing in-hole friction to allow longer boreholes. DHMs available with higher torque characteristics for improved drilling rates have had wear pads designed to reduce abrasion of the bent housing or deflection point of the DHM.

Surveying technology improved from the wire-line, single-shot Eastman survey instruments which were time-dependent on borehole depth. Development has produced real-time electronic instruments located in the drill string to supply and process accurate survey data to the drilling crew. This allowed improved directional control and increased drilling rates.

Directional drilling procedures were established and evolved technology improved. The recording of drilling data has been a key for analysis of drilling performance and to allow geologic interpretation. Operational parameters of DHMs were established and used in to recognize changes in strata through variations in water pressure. The directional responses to orientations of the DHM with the preferred bit diameter and bend have been established and used by drillers for steering. Those responses also set borehole curvature limits for the borehole design process. This process has been developed to give drillers a guide on seam profile and also define the required lateral positioning of each borehole. The process of progressively defining the seam profile has been well established.

Directional drilling technology has now been established as the industry standard to provide effective gas drainage drilling. Boreholes can be accurately located in a designed pattern to provide consistent and adequate coverage for drainage to reduce gas content levels below threshold level before mining advances through the area. Borehole patterns for cross-block drainage of longwall development panels vary from fan patterns of straight boreholes to all variations of the candelabra patterns. The candelabra pattern has initial curve sections to regular parallel spaced sections through the drainage area.
Where access is limited and drainage time allows, longer boreholes to 1400 m along development panels have been used regularly to provide longer term drainage.

Through the development of in-seam directional drilling, alternate applications were identified. The practice of progressive seam definition is an inherent exploration process proving the ground clear of structures. This has been expanded to specifically target known or suspected geological anomalies for identification and definition. Drill procedures are designed to suit each type of geological structure being investigated.

Directional drilling had been used to provide in-rush protection barriers, in-seam water drainage and drainage of water filled voids. Each of these projects required adequate design of standpiping and fittings to manage the expected water pressure. As a result of the fatal in-rush event at Gretley colliery, in-rush protection drilling is now mandatory for underground mines when mining in the vicinity of a water source. Risk assessment defines the controls to be used on each of these drilling projects.

Borehole friction in directionally drilled boreholes was identified as the cause of surging feed, stalling of the DHM and eventual limitation of borehole depth. Surging feed and stalling of the DHM stopped drilling before the maximum thrust capacity of the drill rig was reached. A combination of slide and rotary drilling with a DHM was used to attempt boreholes to 2000 m to provide gas drainage of a new development panel. In-hole friction was greatly reduced which allowed slide drilling to continue to beyond 1800 m. The thrust capacity of the drill rig rather than surging feed became the limiting factor. Rotary drilling continued the boreholes with a maximum depth of 2151 m achieved. Thrust while rotary drilling remained relatively constant over the length of the boreholes. The rotation system and water pressure capacities are the likely limiting factors when using combined rotary/slide drilling.

Directional drilling technology has been introduced to the Chinese, Russian and Polish coal industries for gas drainage through a practice of auditing, design, supply, training and ongoing support. The technology has been utilised in underground drilling for oil production in northern Russia. Industry applications and requirements have continued to serve as a major driver for the development of in-seam directional drilling in new and more challenging frontiers, and as such future research and development is needed to continuously advance this technology to meet these challenges.
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<td>AMT</td>
<td>Advanced Mining Technologies</td>
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<td>Asahi</td>
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<td>ACARP</td>
<td>Australian Coal Association Research Program</td>
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<td>ADS</td>
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<td>AFD</td>
<td>Automatic Firedamp Detector</td>
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<td>BeCu</td>
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<td>CSS</td>
<td>Cableless Survey System</td>
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<td>CO₂</td>
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<td>Coal-bed Methane</td>
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<tr>
<td>CHD</td>
<td>Composite Heavy Duty</td>
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<td>Consol</td>
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<td>DDM</td>
<td>Directional Drill Monitor</td>
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<td>DHM</td>
<td>Down-Hole Motor, also referred to as mud motor</td>
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<td>DGS</td>
<td>Drill Guidance System</td>
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<td>Easy Release Thread Grease</td>
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<td>MECCA</td>
<td>Modular Electrical Connected Cable Assembly</td>
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<td>PDC</td>
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<td>rpm</td>
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<td>RCS</td>
<td>Rod Communication System</td>
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<tr>
<td>USBM</td>
<td>United States Bureau of Mines</td>
<td></td>
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<tr>
<td>UDR</td>
<td>Universal Drill Rigs</td>
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CHAPTER ONE – GENERAL INTRODUCTION

1.1 IN-SEAM DRILLING IN UNDERGROUND COAL MINES

By the early 1980’s, drilling of in-seam boreholes was the accepted means to drain gas from coal seams to reduce the gas content below threshold levels. This mitigated the risk of outbursts, lowered the gas concentration levels in ventilation systems and, in the case of methane seam gas, reduced the threat of explosion.

Gas drainage drilling programs were established using in-seam and cross-measure rotary drilling. Although providing a level of gas drainage, with no directional control, borehole curvature and early terminations in stone, rotary drilling left areas with inadequate gas drainage coverage.

In-seam directional drilling was introduced to the coal industry from the mid-1980’s through research and development programs. The subsequent development of equipment, technology and practices has occurred through a variety of underground directional drilling programs supported by the coal industry. Surveying technology in real-time with in-hole electronic survey instruments and drill rigs are designed specifically for directional drilling. Down-hole motors with higher torque capacity have become available and have been fitted with wear pads, universal joints and bearing components developed in rigorous operational environments. The development has produced equipment specifically for directional drilling with emphasis on reliability. The development of the technology has enabled improved directional control and accuracy as well as increased drilling rates.

Rotary drilling had remained the preferred drilling mode for gas drainage through the period of development of directional drilling. High cost and slower drilling rates were listed as the main reasons. A fatal outburst event due, in part, to boreholes deviating out of position, led to the introduction of Outburst Management Plans which required the use of a drilling system that was able to position boreholes consistently. Directional drilling was the only form of drilling which could provide accurate placement of boreholes with surveying to verify their location.
Directional drilling practices and driller skills were improved to provide increased drilling performance as the technology was developed and adopted. Directional drilling technology has now been established in the industry for gas drainage to satisfy OMP requirements.

Through the development of directional drilling technology, alternate applications were identified. Exploration was an additional benefit with directional drilling as it has the ability to provide exploration data from long boreholes. The technology has been utilised to identify and define known geological anomalies. Directional drilling is a safe and reliable means to provide inrush protection and water drainage ahead of mining.

The author of this thesis has been involved in the development of this technology in the 30 years since its introduction into the coal mining industry in Australia. This thesis intends to provide a comprehensive record of the development and key milestones in the progress and applications of this technology as it stands today.

1.2 RESEARCH OBJECTIVES

The objectives of this thesis are:

1. To present a comprehensive view of the development of directional drilling technology
2. To describe the development of drill rigs, drill bits and survey instruments required for the directional drilling operations.
3. To explain the drilling practices developed for directional drilling in the underground coal industry.
4. To describe the applications of directional drilling which have been established in gas drainage, exploration and water management.
5. To explain the effects of in-hole friction on limiting borehole depth and investigate the use of a combination of rotary and slide drilling to limit in-hole friction and extend borehole depths to beyond 2000 m.
CHAPTER ONE
General Introduction

1.3 THESIS OUTLINE

This thesis is presented in ten chapters. The general layout of the thesis is presented in Figure 1.1.

![Figure 1.1: Structure of chapters in the thesis]
This thesis presents a comprehensive summary of the development of in-seam directional drilling technology and its application in underground mining.

- Chapter 1 presents the general purpose of the research and development and objectives of the thesis.

- Chapter 2 reviews the evolution of in-seam drilling technology from the introduction of routine rotary in-seam drilling in the early 1980’s through to the current directional drilling applications.

- Chapter 3 identifies and explains the initial modifications of imported drill rigs through to the development of drill rigs designed specifically for directional drilling in underground coal mines. The performance, development and applications of the drill rods and down-hole motors utilised are defined.

- Chapter 4 explains the improvements in PCD drill bit technology which have been a major factor contributing to more efficient and economical in-seam drilling.

- Chapter 5 analyses the limitations of initial survey systems and improvements to drilling efficiency provided by the development of electronic survey instruments. A borehole design and plotting system was established for accurate steering and seam profile definition.

- Chapter 6 documents the directional drilling practices which were developed for drilling longer in-seam boreholes located in planned locations while progressively defining the seam profile. The operating and steering characteristics of down-hole motors required for directional control are defined.

- Chapter 7 provides an appraisal of directional drilling practices utilised for both pre and post gas drainage applications. The various drilling patterns and procedures used are evaluated using case studies covering a range of mining situations.

- Chapter 8 shows the versatility and effectiveness of directional drilling practices adapted for a variety of underground explorations applications. Directional drilling has also been utilised for a range of in-rush protection and water management applications which are explained with case studies provided.
• Chapter 9 explains the effects of in-hole friction on limiting borehole depth and reports on an investigation into the use of a combination of rotary and slide drilling to limit in-hole friction and extend borehole depths to beyond 2000 m.

• Chapter 10 summarises the results and principal conclusions of the research and development work presented in the thesis and lists suggested areas requiring further research.
CHAPTER TWO
Development of Underground Drilling in the Coal Industry

CHAPTER TWO – DEVELOPMENT OF UNDERGROUND DRILLING IN THE COAL INDUSTRY

2.1 INTRODUCTION

In-seam drilling to manage gas concentrations in mine ventilation and reduce the effects of outbursts as mining depth and development rate increase has become a part of the underground coal mining industry. Development of drilling technology was undertaken in response to research into outburst control and the need to reduce the gas concentrations in coal mines. As gas drainage was identified as a viable means to reduce gas content and pressure, work has been directed at developing the capability to drill in-seam boreholes to greater depths and position the boreholes accurately to the greatest effect.

The development of underground drilling has involved aspects of surface drilling, underground hard rock drilling and eventually oilfield directional drilling applications and adapting them to the underground coal drilling environment. This work has also been undertaken in different countries in parallel with their separate drilling environment challenges with limited interaction between each of the programs.

This chapter describes the introduction of in-seam rotary drilling to the US, European and Asian coal industries and the following development of directional drilling and its applications. The mine based introduction of rotary drilling in Australia which was driven by a need to manage the outburst risk is explained. Research and development was then the main facilitator of the introduction of directional drilling, the results of which and the progressive introduction to the Australian coal industry are described.

The conditions which eventually led to directional drilling being adopted as the primary form of underground drilling for gas drainage in Australian coal mines are identified. The evolution and application of this technology have been described. The additional applications of underground directional drilling for exploration and water management have been identified, developed and used successfully. These are explained in this chapter.
2.2 UNDERGROUND DRILLING DEVELOPMENT IN EUROPE AND ASIA

After World War II, increased production due to mechanisation and the mining of deeper and gassier coal seams necessitated methane drainage throughout European coal fields. The emphasis on gas drainage drilling was directed at boreholes above and below the working coal seam where it has become a fully integrated part of longwall mining to extract gas produced from adjacent seams and from longwall goafs. Experimental studies were conducted in England from 1957 to develop tools and equipment for drilling horizontal holes in coal (Baxter, 1959) with holes to 135 m being achieved. Although further studies were conducted in the early 1970’s in England with 100 m long in-seam rotary drilling (Highton, 1972) and in the early 1980’s in Germany with rotary scroll drilling to 120 m depth in-seam (Noack, 1982), the emphasis in Europe has always been on cross-measure drilling for post-drainage or goaf drainage to reduce the ventilation required to dilute gas concentrations (Boxho, et al, 1980). The highly stressed and fractured nature of the European coal seams has not been conducive to stable in-seam drilling.

Underground coal mining in Japan had extended out under the sea Figure 2.1 (Taiheiyo Coal Mining Company Ltd, 1996) with high gas content at depths greater than 600 m and large displacement faulting (>100 m). The mine had established a system of rotary drilling for exploration and post drainage from the longwall goaf with 300 m long boreholes. Longer boreholes to 500 m were achieved with a step-down system of rod sizes using a large capacity Top-L rig shown in Figure 2.2 (Hungerford, 1997)
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2.3 UNDERGROUND DRILLING DEVELOPMENT IN THE USA

2.3.1 Rotary Drilling in the USA

Underground drilling in the USA for gas drainage did not seem to be co-ordinated until the United States Bureau of Mines (USBM) began studies from the early 1960’s through the 1970’s into gas content and permeability (Cervik, 1969; Kissell and Edwards, 1975). Early work of gas flows through coal seams was done with information from surface boreholes and underground rib emissions (Zabetakis, et al, 1972). An extensive study on horizontal drilling in 1970 (Williamson, 1970) concluded there were no reliable published data from the previous British studies on thrust, rotational speed or bit configuration, and the effects of these on hole trajectory. Also following that study,
horizontal degasification holes were drilled in the Pittsburgh seam to depths of 255 m (Fields et al., 1973) with no records of the drilling parameters.

There has always been a challenge in drilling with keeping boreholes straight and avoiding adverse deflections (MacDonald and Lubinshi, 1951; Grace, 1974). Trials have even been conducted using rods made of heavy metal (spent uranium) to provide a very hard and heavy drill string in attempts to find advantages in directional drilling and deviation control (Bradley, et al., 1976). The added challenges with horizontal drilling within a coal seam involved the effects of gravity on the drill bit and rods, and the penetration through strata with varying characteristics.

Although work was conducted in 1969 to define the requirements of equipment to drill horizontal holes in coal seams (Hadden and Cervik, 1969), drill rigs were usually low capacity hand-held units (Cervik, et al., 1977), “home-made” or proto-type rigs (Finfinger and Cervik, 1980) before established drill rig manufacturers began modifying hard-rock coring rigs or surface drilling units for use in underground coal mines. The USBM conducted research into attempts to maintain direction control with rotary drilling (Cervik, et al., 1975) in which a proto-type Longyear drill rig was used to drill 89 to 92 mm diameter holes to maximum depth of 637 m. This involved the positioning of stabilisers (Figure 2.3) at various locations behind the drill bit (Figure 2.4) (Kravits, et al., 1985) to achieve upwards or downwards deflection of the hole with variations in rotational speed and penetration rate to try to maintain the hole within the seam. Some level of vertical control was achieved but no control was possible in the lateral plane. These were fixed configurations which required the pulling of the drill string to modify control characteristics. At this stage of development, rotary drilling with vertical control variations in down-hole configuration, rotational speed and penetration rate was regarded as “directional drilling”. By the late 1970’s, Acker (Acker “Big John) and Fletcher (Fletcher LHD) were manufacturing rigs specifically designed for underground coal drilling.
As well as underground drilling in coal mines to achieve gas drainage, research projects included drilling in-seam holes from a large diameter (1.83 m diameter) borehole or shaft (Figure 2.5) with 1.22 m casing installed down to an enlarged gallery (3.05 m diameter) reamed out at the coal seam level (Fields, et al., 1973; Deul and Kim, 1975). Holes were drilled to a maximum depth of 255 m using NX rods, 76.2 mm (3”) replaceable tungsten carbide drag bits and various stabiliser combinations with a drilling machine bolted to the floor. This method was subsequently upgraded with 8 x 180 m
long holes drilled from a 7.2 m diameter ventilation shaft sunk three years in advance of mining in the Beckley Coalbed, West Virginia (Finfinger and Cervik, 1980).

The first work on surface-to-in-seam drilling began in 1977 (Diamond, et al, 1977) with boreholes planned to start from vertical and intersect the Pittsburgh seam at vertical depth of 300 m before being continued in-seam with rotary drilling. Directional control with rotary drilling was not capable of following the seam to any substantial depth. Further trials were proposed (Oyler, et al, 1979) which eventually led to the successful Coal-bed Methane (CBM) industry throughout the USA.

2.3.2 Directional Drilling in the USA

A review of in-seam drilling practices in the USA (Finfinger and Cervik, 1980) identified that several mines had established limited in-house gas drainage programs led by research and development by the USBM. With in-seam drilling proven and established as a viable means to manage seam gas within mine workings, several mines
had developed in-seam rotary drilling programs for degasification in co-operation with the USBM with boreholes to 300 m depths. Two mines had developed in-seam directional drilling using Down-hole motors (DHM). Ker-McGee Corporation used a wire-line single-shot surveying instrument to achieve 20 boreholes to a maximum depth of 900 m while Consolidation Coal Company (Consol) had developed a wireless electronic survey instrument and drilled 20 holes to 300 m depth.

The coal mining industry in USA had not uniformly adopted down-hole motor drilling as the preferred form of drilling to position boreholes for optimum gas drainage effect. Consol was actively developing directional drilling in parallel with their rotary drilling program (Thakur and Poundstone, 1980; Thakur, et al, 1988) while the USBM was still using rotary drilling with stabilisers in their gas drainage research and development (Finfinger, et al, 1982).

With the use of DHMs still being developed for directional drilling, the USBM conducted a study comparing rotary and directional drilling technologies (Kravits, et al, 1985). Two holes were drilled (Figure 2.6 and Figure 2.7), one with rotary drilling control techniques using stabilisers and a drag bit (Figure 2.4) with tungsten carbide cutters, the other using a 70 mm Christensen Navi-drill DHM with a roller cone bit.

Figure 2.6: Profiles of trial boreholes – A. rotary drilled borehole, B. directional drilled borehole (Kravits, et al, 1985)
Deflection of the DHM was provided by a deflection shoe at the front of the DHM (Figure 2.8, Kravits, et al., 1985), rather than the bent housing and wear pads currently employed in Australia. The surveying of each was achieved with an NL Sperry-Sun wire-line single-shot survey instrument. The effects of the orientation of the DHM on borehole trajectory are shown in Figure 2.9, although this diagram did not indicate the effects of DHM torque shown with directional drilling in Australia. This project verified the viability of directional down-hole drilling and advantages over conventional rotary drilling in the USA. Directional control was achieved in both the vertical and horizontal planes with DHM drilling while drilling configurations had to be changed for vertical control with rotary drilling. No control was possible in the horizontal plane with rotary drilling. Initially the rotary borehole curved approximately 14º to the right then maintained that particular alignment. Slotting was occasionally required to continue the rotary drilled borehole after stone intersections while branching was used to continue drilling in the directionally drilled borehole (Figure 2.6). A subsequent project by the USBM demonstrated the accurate use of directional drilling to intersect a known target (Kravits, et al., 1985).
Polycrystalline diamond (PCD) cutter bit technology had been developed by the oilfield industry (Varnado, et al, 1980) and utilised in the coal industry for stone drilling in cross-measure drilling applications in the late 1970’s. The benefits of PCD bits were apparent with the ability to provide good penetration rates in coal and maintain good penetration when stone strata was intersected without having to change bits. As the use of PCD bits expanded, improvements were made in the technology to improve
performance and overcome problems as they were identified (Huang and Iversen, 1981; Swenson, et al, 1981).

Borehole surveying had been conducted using wireline single-shot Sperry-Sun instruments from the start of the development of in-seam drilling technology in the USA. Consol developed the Conoco electronic survey instrument in late 1980 which utilised acoustic signal transmission along the drill rods. This instrument was licenced to DuPont for use in Australia. Geoscience produced the Model 24 survey instrument which utilised electromagnetic signal transmission. Even though these electronic survey instruments had been developed and were operational in the USA, the USBM developed its version of the survey instrument which used electromagnetic transmission as a pre-production unit and conducted in-mine trails (Kravits and Millhiser, 1990). By the 1990’s, the technology was commercially available in the USA capable of surveying long-holes to 1000 m ahead of development for gas drainage.

Seam gas drainage has been developed with short directionally drilled borehole within longwall blocks (Figure 2.10) to reduce general gas content in both longwall block coal and development roadways (Dudley, 2010).

![Figure 2.10: Short-hole drilling for seam drainage (Dudley, 2010)](image)

Research and development had been undertaken into gas drainage from and around advancing longwall faces and associated gob (goaf). The research involved vertical goaf holes from the surface, cross-measure rotary holes and directionally drilled long-holes above and below the longwall (Figure 2.11, Kravits and Li, 1995). Comparisons were made with all modes of post-drainage to prove the viability of long-hole drilling above the longwall (Figure 2.12). This drilling was targeted at both stone strata above the seam and overlying seams. Directional drilling is now in regular use for goaf drainage.
from above extracted longwall blocks or advancing longwall faces (Diamond, *et al.*, 1992; Kravits *et al.*, 1993; Kravits and Li, 1995; Brunner and Schwoebel, 2010).

Directional drilling was also being established as an in-seam exploration tool in the underground coal mines in the USA for the definition of faults (Kravits and Schwoebel, 1993).

The USBM was abolished in 1995 which left further development of underground directional drilling technology in the USA to the mines conducting in-house drilling projects and the two main contract drilling companies at the time - Resource Enterprises, Inc. (REI) and Target Drilling, Inc. (formerly Advanced Mining Technologies (AMT) Drilling International).

![Diagram of long-hole goaf drainage applications](image1)

Figure 2.11: Long-hole goaf drainage applications (Kravits and Li, 1995)

![Diagram of long-hole goaf drainage](image2)

Figure 2.12: Long-hole goaf drainage (Kravits and Li, 1995)
From the start of the 1990’s, long-hole in-seam drilling was established as a viable gas drainage application in US coal mines (Kravits, et al, 1999) with holes regularly drilled to beyond 1200 m (Figure 2.13, DuBois, et al, 2006). Target Drilling established a capability of drilling long in-seam boreholes for gas drainage with one project involving 33 in-seam boreholes drilled to depths greater than 1219 m (average 1473 m, Kravits, et al, 1999). In support, Target Drilling have developed a system of pumping a cross-linked polymer gel into long boreholes (Kravits, et al, 2006) after effective drainage to provide an inert borehole to be subsequently intersected by mining without gas issues (Figure 2.14, Target Drilling, 2006)
Both companies have developed and refined capabilities in long-hole directional drilling for exploration to define seam displacements, igneous intrusions and associated burnt areas (Figure 2.15, Bohan, 2012), in-seam and gob degasification (Kravits, et al, 1999; Brunner and Schwoebel, 2007), and cross-measure drilling for water drainage/management. To support underground mine degasification, surface-to-in-seam (SIS) drilling has been established and offers a viable alternative (American Longwall, 2007).

Potential inrush has been identified as a threat when mining adjacent to flooded old mines. Drilling flanking in-seam boreholes to provide a protection barrier and delineate reserves when mining within 60 m of old workings (Figure 2.16) has been established to satisfy MSHA requirements in USA coal mines (Bohan and Brunner, 2005).

![Figure 2.15: Directional drilling to characterize the extent of burn into the reserve. (Bohan, 2012)](image1)

![Figure 2.16: Barrier definition with in-seam boreholes (Bohan and Brunner, 2005)](image2)
2.4 UNDERGROUND ROTARY DRILLING DEVELOPMENT IN AUSTRALIA

2.4.1 Early Rotary Drilling Development

As early as the 1920’s, in-seam drilling was being used in Australia in attempts to solve the problems presented by high gas content in underground coal mines. After a fatal outburst occurred at Metropolitan colliery in 1925, the mine adopted a practice of drilling two 43 mm diameter boreholes up to 40 m in advance of each development face along the extension of the roadway rib-lines (Battino and Hargraves, 1982). In the carbon dioxide (CO₂) environment that existed, this practice was regarded as unsuccessful at preventing outbursts so was discontinued after a fatal outburst in 1954. Studies at Metropolitan colliery in 1963 on the drainage effect of boreholes in a CO₂ environment had stated they were ineffective due to very short drainage times (Clark, 1983). Consequently, gas drainage holes were abandoned for some time in favour of drilling and reaming larger diameter boreholes (to 300 mm diameter) (Hargraves, 1983) to depths of 80 m ahead of the development faces at Metropolitan and Corrimal collieries for stress relief in an attempt to reduce the occurrences and effects of gas outbursts. This practice was eventually discontinued as in-seam drilling for gas drainage became the established practice to manage gas and outbursts (Clark, 1983).

From the late 1970’s, trials were conducted at West Cliff colliery (Lama, et al, 1980) and Appin colliery to determine the effectiveness of gas drainage by in-seam drilling. Rotary drilling programs were conducted using a compressed air/hydraulic powered Diamec 250 drill (modified for use in underground coal mines) mounted on posts secured between roof and floor to drill patterns of 40 and 120 m length, 43 mm diameter holes (Levers, 1979; Jagger, 1978) as a precursor to developing full scale gas drainage drilling programs at each mine.

Cross- measure rotary drilling for post-drainage was introduced at Appin colliery from late 1980 with 120 m long, 41 mm diameter boreholes drilled down at a variety of angles through the strata down to and into the Wongawilli seam under Longwall 7 and 80 m long boreholes drilled into the overlying strata for goaf drainage (Battino and Regan, 1982). This was undertaken using an Edeco Hydrack drill rig, 41 mm diameter E rods and a 55 mm PCD Stratapax bit.
Research into gas drainage with in-seam boreholes was also undertaken at Leichhardt, Cook and Metropolitan collieries (Clark et al, 1983), Tahmoor colliery (Stone and Davis, 1983) and Collinsville No. 2 mine (Williams, et al, 1983). The smaller drilling projects at Cook and Leichhardt collieries (Gray, 1982) involved drilling 43 mm diameter holes with an air-leg assisted borer (Hanes, 1997a).

### 2.4.2 Rotary Drilling for Gas Drainage

Although improvements to rotary drilling technology were being achieved through research projects, the mines using rotary drilling developed techniques suited to their specific conditions very early due to the high pressures placed upon the gas drainage sections to be effective. The big benefit of rotary drilling in most cases was the very high drilling performances with 250-300 metres drilling possible per shift. This was achievable with penetration rates of 2-3 m/min in coal in conditions where the boreholes would follow a particular section of the seam or the bit deflected off low angle intersections with the harder roof or floor strata.

Full scale gas drainage drilling programs were established at West Cliff colliery (Marshall, et al, 1982) and Appin colliery (Kelly, 1983) in 1980 and 1982 respectively.

The in-seam drilling at Appin colliery achieved 230 m long 80 mm diameter in-seam boreholes with BQ rods and Widia bits which had the tendency of deflecting off very low angled intersections with roof or floor to remain in-seam. Nominal lateral curve trends were established with standard drilling parameters for borehole layout planning with borehole intersections surveyed for verification.

Initial trials at Collinsville No. 2 mine involved 100 mm holes drilled to 80 m (Gray, 1982) and 44 m (Williams, et al, 1983) with a compressed air operated Atlas Copco BBR 601 drill rig (Hungerford, 1981).

A NERDDC sponsored in-seam drilling program was undertaken by ACIRL at Tahmoor colliery (Allan and Richmond, 1982) to assess the viability of drilling long boreholes in-seam. The ability to drill long-holes in-seam was proven to a maximum depth of 732 m at Tahmoor colliery (Figure 2.17) and 471 m at West Cliff colliery but lateral control was identified as a problem for consistent gas drainage coverage with one hole deviating by as much as 68 degrees over 260 m (Hebblewhite, et al, 1982). This
was probably either the effect of cleat or the result of drilling an extended distance skimming along the floor of the seam.

![Figure 2.17: Plan of rotary boreholes – Tahmoor colliery (Allan and Richmond, 1982)](image)

An in-seam gas drainage drilling research project (NERDDP No. 495) at Collinsville No.2 mine (Beamish, et al. 1985) employed an electro-hydraulic Diamec 251 drill rig with BQ rods and a selection of 80 mm tungsten carbide drag bits. Vertical control was managed through variations in rotational speed and penetration rate with progressive Eastman wire-line surveying at 30 m intervals to monitor the borehole position and trend. Most drilling operators in Australia preferred not to use stabilisers due to the fact that each configuration had a fixed deflection trend and required the extraction of rods to change the configuration. The Widia bit design of pilot and reamer had a tendency to deflect off low angled stone intersections but terminated in high angled intersections with stone due to rapid wear.

The Widia bit proved the most effective tungsten carbide drag bit but still blunted rapidly with stone intersections. The introduction of the PCD “Terratek” bit led to good penetration in both coal and stone without reduction in bit performance due to wear. Deflections off the roof resulted in a deviation to the left (Figure 2.18) while deflections off the floor were expected to deviate to the right.

The outburst research project (NERDDP No. 81/1006) conducted at Collinsville No. 2 mine (Beamish, et al, 1985) identified limitations of rotary drilling for long-hole gas
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Development of Underground Drilling in the Coal Industry

drainage projects, problems which have to be managed with continuous drilling operations and established guidelines for conducting safe and efficient gas drainage drilling in gassy conditions.

![Borehole deflection groove in roof exposed by development](image)

Figure 2.18: Borehole deflection groove in roof exposed by development (Beamish, et al, 1985)

Rotary in-seam drilling was re-introduced to Tahmoor colliery to provide gas drainage through a soft clay dyke (Figure 2.19, Hungerford, 1985) after a fatal outburst in 1985 (Harvey and Singh, 1998). With the success of this project, the mine developed an in-seam rotary drilling program of 80 mm boreholes with 45 mm diameter rods. The smaller diameter drill rods operating in the 80 mm hole resulted in increased and varied lateral curvature making consistent coverage for effective drainage difficult (Hungerford, 2011).
The West Cliff colliery management worked with Longyear to develop and introduce the LMC55 drill rig in 1991 (Walsh, 1997) for underground in-seam drilling to cater for expanding widths of longwall blocks.

### 2.4.3 Drilling Patterns

Four modes of drilling and surveying were used to try to provide effective coverage of target drainage areas:

- Drill regularly spaced parallel holes across a longwall assuming they all remain straight or follow a consistent curvature to attempt to provide a consistent spacing (Figure 2.20, Tonegato, 1998).
• Drill fan patterns based on previous knowledge of borehole curvature (Figure 2.21, Kelly, 1983). The final position was defined when intersected by development mining and any variations to lateral deviation trends were noted.

• Drill fan patterns with each hole surveyed progressively with wireline single-shot surveying with vertical control through variations in rotational speed and feed rate. Additional holes are drilled in attempts to fill gaps produced by lateral curvature (Beamish et al., 1985)

• Drill fan patterns with each hole surveyed after drilling (Figure 2.22, Hungerford and Ren, 2011). Each subsequent hole in the pattern was planned to try and fill the gaps produced by deflections/curves of previous drilling.

![Figure 2.20: Parallel drilling borehole patterns (Tonegato, 1998)](image)

![Figure 2.21: Planned rotary fan pattern (Kelly, 1983)](image)
The first two methods did not provide confirmation of borehole location or adequate drainage coverage prior to mining. The surveyed holes, although providing prior knowledge of borehole location, required excessive drilling while producing inconsistent coverage (some gaps and some overlap).

2.5 EVOLUTION FROM ROTARY DRILLING TO DIRECTIONAL DRILLING, 1984-1994

2.5.1 Development of Directional Drilling in Australian Coal Mines

With technology development in the US, directional drilling technology had been reduced in size from that used regularly in the surface drilling industry for oil (Figure 2.23, Baker Hughes, 2011) and modified for use in the underground coal environment. Directional drilling using DHMs in Australian coal mines began as a NERDDC sponsored project (NERDP No.81/1075) with ACIRL drilling long 80 mm diameter in-seam boreholes and West Cliff collieries. Drilling utilised a 60.2 mm Dyna-dril DHM with Eastman single-shot surveying to complete boreholes to 825 m depth (Allan, 1984). Although papers from US operations were referenced, these were mostly theoretical (Thakur and Poundstone, 1980; Thakur, et al, 1988), so directional drilling practices were developed in Australian conditions through trial and progressively increased experience. With access to directional drilling technology, the prospects of drilling long in-seam boreholes to achieve effective gas drainage were investigated through both private and research development projects (Williams, et al, 1986).
AGL established Methane Drainage Pty Ltd (MDPL) in 1985 with the aim of establishing a directional drilling program of 1000 m in-seam boreholes at Tower colliery to commercially exploit coal seam gas reserves (Davis, 1986). The project employed a Fletcher drill rig with CHD rods and a 74 mm Slimdril DHM to drill 89 mm diameter boreholes in the Bulli seam. Surveying was undertaken with a Model 24 Cableless Survey System (CSS) manufactured by Geoscience Electronics Corporation (Williams, 1991). Although boreholes to depths beyond 800 m were achieved in the Bulli seam, drilling was not as successful in the Wongawilli seam with the loss of equipment due to bogging eventually leading to the termination of the project.

A research and development project (NERDDP No. 874) in co-operation with BHP, Appin colliery and ACIRL started in 1986 to investigate the viability of using directional drilling to produce long in-seam boreholes for gas drainage (Hungerford, et al, 1988a). Although DHMs had been used by ACIRL previously, the key aspect of this project was access to the Geoscience CSS from MDPL. This project completed the first in-seam borehole beyond 1000 m (Figure 2.24, Hungerford, et al, 1988a). Progressively increasing surging slowed penetration rates as the borehole depth increased. The in-seam borehole to 1005 m into a virgin area produced high gas flows over an extended period.
Cross-measure rotary drilled boreholes were used to capture gas released from underlying seams as the longwall passed (Kelly, 1983) to enhance gas capture and limit gas-outs on the longwall face. The concept of long-hole cross-measure/in-seam directional drilling was proposed (Figure 2.25, Williams, 1991) in an attempt to provide both pre-drainage and post-drainage of underlying seams.

The Balgownie seam was defined as the target seam (Hungerford, et al, 1988b) and successful cross-measure in-seam boreholes were completed to 923 m and 837 m.
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(Figure 2.26 and Figure 2.27, Hungerford, et al, 1988a). This cross-measure drilling was the first time borehole design was used successfully in vertical profile to assist drillers to “land” the borehole in the Balgownie seam (Hungerford, et al, 2013a). Borehole design was subsequently used in the lateral plane to plan the positioning of the second leg 40 m from and parallel to the first leg (Figure 2.27).

![Figure 2.26: Profile of cross-measure, in-seam borehole, Appin colliery (Hungerford, et al, 1988a)](image1)

![Figure 2.27: Plan of cross-measure in-seam boreholes, Appin colliery (Hungerford, et al, 1988a)](image2)

After the longwall face passed over the end of the cross-measure boreholes, high gas flows of around 180 l/sec were produced from the boreholes and no further “gas outs” were experienced on that longwall face. Subsequent cross-measure/in-seam programs at Appin, West Cliff and Tahmoor collieries had mixed results (Williams, 1991).
The achievements from the project developed experience in DHM drilling operational practices, established DHM steering parameters and improved PCD bit design. These were compiled to produce a long-hole drilling manual (Hungerford, 1989) which formed the basis for directional driller training in translating the technical successes of the project to mine operational practice (Hungerford, et al, 1990).

The availability of the Geoscience CSS ceased with the demise of MDPL (Williams, 1991). The Directional Drill Monitor (DDM) produced by Dupont/Conoco had been tested during the project but had limited success due to interference of the signal transmission due to in-hole gas flow noises.

2.5.2 Development of Directional Drilling for Exploration

Geologists realized the potential of in-seam directional drilling as an exploration tool to investigate future longwall domains (Walsh, 1991), structures ahead of mining (Ross, 1991) and undulating thick seams (Beamish, et al, 1991). Directional drilling (compared to rotary drilling) involved more expensive equipment at risk in the borehole and required higher levels of driller training and supervision to acquire the necessary skills (Hungerford, 1991; Ross, 1991). Although the benefits of directional drilling of long-holes for gas drainage had been established (Williams, 1991), the lack of the reliable electronic survey instruments meant directional drilling reverted back to operating with Eastman single-shot surveying. This meant drilling rates and performance would be severely limited compared to that with electronic surveying (Williams, 1991). Directional drilling in NSW was mostly confined to in-seam exploration and water management drilling projects by ACIRL with Eastman surveying.

Although directional drilling was regarded as expensive and not being able to match the drilling rates of rotary drilling, the cost benefits of long-hole drilling needed to be assessed in terms of the detail of information gained and its importance to mining (Williams and Hungerford, 1988). Given the financial penalties resulting from inadvertent intersections with structures around longwall blocks and the importance of defining areas in longer term planning, long-hole drilling in most situations, would prove highly cost beneficial. Long-hole drilling was the only method that directly tests the area prior to mining, hence providing a higher level of confidence and forewarning.
Although a depth of 1005 m had been achieved with the earliest NQ size configuration, penetration started surging beyond 60 m so penetration rate was progressively reduced as borehole depths increased to prevent stalling of the DHM (Hungerford, et al, 1988a). A 2-7/8” Accu-dril DHM was offered to the industry in 1992 through Asahi with the expectation of improved penetration rates and depth capacity (Walsh and Hungerford, 1993.). This unit had a non-magnetic, high-torque, low-speed 4-5 lobe motor section which offered improved performance (Hungerford, 1995). In an attempt to reduce in-hole friction, a 96.1 mm diameter PCD bit was introduced and the DHM fitted with a 1.25 degree bend for directional control. Surging (which had been attributed to in-hole friction) was greatly reduced and drilling rates improved. In 1993 and 1994, the first two boreholes drilled with this configuration achieved lengths of 1233 m and 1535 m at North Cliff colliery (Figure 2.28, Walsh and Hungerford, 1993). This combination of DHM bend size and bit diameter was also introduced to gas drainage operations and established as the standard for in-seam drilling in Australia and eventually the world.

![Figure 2.28: Profile of in-seam borehole NC94-1 (Walsh and Hungerford, 1993)](image)

With the achievement of in-seam borehole lengths beyond 1000 m, the specifications established for drill rig and support equipment (Hungerford and Thomson, 1996) were:

- 75 kW, 1000 volt hydraulic power unit to power the rig and water pump,
- Methane monitor and automatic trip of electrical supply,
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- 250 l/min @ 8 MPa high pressure water pump,
- Automatic rod handling,
- 135 kN thrust and pull,
- 1500 to 2000 Nm torque, NQ capacity rotation unit,
- Track mounting.

Subsequent drilling projects also showed depth capacity was limited by in-hole friction and surging (Hungerford, et al., 2012). Composite Heavy Duty (CHD) rod threads were preferred by drillers due to easy handling and resistance to damage. Analysis of thread strength (Gray and Daniel, 2000) showed these threads to be the strongest of the N sized threads in use.

In-seam directional drilling has been a proven form of exploration. Each exploration project has challenges specific to the environment and nature of the geological data gathering required. Before each project, the exploration goals are defined and the drilling planned and modified to achieve the desired outcome. These various applications of in-seam directional used for explorations are defined in Chapter 8.

2.5.3 Directional Drilling for Water Management

The ability to steer and position boreholes within a coal seam presented the opportunity to use directional drilling for water management purposes. Numerous applications were identified and utilised. These included in-seam drilling for water pumping at Ulan colliery (Allan, 1988) water drainage of seams with high water content ahead of development at Gordonstone mine (Hungerford, 1993), intersecting flooded old mines for water drainage/utilization (Thomson and Hungerford, 1993). Comprehensive standard safety and drilling operation procedures were developed to manage the challenges associated with intersecting water filled voids under potentially high pressure.

The fatal water in-rush which occurred at Gretley colliery in 1996 (Staunton, 1997) resulted from the position of an old flooded mine being recorded incorrectly. The mine was inadvertently intersected by development with high pressure water being injected into the development panel, resulting in the fatalities. From this, legislation has been introduced that required all underground coal mines to put systems in place to ensure such an event is not repeated (Hungerford and Ren, 2013). Inrush protection awareness
requires in-seam drilling ahead of any development where there may be a risk of inrush (McNaughton, 2002).

The applications of directional drilling utilised for water in-rush protection, drainage and management are detailed in Chapter 8.

### 2.5.4 Development of Directional Drilling for Gas Drainage

Since directional drilling rates could not match those usually associated with rotary drilling in environments conducive to rotary drilling, mines in NSW with gas drainage drilling operations opted to retain their rotary drilling programs. As an exception to the trend with rotary drilling, in 1990 Tower colliery converted to directional drilling for in-seam drilling with Eastman single-shot surveying (Benson, 1993) and from 1991 converted to electronic surveying while providing an operational environment for the development of the DDM (Knight, 1991).

Tahmoor, West Cliff, Appin and Metropolitan collieries began limited training programs to introduce directional drilling capabilities to their operations (Baker, 1994a).

The drilling conditions in Queensland underground coal mines are not as conducive to rotary drilling as in NSW where the bit tends to deflect off seam roof and floor to remain in-seam (Williams, et al, 1986). Coal strength and modulus were likely to prove an important indicator of drilling conditions (Williams and Hungerford, 1988), with Bowen Basin coal on average, having a significantly lower strength and modulus than Sydney Basin coals (Rawlings, 1985).

In 1989, ACIRL undertook a NERDDP funded gas drainage project at Moura No. 2 mine (Truong, et al, 1990a). The project involved the use of a Diamec 260 drill rig, CHD rods, 74 mm Slimdrill DHM, FMC 6-60 flush pump and DuPont (Australia) Ltd cableless electronic survey instrument. Boreholes to depths of 417 m were drilled to drain gas ahead of development. Although improvements were made with the survey instrument, the borehole had to be shut off to suppress the in-hole gas noise for each survey (Truong, et al, 1990b). Drillers from the surface contract drilling company (Strata Drilling) were trained in directional drilling to complete the research project. The mine acquired a Universal Drill Rigs (UDR) B15 drill rig to continue gas drainage drilling operations to 500 m at the completion of the research project. As ongoing
improvements were made to the DuPont instrument, the instrument was reintroduced to the Moura drilling operations.

When underground mining operations began at Central colliery, in-seam drilling for gas drainage became an integral part of the operations (Caffery, *et al*., 1992; Robertson, *et al*., 1995). A UDR B15 drill rig with BQ rods, Drillex DHM and Eastman survey instrument was acquired by the local drilling contractor (Blanch, 1994) for directional drilling to manage control in difficult drilling conditions which included a mid-seam shear zone. In-seam drilling expertise had been acquired through the previous open-cut high-wall drilling (Hungerford, 1988a) and involvement in the Moura project.

An ACARP funded project (C3075 – In-seam Drilling Project Co-ordination) was started in 1993 which provided a forum for the underground drillers and supervisors to discuss, compare and exchange ideas on the current drilling practices employed at each mine (Hanes, 1997b). The slow acceptance of directional drilling and the reluctance to convert to DHM drilling was due to higher cost, lower productivity and increased training requirements (Hanes, 1997a). The emphasis was on wanting improvements in accuracy and straightness with the rotary drilling (Hanes, 1998). Research was directed towards developing a rotary drill rig monitor (Danell and Hopkins, 1991) to attempt to better detect potential outburst zones (Sheldon, 1993). With the eventual acceptance of directional drilling, this project was discontinued even though it showed promise at detecting structurally disturbed coal (Hanes, 2002).

Numerous alternative survey instruments were tested through this period to try to verify the location of boreholes while limiting the delays and disruption to the drilling process. All NSW mines had reverted back to using the Eastman single-shot survey instrument where surveying of rotary drilled boreholes is required.

The research and development by Dupont (Knight, 1991) and AMT that had gone into refining the DDM-Upgrade survey instrument produced the DDM-MECCA, the Directional Drill Monitor using Modular Electrical Connected Cable Assembly (MECCA) installed in each rod (Hungerford, 1995). This unit was released at the end of 1994 and allowed accurate electronic surveying to depths beyond 1500 m without the time delays associated with wire-line surveying (MacCabe and Hellyer, 2013). The
DDM-MECCA survey system allowed progressive surveying of directional drilling boreholes with the following benefits:

- Very short survey times not affected by depth,
- Regular, short 6 m survey intervals to match 6 m sections of DHM orientations,
- Improved directional control,
- Increased drilling time,
- Improvements in drilling rates and
- Mines were able to design gas drainage drilling patterns to suit their environment and the abilities of their drillers.

This made directional drilling an option more easily adopted by gas drainage drilling departments to replace traditional rotary drilling.

Further development of the survey system was undertaken to utilise the power of the rig to overcome the need to recharge the Up-hole Unit (UHU) of the DDM-MECCA. This produced the Drill Guidance System (DGS) which also utilised MECCA installed in the drill rods to communicate with the down-hole instrument. The upgraded down-hole unit has the capacity to support additional geological sensors. Recent upgrades in the DGS have improved the signal depth capacity to beyond 1500 m. The DDM-MECCA had been regarded as the superior instrument in depth capacity with better signal quality in very long holes.

2.6 APPLICATION OF DIRECTIONAL DRILLING FOR GAS DRAINAGE

2.6.1 Adoption of Directional Drilling for Gas Drainage

Following the triple fatality outburst at South Bulli colliery in 1991 (Agrali, 1995), the Inspectorate introduced the concept of an Outburst Management Plan (OMP) (Tonegato, 1998). This required the mines to develop and implement outburst management plans but with no formal approval process. As seen in the notes from the ACARP sponsored drillers meetings (Hanes, 1993), rotary drilling continued as the main form of drilling for gas drainage for outburst control.

An outburst at West Cliff colliery in 1994 (Lama and Bodziony, 1996, Walsh, 1997, Walsh, 1999), which caused the fatality of the miner driver operating under “bomb
"squad" conditions resulted, in part, due to rotary drilled drainage holes deviating off-line without surveying being conducted to verify their location (Tonegato, 1998). The practice at the mine had been to assume that all boreholes were straight (Figure 2.20), despite the long held understanding that rotary holes tended to the right (Walsh, 1997). Previously drilled boreholes at several mines were surveyed to determine location and deviation trends (Roberts, 1994). At West Cliff colliery, the recovery and surveying of old rotary boreholes established just how variable the holes were. The actual experience was that holes deviated both left and right, often leaving large areas undrained. At Tahmoor colliery, boreholes were found to deviate significantly (Figure 2.22), not reach to the target drainage area or provide inadequate drainage coverage (Baker, 1994b; Hungerford, et al, 2013a).

Following the fatality at West Cliff colliery, the Inspectorate drafted an Outburst Management Guideline requiring an OMP to be developed at each gassy mine. It became imperative to use a drilling system that was both accurate and steerable, thus being able to position boreholes consistently. Directional drilling was the only form of drilling which could provide accurate placement of boreholes with surveying to verify their location. Such technology allowed for practical compliance with OMPs and lent itself to such coverage shown in Figure 2.29. This prompted all gassy coal mines to move towards adopting directional drilling practices for their gas drainage programs (Wynne, 2002).
Gas drainage drilling patterns in Figure 2.30 (Thomson, 1997) have been largely governed by drill rig access, efficiencies associated with site moves, driller and support management skills and borehole spacing required to provide adequate gas drainage. Regular parallel holes (Type A) as a layover from the rotary drilling patterns require multiple site moves so are no longer favoured. Straight holes originating from the same site (Type B) are preferred by some sites due to the benefits of single site and the drillers only having to drill straight holes. Multiple holes can be provided from a single standpipe through branching (Type C). Where site access is limited, long holes have been drilled parallel to proposed gate-road developments (Type D). Long exploration holes (Type E) have provided gas drainage from a large area over a long time.

![Figure 2.30: Typical drill pattern layouts for a longwall operation practicing gas drainage (Thomson, 1997)](image)

Although each mine has its characteristics of borehole layout design, mines progressively revised and modified their designs to provide better drainage coverage as driller skills and directional control improved (Wynne, 2002). An example of the progressive improvement in drilling techniques from 1994 to 2001 is well illustrated in Figure 2.31 (Wynne, 2002). The boreholes to the right were a combination of rotary drilling and directional drilling with Eastman single-shot surveying. As directional drilling and steering skills improved and technology advances produced electronic
surveying, drilling patterns became more refined and provided consistent drainage coverage.

![Diagram](image.png)

Figure 2.31: Progressively improved drilling patterns, Tahmoor colliery 1994 – 2001 (Wynne, 2002).

The standard has been established for the drilling of fan patterns across each longwall block from sites following development to provide drainage of the next gate-road development as well as the longwall block. The pattern adopted at each mine usually
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has some relation to the original rotary drilling patterns used. Appin and Metropolitan collieries prefer the simple drilling of straight fanned holes to provide the drainage coverage (Peace, 2006). The candelabra pattern of initial curved sections around to parallel sections equally spaced through the proposed gate-road panel has been established as an efficient pattern to provide consistent drainage at Tahmoor colliery and mines in Queensland.

2.6.2 Application of Directional Drilling for Gas Drainage in Queensland Coal Mines

With drilling conditions in Queensland underground mines not conducive to rotary drilling, directional drilling has been preferred for all in-seam drilling projects. With the current underground mines in Queensland being relatively new and not possessing established drilling crews from rotary drilling programs, mining companies have preferred to employ experienced drilling contractors to undertake the gas drainage drilling as required (Caffery, et al, 1992; Hanes, 2002). While full Underground In-Seam (UIS) drilling gas drainage projects have been undertaken recently at Grasstree mine (Figure 2.32) and North Goonyella mine (Allonby, 2002; Salisbury, 2005), the drilling conditions have been vastly different with high drilling rates achieved in good conditions at Grasstree mine while borehole stability problems limited drilling performance at North Goonyella mine.

Where surface and seam conditions permit, surface to in-seam (SIS) drilling for gas drainage has been employed such as at Moranbah North mine (Robertson, 2005a; Luckhurst-Smith and Williams, 2007) and Oaky North mine (Stanton, 2005). Both UIS and SIS drilling have been used to satisfy the gas drainage requirements at each mine.

Tight radius drilling from the surface has been employed prior to the initial development at Grasstree mine but has not been used since (Robertson, 2005b).

In-seam water jet drilling had been proposed in the late 1980’s and began with trials at German Creek mine (Kennerley, et al, 1991; Phillips, et al, 1991). Development has continued since (Liu, et al, 1998: Dunn, et al, 1999) but it has not been embraced as a viable technology suited to UIS drilling.
2.6.3 Managing Zones of Instability or Low Permeability

Directional drilling is not always successful at providing adequate gas drainage to satisfy the OMP requirements. Some areas have been found very difficult to drill (Hungerford and Ren, 2014), but cannot be avoided if gas drainage is the requirement. These areas are either impossible to drill with conventional means or collapse immediately after drilling. Scroll drilling has been used successfully at West Cliff colliery through boggy ground (Benson, 2004). Where instability is likely to cause the collapse of boreholes, perforated conduit is usually installed immediately after drilling is completed. Longer boreholes prone to collapse present greater adverse consequences to the drainage. Conduit is also installed in down-dip boreholes with high water make for dewatering.

Areas of the Bulli seam with the predominant seam gas being CO₂, have been found to have very low permeability which present problems to the drainage effects (Titheridge,
2003). Very high density drilling (Figure 2.33, Tonegato, 1998) usually has little effect on drainage rates when gas flow from boreholes is minimal. When very low permeability or instability has limited drainage in an area and/or the area is regarded as an outburst risk through faulting, either grunching (Borg, 2014) or remote mining (Henderson, et al, 2008) has been used successfully in developing through these areas without putting lives at risk from outburst.

Figure 2.33: High density drilling in a low permeability zone (Tonegato, 1998)

As adverse drilling and/or gas drainage environments are intersected and defined, the standard patterns have been modified to suit the specific requirements. For example, the drilling patterns can be modified to cater for limited access or to cover areas not able to be accessed because of faulting.

The applications of directional drilling for gas drainage are detailed in Chapter 7.

2.7 INCREASING BOREHOLE DEPTH CAPACITY

Drilling departments have experimented with rotary slide drilling to improve performance (Eade, 2002). This technology was employed as a development project to overcome in-hole friction to attempt to drill 2000 m long holes at Metropolitan colliery to cover longwall development panels with limited access. All boreholes surpassed the previous record for long-hole in-seam drilling with the longest borehole reaching 2151 m in length (Hungerford and Green, 2016). Cross-measure in-seam drilling has
been trialed on various occasions with inconsistent results (Williams, 1991). The concept of drilling cross-measure boreholes to and along horizons above the longwall for post-drainage has been proposed and instigated with success (Brown and Hobden, 2014). This is a new concept in Australia with the positioning of the boreholes crucial to successful application. The drilling challenges involve achieving good penetration rates in stone and reaming to larger diameters (Hungerford, et al, 2013a).

The application of a combination of rotary/slide drilling to achieve boreholes to and beyond 2000 m in depth is detailed in Chapter 9.

### 2.8 INTRODUCTION OF DIRECTIONAL DRILLING TECHNOLOGY TO OVERSEAS MINING INDUSTRIES

The directional drilling technology developed in Australia has been successfully introduced to the coal mining industry in other countries. It was introduced to New Zealand in the late 1980’s initially for in-seam exploration to define the seam profile in the thick undulating seam at Huntly West mine (Beamish, et al, 1991) before also being utilised for gas drainage. Directional drilling was being used for seam floor profile definition for the water jet mining operations before the explosion at the Pike River mine (Robbins, 2011).

In 1998, directional drilling was introduced to Kushiro colliery in Japan for exploration drilling (Hungerford, 1998a). The mine has since closed. Also in 1998, directional drilling was introduced to in-seam exploration at New Denmark colliery in South Africa to locate and define igneous intrusions (Hungerford, 1998b).

The introduction of directional drilling technology to the Chinese coal industry started unsuccessfully with a United Nations sponsored project in 1997-98. Most coal seams in China are very fractured and unsuitable for supporting in-seam drilling. For drilling to be successful in China, suitably competent coal seams had to be targeted. The Series 1000 drill rig was developed by Valley Longwall Drilling (VLD) to compliment the equipment package for long-hole in-seam directional drilling. The technology was successfully introduced to Daning mine in 2003 (Hungerford, et al, 2005) and Sihe mine (Hungerford, et al, 2006) before being accepted by the industry. In 2007, a demonstration project at Baijigou mine successfully drilled and drained a longwall block by drilling cross-measure/in-seam from a roadway in the underlying stone strata.
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(Hungerford, 2007). Directional drilling systems including all hardware, training and support have been sold to numerous coal mines in China which have been audited and found to have suitable drilling environment and safety standards.

Directional drilling has now been introduced to Russia in the coal industry of the Kus bass area of Siberia and the underground oil drilling industry of Lukoil in Ughta, Northern Russia. Directional drilling systems have been sold to mines in Kazakhstan, Poland and Ukraine with the main emphasis on stone drilling above the seam for post-drainage due to the fractured nature of the coal not being able to support in-seam drilling.

2.9 SUMMARY

In-seam drilling for gas drainage was identified in the 1970’s as the most effective means to reduce the risk of outburst. Rotary drilling programs were progressively introduced at coal mines as higher gas content coal was encountered. Although the technology was being developed in both the USA and Australia, there was little interaction between the coal industries in each country. The gas drainage technology was developed in parallel to suit the particular environments of each country’s coal seams.

The introduction of directional drilling in the mid-1980’s offered more accurate positioning of boreholes but was limited by the state of the technology of the time. DHM’s were low powered, the survey systems available limited drilling rates and technical support was not available since this technology had been imported from the USA. Mines were slow to accept directional drilling and reluctant to convert from rotary drilling due to higher cost, lower productivity and increased training requirements associated with directional drilling.

The unexpected fatal outburst in 1991 at South Bulli colliery and the fatal outburst in 1994 at West Cliff colliery which was not managed by the precautions in place at the time prompted the introduction of OMPs. From the drilling aspect, these required that all boreholes be positioned as planned and be surveyed to verify their location. Directional drilling was the only form of drilling which could provide accurate placement of boreholes with surveying to verify their location. Such technology allowed for practical compliance with OMPs.
As directional drilling was slowly being accepted by gas drainage drilling departments, the DDM electronic survey instrument was being refined in operational environments at Tower and Moura collieries. The release of the DDM-MECCA coincided with the increase in directional drilling. The fact this system made the introduction of the technology easier through training, provided better control and allowed much improved drilling performance made the additional capital expenditure and conversion from rotary drilling easier to justify and accept. Directional drilling then became an integral part of each mine’s OMP.

A much higher volume of in-seam drilling has been required in Australian coal mines to manage outburst risk than in mines in the USA. This meant there has been a greater need for technical development and training to improve drilling performance. This was evident in the development of the DDM survey instrument which transformed directional drilling. Due to limited research support, most development through the 1990’s was either by contract drilling companies or trial and error within in-house drilling departments at the various mines. The industry gained an extended use of experienced directional drillers through mentoring and experience based training. The Australian underground coal industry has been dependent on the supply of DHMs from the USA but has been more interactive with suppliers in solving problems with DHMs as they occur.
CHAPTER THREE – DRILL RIGS, RODS AND DOWN-HOLE MOTORS

3.1 INTRODUCTION

The chapter describes the range of drill rigs used from the early days of rotary drilling to the latest directional drilling for gas drainage. As drilling depths increased, drill rigs used in the hard rock mining industry had to be imported and modified for use in underground coal mines. The characteristics of these imported drill rigs were used as the basis for designing and building larger and more complete drilling systems to meet the requirements of drilling in-seam boreholes. The development of drill rigs is described.

Drill rods are a crucial component of any drilling project. The size and style of the drill rod varies with different drilling application and borehole depth. The drill rods utilised during the development of in-seam directional drilling are identified and their characteristics explained.

In the early days of directional drilling, drilling projects had to use whatever DHMs were available. Smaller capacity DHMs with high rotational speeds were not really suitable for coal drilling but provided the initial knowledge of DHM operations. The introduction from the mid 1980’s of DHMs with improved technology is explained. The improved torque capacities available and the rotational speeds of each are identified. The design and construction characteristics of these DHMs are discussed.

3.2 DRILL RIGS

3.2.1 Development of Drill Rigs for Rotary Drilling Applications

Early drills were limited by the technology and the state of drilling knowledge of the time. Hand-held drills had compressed-air power for the rotation with the operator having to provide the thrust. Hand-held power borers took the form of a hand-held air drill (A, Figure 3.1, Cervik, et al, 1977) and the Victor air borer (B, Figure 3.1) using 34.9 mm diameter (EW) rods and drag bits to drill 43 to 47 mm diameter holes. The driller had to provide the thrust to overcome in-hole friction and achieve penetration while anchoring the drill to resist the reactive torque created by the drilling action. Flushing of cuttings was by either air or water circulation. This drilling was usually limited to depths of 30 m.
Borehole depths were increased with "home-made" modifications of drills and smaller hand-held power borers being mounted on frames with air-legs, using compressed air power providing the thrust. This drilling was restricted to depths of 50 to 100 m.

Early electro-hydraulic drill rigs were usually heavy units like the Mindrill, (Figure 3.2, Hungerford, 2011) and the USBM post-mounted drill rig of the 1970’s (Figure 3.2, Cervik, et al, 1977) which used separate hydraulic pump circuits to provide independent control of rotation and feed. The rotation unit clamp was a manually operated unit tightened and released by hand, similar to the chuck of a lathe. This operation was slow and did not lend itself to rapid rod handling. The rod joints were either tightened/separated by hand with pipe wrenches or chuck assisted with the other side of the joint secured by a pipe wrench wedged against the frame of the rig.

These drill rigs were operated with either rods fed through the chuck or as a “top-end drive” with a “Kelly rod” fixed through the chuck. Top-end-drive reduced the time delay associated with using the manual chuck but was only possible if the feed length was longer than that of the rods being used.

Figure 3.1: (A) Hand-held air drill (Cervik, et al, 1977); (B) Victor air borer

Figure 3.2: Mindrill (Hungerford, 2011); Electro-hydraulic drill (Cervik, et al, 1977)
As part of the development of in-seam rotary drilling, the USBM had Longyear build a horizontal drill (Figure 3.3 and Figure 3.4, Cervik, et al, 1975) which provided independent control over rotation and feed to allow variations in drilling parameters to provide some form of directional control. This unit was a top-end drive running the rods through a guide bush at the front requiring manual handling of the rod joints with pipe wrenches.

Figure 3.3: Longyear horizontal drill (Cervik, et al, 1975)

Figure 3.4: Power unit for Longyear horizontal drill (Cervik, et al, 1975)

Compressed-air operated top-drive rigs from surface applications were used underground for drilling shorter boreholes. These units, such as the track-mounted Atlas Copco BBR 601, used at Collinsville for preliminary drilling programs (Hungerford, 1981), drilled boreholes up to 100 mm in diameter but were limited to depths of 60 m with no monitoring of borehole deviations.

3.2.2 Modification of Hard-rock Drill Rigs for Rotary Drilling Applications

The development of high volume wire-line core drilling in the hard rock industry produced drill rigs with fast feed rates and a rotation unit designed for wire-line coring rods. The evolution of the hydraulically powered drill rigs produced units with a hydraulic power pack driven by compressed air. Hydraulic power was provided by the power pack to operate the drill rig with hydraulic oil directed through a valve bank in the control panel to control all functions of the drill rig (Figure 3.5, Beamish, et al,
The chuck and rod clamp system allowed fast automatic rod handling from the control panel.

The original Craelius Diamec 250 drill rig imported from Sweden (Figure 3.6) was mounted on posts secured between the roof and floor and had compressed air powered hydraulic power units. These rigs were eventually mounted on tracks to provide greater mobility and ease of setting up (Figure 3.6). Automatic rod handling was a feature that greatly increased rod handling and running speeds.

With the requirement for longer boreholes, higher capacity drill rigs became available with the introduction of electric powered hydraulic power packs. The sled mounted Diamec 251 drill rig (Beamish, et al., 1985) and the rubber-tyred, jack mounted Acker Big John drill rig (Figure 3.7, Hebblewhite, et al., 1982) were introduced in the early 1980’s. Each provided the independent control of feed and rotation required for directional control with rotary drilling.
The introduction of the Kempe U4-450 drill rig (Kelly, 1983; Beamish, et al, 1985) originally from South Africa, saw a robust rig more specifically designed for coal drilling. This rig had increased power through electric powered hydraulics with a depth capacity greater than 500 m.

The modified rotary hard-rock drill rigs used are listed in Table 3.1 with their specifications and nominal depth capacities:

**Table 3.1: Modified hard-rock drill rigs**

<table>
<thead>
<tr>
<th>Drill Rig</th>
<th>Power Supply</th>
<th>Mount</th>
<th>Rod Capacity</th>
<th>Push/Pull Capacity (kN)</th>
<th>Nominal Depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Diamec 250</td>
<td>Air/Hyd</td>
<td>Track</td>
<td>BQ/BCQ</td>
<td>32 / 24</td>
<td>250</td>
</tr>
<tr>
<td>Diamec 251</td>
<td>Elect/Hyd</td>
<td>Sled</td>
<td>BQ</td>
<td>43 / 33</td>
<td>350</td>
</tr>
<tr>
<td>Acker Big John</td>
<td>Elect/Hyd</td>
<td>Sled</td>
<td>BQ</td>
<td>45 / 45</td>
<td>500+</td>
</tr>
<tr>
<td>Kempe U4-450</td>
<td>Elect/Hyd</td>
<td>Sled</td>
<td>BQ</td>
<td>31 / 52</td>
<td>500+</td>
</tr>
<tr>
<td>Cram ProRAM</td>
<td>Air/Hyd</td>
<td>Sled</td>
<td>AW</td>
<td>11 / 11</td>
<td>250</td>
</tr>
<tr>
<td>Air Tracks</td>
<td>Air/Hyd</td>
<td>Track</td>
<td>AW/BW</td>
<td>-</td>
<td>100</td>
</tr>
<tr>
<td>Mindrill</td>
<td>Air/Hyd</td>
<td>Sled</td>
<td>AW</td>
<td>-</td>
<td>30</td>
</tr>
<tr>
<td>S&amp;K (Tahmoor)</td>
<td>Elect/Hyd</td>
<td>Track</td>
<td>AWJ</td>
<td>-</td>
<td>300</td>
</tr>
</tbody>
</table>
3.2.3 Reconfiguration of Hard-rock Drill Rigs for Directional Drilling Applications

In producing drill rigs for underground drilling, the trend continued with higher capacity drill rigs from the hard-rock industry being rebuilt/reconfigured with modifications to suit the underground coal environment (Hungerford, 1995). These modifications included:

- Replacement of all aluminium parts with steel equivalents.
- Installing flameproof electric motors, starters, light fittings and emergency stop buttons.
- 250 l/min @ 10 MPa high pressure water pump.
- Facility to lock the rotation for directional drilling.
- HQ rod capacity jaws.
- Using hydraulic oil approved for underground use.
- Mounting on robust sleds or tracks which could be secured between the seam roof and floor.

The components of Boyles B15 underground coring rigs were rebuilt with both track (Figure 3.8, MacCabe and Hellyer, 2013) and sled mounted (Figure 3.8) configurations for directional drilling. These rigs had a two speed gear box in the rotation unit which allowed rotational speeds up to 1200 rpm. Due to low rotational speeds used for rotary drilling in coal or directional drilling, the high speed setting was never utilised. They were the first rigs to introduce the rotation lock on the shaft drive of the rotation motor.

Figure 3.8: Track mounted Boyles B15 drill rig (MacCabe and Hellyer, 2013)
The B20 drill rig has a larger capacity than the B15 with similar characteristics (Figure 3.9). The B20 utilised a multi-stage direct-drive hydraulic cylinder in the feed frame.

Cram rebuilt the Diamec 262 drill rig with all components included on a track mounted chassis (Ramtrack, Figure 3.10) to provide an intermediate depth capacity drill rig suited for high volume drilling for gas drainage.

The thrust and nominal depth capacities of each of the reconfigured drill rigs are listed in Table 3.2.
The major exception to the trend towards more robust electric powered drill rigs was the development of the Cram ProRAM which was a light, easily transported drill rig powered by compressed air. This rig utilised carbon fibre technology and was an ideal rig for rapid rotary drilling of short to medium length in-seam and cross-measure boreholes for gas drainage or in-seam boreholes for gas content coring. The ProRAM was available mounted on carbon fibre, hydraulically powered stingers anchored between roof and floor similar to the Diamec 250 in Figure 3.6. It was eventually made available mounted on sled or on tracks as shown in Figure 3.11 (Hungerford, 2011) and used 43.7 mm diameter (AW) rods to rotary drill 65 mm diameter boreholes. It became an essential drill rig being utilised for gas content coring to satisfy “approval to mine” requirements of OMPs.

Table 3.2: Hard-rock drill rigs modified for rotary and directional drilling

<table>
<thead>
<tr>
<th>Drill Rig</th>
<th>Mounting</th>
<th>Rod Capacity</th>
<th>Push/Pull Capacity (kN)</th>
<th>Nominal Depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Boyles B15</td>
<td>Sled/Track</td>
<td>NQ/HQ</td>
<td>73 / 73</td>
<td>800</td>
</tr>
<tr>
<td>Boyles B20</td>
<td>Sled/Track</td>
<td>NQ/HQ</td>
<td>89 / 89</td>
<td>1200</td>
</tr>
<tr>
<td>Cram Ramtrack (Diamec 262)</td>
<td>Sled/Track</td>
<td>BQ/NQ</td>
<td>48 / 48</td>
<td>600</td>
</tr>
</tbody>
</table>

Figure 3.11: Track-mounted Pro-Ram drill rig (Hungerford, 2011)
3.2.4 Development of Drill Rigs for Directional Drilling Applications

Through this period, drill rig manufacturing companies started working with sections of the coal industry to produce drill rigs suited to the drilling applications required and the environment. These rigs were generally more robust and configured for higher torque and lower rotational speeds for in-seam rotary drilling and directional drilling. Track mounting was supplied for high repetition, cross-block drilling to provide more manoeuvrability for site moves while sled mounting was provided for bigger capacity rigs to be employed in long-hole drilling which required a more secure anchorage. These rigs were produced without aluminium parts and included flameproof electric motor, starter, lighting and emergency stops.

The Fletcher LHD rig was a large capacity drill rig produced in the USA specifically for drilling operations in coal (Figure 3.12). This drill rig was originally imported into Australia in 1985 by MDPL with the aim of establishing a directional drilling program of 1000 m in-seam boreholes at Tower Colliery (Davis, 1986). The unit was rubber-tyred with a separate hydraulic power pack tender which included flameproof electrics, hydraulic pumps and oil reservoir. The high pressure water pump was originally a separate item. The large size of the unit meant that it had to be sited at intersections, very deep cut-outs or at the end of roadways.

![Figure 3.12: Fletcher LHD drill rig](image)

Two companies developed and produced drill rigs designed specifically for the coal environment with the added benefit of being convertible to hard-rock environments. Longyear worked with West Cliff colliery to produce the track mounted LMC55 drill rig for high volume, cross-block rotary drilling (Walsh, 1997). Longyear subsequently worked with ACIRL to produce a sled mounted LMC75 drill rig for long-hole in-seam directional drilling (Walsh and Hungerford, 1993). They also produced a track mounted
version of the LMC75 drill rig with the feed frame supported on an extension arm. Kempe produced the sled mounted K200 and K200C drill rigs for BHP for cross-block and mid to long-hole directional drilling (Benson, 1993). The thrust and nominal depth capacities of each of these directional drill rigs are listed in Table 3.3.

<table>
<thead>
<tr>
<th>Drill Rig</th>
<th>Mounting</th>
<th>Rod Capacity</th>
<th>Push/Pull Capacity (kN)</th>
<th>Nominal Depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Longyear LMC55</td>
<td>Track</td>
<td>BQ/NQ</td>
<td>42 / 40</td>
<td>500</td>
</tr>
<tr>
<td>Longyear LMC75</td>
<td>Sled/Track</td>
<td>NQ/HQ</td>
<td>130 / 130</td>
<td>1500</td>
</tr>
<tr>
<td>Fletcher LHD</td>
<td>Tyred</td>
<td>NQ</td>
<td>165 / 220</td>
<td>1500+</td>
</tr>
<tr>
<td>Kempe K200</td>
<td>Sled</td>
<td>BQ/NQ/HQ</td>
<td>74 / 62</td>
<td>1000</td>
</tr>
<tr>
<td>Kempe K200C</td>
<td>Sled/Track</td>
<td>BQ/NQ/HQ</td>
<td>119 / 90</td>
<td>1000+</td>
</tr>
</tbody>
</table>

With the success of achieving in-seam boreholes to 1500 m with the LMC75, the following equipment specifications for long-hole directional drilling were established as a recommended requirement (Thomson and Hungerford, 1996):

- 75 kW, 1000V hydraulic power unit to power the rig and the water pump,
- 250 l/min @ 10 MPa high pressure water pump,
- 135 kN thrust and pull,
- 1500 to 2000 Nm torque, NQ capacity rotation unit.

### 3.2.5 Current Status of Directional Drill Rod Design

Boart Longyear have continued development on their LMC series drill rigs and produced an upgraded LMC90 modular drill rig, and a track mounted LMC90 unit with the feed frame mounted along the length of the chassis (Figure 3.13).
Valley Longwall International (VLI) entered the market to supply directional drilling systems into the Chinese underground coal industry. The track mounted Series 1000 drill rig (Figure 3.14) was designed using what was regarded as the preferred technology from several of the drill rigs in operation. The slide bars of the Kempe drill rigs were adopted for the feed frame due to their low wear and maintenance requirements. The rotation unit and rod clamp from the Boyles B20 were copied and upgraded. The hydraulic system of the Boyles B20 was adopted but with a pilot operated control panel duplicating the layout of the B20 direct flow control panel. A standard high pressure triplex water pump with a capacity of 250 l/min at 7.0 MPa was included in the package.

The result was a compact, well anchored, robust and reliable track mounted drill rig. A thrust capacity of 140 kN ensured the Series 1000 drill rig was capable of drilling in-seam boreholes to depths beyond 1500 m. The success of the drill rig in China (Hungerford, et al, 2005) has been verified by the number of copies being produced.
Modular sled mounted (Figure 3.15) and rail mounted versions of the Series 1000 have been produced for specific applications where site access is limited. The thrust capacity has been reduced to 104.6 kN for use with the shorter feed frames available.

In recent years, Australian Drilling Services (ADS) have produced the ADS 2338 in-seam directional drill rig. As with the Series 1000, the design was based on utilising
proven features of drill rigs already in operation. This unit has achieved high volume drilling to depths beyond 500 m but is still in development with feed frame stability problems when high thrust loading is experienced.

3.2.6 Integrated Gas Monitoring

Originally, any gas monitoring was by a combination of CO$_2$ Draeger tubes, hand-held CH$_4$ monitors or Automatic Firedamp Detectors (AFDs) hung at the site. Electrical protection through integrated CH$_4$ monitoring was not mandatory so hand held monitors or general body gas monitors were only used in specific circumstances.

As in-seam drilling for gas drainage became an established practice, control and monitoring of gas became critical to maintaining a safe environment at the drill site. Integrated methane monitoring systems (Figure 3.16) are now mandatory to ensure electrical power is disconnected if methane levels reach specified limits with both visual and audible alarms ensuring workers on site are aware of any developing threats. The sensor is provided with an extended lead so it can be positioned in the “general air body” above the electric components of the drill rig. The system is calibrated on site on a weekly basis and the sniffer head must be calibrated by a statutory body at intervals not greater than six monthly.

![Image of integrated methane monitoring system](image)

**Figure 3.16: Integrated methane monitoring system**
3.2.7 Power Pack

The power pack is either a free standing unit in a modular drill rig (Figure 3.5) or incorporated in the design of a track mounted unit (Figure 3.10). Power is supplied through a flameproof enclosure to a flameproof electric motor; usually of 1000 volt power. The capacity of the electric motor has been increased to 90 kW to manage the increased load requirements with larger borehole depths and more functions in the hydraulic system.

With faster drilling rates and consecutive shifts being drilled, oil cooling is necessary. Kempe introduced oil cooling with radiator and fan to maintain the oil temperature within limits relative to the ambient temperature. The established standard is to divert a portion (approximately 10 l/min) of the incoming water for flushing through an oil cooler (Figure 3.17) to the waste water system. Low pressure return hydraulic oil is passed through the oil cooler before being returned to the oil reservoir.

![Figure 3.17: Water cooled heat transfer oil cooler](image)

3.2.8 Control Panel

Control panels were originally relatively small and designed to divert hydraulic oil flow to the various drill rig components (Figure 3.5 and Figure 3.8) in what is termed direct flow control. All oil flow is directed from the power unit to the control panel and diverted from there to the components of the drill rig as required. When the capacities of the drill rigs were smaller, the oil flows were manageable at the control panel. Hydraulic systems were relatively simple (Figure 3.18, DB Craelius, 1988). As thrust and torque capacities of the drill rigs have increased, the hydraulic oil quantity has increased to provide that increase in capacity. To keep the size of control panels manageable and
improve hydraulic efficiency, pilot systems were introduced to activate a larger valve bank conveniently located nearby. The Longyear control panel (Figure 3.19) is connected with one bundle of pilot hoses to the power unit, one bundle to the feed frame and can be free standing in a convenient location near the drill rig.

Figure 3.18: Hydraulic circuit with direct control (modified from Craelius, 1988)

Figure 3.19: Longyear control panel
The control panel of the VLI Series 1000 drill rig (Figure 3.20) was modelled on the Boyles B20 direct flow control panel (Figure 3.8) but with pilot valves replacing the direct flow valves. Hydraulic diagrams have become more extensive and complicated (Figure 3.21, VLI, 2009).
3.2.9 Feed Frame

The feed frame of the drill rig has several functions in the operation of the drill. It is an anchored frame through which the feed thrust is transmitted to the rotation unit and through to the drill rods, rotation torque can be transmitted to the drill rods and the rotation unit can travel along the feed frame to feed the rods in and out of a borehole for drilling and rod handling.

The thrust is transmitted from a hydraulic cylinder mounted within the feed frame to a carriage (on which the rotation unit is mounted) which slides along the length of the feed frame. The load transmission is either direct from the cylinder or via a chain drive.

The most common system is a chain drive connected between the frame, hydraulic cylinder and carriage (Figure 3.22). To provide travel over the full length of the feed frame, the chain pulley design provides the carriage with twice the travel distance of the hydraulic cylinder. The thrust provided to the carriage is half that provided by the hydraulic cylinder.

![Figure 3.22: Feed frame with chain drive](image)

The direct drive system has an advantage of reduced moving parts, service requirements and potential failure points. To provide the full travel of the carriage, a multi-stage hydraulic cylinder is utilised with a lug/bracket connection to the carriage (Figure 3.23). This offers some movement between the cylinder and carriage to allow for wear and flexing under load. To prevent hard impact at the end of each stroke, internal cushioning is provided to progressively slow the feed rate to zero as the cylinder reached the end of the stroke. The multiple-stage hydraulic feed cylinder in the Boyles B20 drill rig was supplied without internal cushioning so rubber pads were fitted at each end of the cylinder travel to provide the cushioning.
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When a conventional cylinder is used as a direct drive, the cylinder protrudes beyond the end of the feed frame when extended. Protection/confine ment is required for the cylinder in its extended position to avoid possible damage and injury. The Longyear LMC55 has a single direct drive cylinder which extends into an extension added to the back of the feed frame to provide the full stroke length.

3.2.10 Carriage

The carriage which holds the rotation unit has evolved in size and slide connection with the feed frame as the load requirements have increased. Early slides as used on the Diamec 250, Diamec 251 and Boyles B15 drill rigs only required nylon bushes. The Diamec 262 used internal “V” block slide bushes on internally facing channel slides.

Internally mounted roller bearings were used on the B20 drill rigs while externally mounted roller bearings were use on the Longyear LMC75. Longyear then designed large externally positioned “V” channels for the slides with the upgrading to their LMC90 drill rigs (Figure 3.13).

Kempe used the hard chromed slide bar system with their U4-450 drill rigs and continued that system with their K200 drill rigs. This system was adopted in the designs of the Series 1000 drill rig (Figure 3.14) and the ADS 2338 drill rig.

The large and heavy carriage of the Fletcher LHD (Figure 3.12) has hardened steel plates welded under the carriage. These plates slide on hardened steel plates welded to the top of the rectangular section of the feed frame.
3.2.11 Rotation Lock

Initial directional drilling operations had identified the backwards rotation of the drill string due to the torque of the DHM. To prevent the orientation of the DHM changing during drilling, the rotation had to be locked. The first project employed the use of pipe tongs during drilling until in-hole friction was adequate to prevent the backwards rotation. As safety attitudes changed and drilling efficiency improved, this was not an acceptable practice and rotation locks had to be designed into the drill rig design.

The first chuck lock employed on a Diamec drill rig was in the form of a pin or key wedged against the carriage (Figure 3.24). After rotating the drill string to the correct orientation, the chuck jaws were released, a pin inserted into a hole in the side of the chuck. The chuck was rotated backwards to wedge the pin against the carriage before the chuck was re-engaged to start drilling.

![Figure 3.24: Manual lock of the Diamec chuck](image)

This system was eventually replaced when Cram incorporated a toothed wheel to the back of the rotation spindle which allowed a hydraulically operated pin to lock the rotation (Figure 3.25). This was relatively weak and inadvertent hydraulic rotation of the chuck usually snapped the pin. The pin was easily replaced to allow drilling to continue.
The Fletcher LHD rig had a large carriage which allowed an additional rod clamp to be installed on the carriage behind the rotation unit to provide the locking of the rotation.

Kempe and Longyear successfully added chuck clamps to their established rotation units. Kempe added opposed hydraulically operated external clamps which clamped the chuck. Longyear added a toothed ring to the outside of their chuck and enclosed it with a ring fitted with a hydraulically operated pin to lock the chuck (Figure 3.26).
The established design taken from the Boyles drill rig design is to install a hydraulically operated clamp between the rotation motor and the gear box of the rotation unit (Figure 3.27). The rotation lock clamps the drive shaft and prevents rotation. The carriage design has to cater for the altered weight distribution with the rotation motor moved back relative to the rotation unit.

![Figure 3.27: Rotation lock incorporated between hydraulic motor and rotation unit](image)

### 3.2.12 Water Pump

Most early rotary drilling was able to make do with water flushing by the mine supply at mains pressure although high pressure water pump flushing was becoming more prevalent to improve flushing capabilities in attempts to prevent bogging of rods. These pumps were stand-alone units driven by an electric motor through a direct drive flexible coupling (Figure 3.28, Beamish, et al, 1985) providing a fixed flow rate. To provide variable water flow rates, pumps were driven through a gear box and chain drive from an electric motor (Figure 3.29, Hungerford, 1988b). Both systems drew the water from a tank or dam storage.
The introduction of directional drilling required the use of constant displacement pumps capable of delivering the required water flow and pressure to run DHMs as well as providing the means for flushing cuttings. Bean triplex pumps were the pumps of choice with relatively constant flow delivered from the combination of three pistons (Figure 3.30).
Figure 3.30: Combined flow from three cylinders of a triplex pump

Constant displacement pumps are preferred as they deliver flow at a relatively constant rate with variations in load or flow restriction indicated as variations in water pressure. This aspect is utilised to monitor the load on the DHM created when penetrating the strata, the progressive increase in flow resistance with increased drill string length and any resistance developing due to in-hole blockages.

Good water quality is preferred so disc valves can be used in the valve block (or “wet end”) for a higher flow rating than that available with ball valves. Ball valves are preferred if high quantities of solids are present in suspension. The cylinders are ceramic lined to resist wear and to avoid corrosion. High pressure rated cast iron valve blocks are used in preference to aluminium to suit underground coal mine requirements.

Triplex pumps are now integrated into the physical and hydraulic design of drill rigs for directional drilling (Figure 3.31). Hydraulic systems have the capacity to run the water pump, preferably on a separate circuit, in addition to all the drilling and tramming functions. The system now incorporates a hydraulic motor driven by the hydraulics of the rig to run the water pump via a direct drive coupling. The hydraulic oil flow and pressure are matched with the specifications of the hydraulic motor capable of running the triplex pump to its maximum rated capacity.
An accumulator is incorporated on the pressure side of the water circuit to cushion variations in water pressure created by the piston action. The system is protected from overload by a pressure relief valve usually set at the specified maximum pressure rating of the pump. With the hydraulic drive, the hydraulic drive system can be set to de-stroke at a prescribed pressure rather than on the relief valve setting.

Water pump operation is now controlled from the control panel with the flow directed past the control panel (Figure 3.19 and Figure 3.20) to allow water flow and pressure to be monitored. If direct mine pressure is supplied, a valve included in the high pressure line is used to shut water flow off. A diversion valve is included beyond that point to dump any water pressure remaining in the hose to allow the removal of the water swivel.
3.3 DRILL RODS

3.3.1 Drill Rod Usage

Drilling operations have used a variety of drill rods with the size and thread type determined by the drilling equipment, borehole depth and application. Details of the various rods and thread types used in the coal industry are shown in Table 3.4 (Sandvik, 2016).

Table 3.4: Drill rod dimensions (Sandvik, 2016)

<table>
<thead>
<tr>
<th>Rod Series</th>
<th>Rod OD mm</th>
<th>Rod ID mm</th>
<th>Coupling ID mm</th>
<th>Thread Pitch Thread/inch</th>
</tr>
</thead>
<tbody>
<tr>
<td>EW</td>
<td>34.9</td>
<td>23.8</td>
<td>11.1</td>
<td>3.0</td>
</tr>
<tr>
<td>AW</td>
<td>43.7</td>
<td>31.0</td>
<td>15.9</td>
<td>3.0</td>
</tr>
<tr>
<td>BW</td>
<td>54.0</td>
<td>44.5</td>
<td>19.0</td>
<td>3.0</td>
</tr>
<tr>
<td>NW</td>
<td>66.7</td>
<td>57.2</td>
<td>32.9</td>
<td>3.0</td>
</tr>
<tr>
<td>AQ</td>
<td>44.5</td>
<td>34.9</td>
<td>34.9</td>
<td>3.0</td>
</tr>
<tr>
<td>BQ</td>
<td>55.6</td>
<td>46.1</td>
<td>46.1</td>
<td>3.0</td>
</tr>
<tr>
<td>NQ</td>
<td>69.9</td>
<td>60.3</td>
<td>60.3</td>
<td>3.0</td>
</tr>
<tr>
<td>HQ</td>
<td>88.9</td>
<td>77.8</td>
<td>77.8</td>
<td>3.0</td>
</tr>
<tr>
<td>PQ</td>
<td>117.5</td>
<td>103.2</td>
<td>103.2</td>
<td>3.0</td>
</tr>
<tr>
<td>NT</td>
<td>69.9</td>
<td>60.3</td>
<td>60.3</td>
<td>2.0</td>
</tr>
<tr>
<td>CHD76</td>
<td>69.9</td>
<td>60.3</td>
<td>54.8</td>
<td>2.5</td>
</tr>
<tr>
<td>BWJ</td>
<td>54.0</td>
<td>44.5</td>
<td>19.0</td>
<td>5.0</td>
</tr>
<tr>
<td>NWJ</td>
<td>66.6</td>
<td>57.1</td>
<td>29.0</td>
<td>4.0</td>
</tr>
<tr>
<td>HW Casing</td>
<td>114.3</td>
<td>101.6</td>
<td>101.6</td>
<td>4.0</td>
</tr>
<tr>
<td>HWT Casing</td>
<td>114.3</td>
<td>101.6</td>
<td>101.6</td>
<td>2.5</td>
</tr>
<tr>
<td>PW Casing</td>
<td>139.7</td>
<td>117.0</td>
<td>117.0</td>
<td>3.0</td>
</tr>
<tr>
<td>PWT Casing</td>
<td>139.7</td>
<td>117.0</td>
<td>117.0</td>
<td>2.5</td>
</tr>
</tbody>
</table>

Initial hand-held drilling and drilling with smaller capacity drill rigs used EW, AW and AQ drill rods. Higher capacity rotary drilling used BQ and BWJ rods before long-hole drilling required larger N sized rods with higher strength and rigidity to manage the higher thrust loading expected. Wireline capable rods (NQ, NT, NO, NRQ and CHD 76) were used to allow wire-line surveying, reduce water flow resistance and eventually allow the installation of MECCA for electronic surveying.
3.3.2 Drill Rods Used for Directional Drilling in Underground Coal Mines

NQ rods (Figure 3.32, Longyear, 1984a) were the original rods of choice for long-hole directional drilling as they provided high strength and increased rigidity over smaller rods with a wire-line capability for Eastman single-shot surveying. The CHD 76 drill rod (Figure 3.33) is a composite rod with internally up-set walls (Figure 3.34, Technidrill, 2012) through the tool joint. The increase in wall thickness allows for a heavy duty thread compared to that of the standard Longyear Q series thread (Longyear, 1984b). The thread profile allows easier thread jointing with reduced cross-threading and extended life.

Figure 3.32: Q-series pin and box threads (Longyear, 1984a)

Figure 3.33: CHD 76 pin and box threads
Longyear developed the NRQHP thread with acute angles on the outer edge of the thread. Although gaining an initial interest, limited strength and in-hole failures prompted the return to CHD 76 as the preferred rod.

Figure 3.34: Construction of the CHD 76 rod (Technidrill, 2012)

The usual failure mode is similar with both Q series and CHD 76 joints with failure of the pin thread common within 10 mm of the shoulder (Figure 3.35).
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3.3.3 Non-Magnetic BeCu Drill Rods

With surveying being a key element of directional drilling, non-magnetic drill rods are required to provide an environment free of magnetic influence for accurate azimuth surveys. Rods were originally produced of austenitic stainless steel or K-monel. These materials were relatively low in strength compared to that of the high tensile steel drill string and also suffered galling of the threads. Each material was also proven to become magnetised after machining and repeated use.

Access to Beryllium Copper (BeCu) provided a non-magnetic material with similar strength characteristics to that of high tensile steel. The BeCu used for drill rod manufacture is listed as Alloy 25 (C17200) which contains 1.8-2.0 % Beryllium, 2 % Nickel + Cobalt with the balance of Copper (Materion, 2013). The alloy is precipitation age hardened (heat treated) and cold drawn to increase the hardness and tensile strength. The material is prone to work hardening leading to brittle failure (Figure 3.35).

To avoid work hardening of box threads through the usual stretching associated with rod jointing, threads have the box thread machined slightly over-sized rather than with the usual interference fit. BeCu threads can be hand tightened to shoulder contact (no standoff). Loktite is used on these threads to bind the joint instead of relying on the friction of an interference fit.

![Figure 3.35: Failed CHD 76 pin thread](image_url)
3.3.4 Drill Rods and Casing Used in Over-Coring Applications

HQ rods are the preferred rods for over-coring recovery of a bogged N-sized directional drill string. HQ is the next size larger than NQ with drill rigs easily fitted with chuck and rod clamp jaws to suit.

PQ rods and rod quality HW and PW sized casing have been used to install extended standpipe lengths to and through zones of instability to provide access to target drilling zones beyond. This casing was originally installed after directional drilling a pilot hole through the design profile then reaming to a larger diameter. This faces stability problems between the reaming and installation stages. Now the practice is to drill a pilot hole on a designed profile then ream the casing in following the pilot hole with a wire-line casing advancer (Longyear, 1988). When the casing is in place, rods are inserted inside the casing to detach and withdraw the internal drill bit of the casing advancer.

PW rods are used to over-core and remove 100 mm diameter standpipes in preparation for larger diameter over-coring operations. All drilling with the casing rods and PW rods is by top-end drive in front of the rotation unit.

3.3.5 Casing

Casing rods are available in a variety of threads similar to that used for drill rods. The material is usually reduced in quality for casing which is installed to be sacrificed in place. If drill or reaming is required to get the casing in place, rod quality material is specified to ensure adequate thread strength. Casing sizes and types used in underground drilling operations are listed in Table 3.4.

3.3.6 Thread Lubricant

Specially formulated grease has long been used to provide an anti-fouling lubrication between drill rod threads. To enhance the anti-seize properties, small particles of copper have been incorporated in greases like Kopa-Kote (Figure 3.36, Longyear, 1984a).
Further development produced lithium soap based greases which were tacky to reduce fling-off during rotation and provided adhesion when applied to wet metal surfaces. Molygrease became popular as a lithium complex soap based grease containing the inorganic solid molybdenum disulphide.

In the period 2000-to 2005, Chevron Lubricants developed Talcor Breakout Blue which was a very distinctive blue coloured anti-seize rod grease (Figure 3.37). Heavy metal fillers had been replaced with a new generation of bio-stable non-metallic fillers in a complex soap base. The soap based grease had good adhesive and cohesive qualities to ensure excellent adhesion even in wet conditions while minimising fling-off. As with most thread greases, this grease reduced the break-out torque required to separate rod joints.

Breakout Blue has become more difficult to acquire, so two alternative products supplied by Hard Metal Industries (HMI) have been introduced. Easy Release Thread Grease (ERTG) is similar to the Talcor Breakout Blue being a lithium complex soap based grease containing no metal fillers. It contains mineral oil and calcium compound additives. The grease designated DRTG ZN 50 (Figure 3.38) is heavy paraffinic based grease with a zinc powder additive. This has provided good adhesive characteristics in wet conditions and low torque requirements for separation of rod joints. It was recently used successfully on a project of extra-long borehole drilling and the metal content in the grease may have contributed to improved signal transmissions.
3.3.7 Drill Rods Used for Directional Drilling in Underground Coal Mines

All rod joints need to be screwed up to shoulder contact then have a prescribed level of torque applied to the joint. The prescribed pre-torque differs for each thread type and size. It is designed to ensure there is a friction bond between the contacting surfaces in the joint to provide maximum design strength for the transfer of tensile, compressive, bending and torque loading. The strength of the joint then has the potential to handle any axial, torque and bending loading and vibration it experiences. The prescribed pre-torque for the three drill rod types used in directional drilling are listed in Table 3.5 (Longyear, 1984a).
Each drill rig, when in rod handling mode, should have torque settings suited to each style of rod being used to prevent inadvertent overtightening damaging the threads.

Table 3.5: Pre-torque requirements for wire-line coring rods (Longyear, 1984a)

<table>
<thead>
<tr>
<th>Rod Type</th>
<th>Weight</th>
<th>Torque</th>
</tr>
</thead>
<tbody>
<tr>
<td>NQ</td>
<td>23.4 kg</td>
<td>750 Nm</td>
</tr>
<tr>
<td>NRQ</td>
<td>23.4 kg</td>
<td>2400 Nm</td>
</tr>
<tr>
<td>CHD 76</td>
<td>24.5 kg</td>
<td>2430 Nm</td>
</tr>
</tbody>
</table>

3.4 DOWN-HOLE MOTORS

3.4.1 DHM Operation and Construction

A DHM (Figure 3.39, Drillex, 1992) consists of a power unit, bent housing and bearing pack (Figure 3.40). The power unit is a stator and rotor through which high pressure water flow is converted to rotational power. The bent housing provides the deflection required for directional steering control. Deflection of the DHM can be provided by any combination of bent sub, bent housing, deflection pads and off-set stabilisers (Figure 3.41). Within the bent housing, a set of universal joints transmit the eccentric rotation of the rotor through to the drive shaft of the bearing pack. The bearing pack provides concentric rotation of the drive shaft which is fitted with a drill bit for drilling.

Figure 3.39: Sectioned view of Drillex DHM (Drillex, 1992)

Figure 3.40: Components of a DHM
CHAPTER THREE
Drill Rigs, Rods and Down-hole Motors

High pressure water is pumped through a DHM to produce rotation of the drill bit for drilling. After the water passes through the drill bit, the water becomes the flushing medium to flush cuttings from the borehole.

Figure 3.41: Deflection options of a down-hole motor (Drilex, 1992)

3.4.2 Introduction and Use of DHMs in Australian Underground Coal Mines

Initial directional drilling was undertaken with hired 60.3 mm (2-3/8”) diameter Dynadrill DHMs suited to the BQ drill rod diameter (Allen, 1984). These units had a 1-2 lobe configuration which produced high rotational speed (Figure 3.42) at relatively low torque (Figure 3.43) not particularly suited to coal drilling.
As research targeted long-hole drilling, an NQ rod size was chosen to provide a stronger and more rigid drill string. The only DHM easily available in Australia was the 73 mm (2-7/8”) Slimdril DHM. This unit was supplied with a non-magnetic (K-monel) power section (rotor/stator), laser-cut universal joints and a ball-bearing bearing pack. The bearing pack was easily serviced but the bearings had a limited operating life before
replacement. The 1-2 lobe power section offered relatively high rotational speed (Figure 3.42) and lower torque (Figure 3.43); again not really suitable for coal drilling. A bent housing with a 0.75° deflection was combined with an 89 mm diameter PCD bit to provide directional control. Although a depth of 1005 m had been achieved with this configuration, penetration started surging beyond 60 m and the feed rate was progressively reduced to prevent stalling of the DHM as borehole depths increased.

A 73 mm (2-7/8”) diameter Accu-dril DHM (Figure 3.44) was offered to the industry in 1990 by Asahi. This unit had a non-magnetic, high-torque, low-speed 4-5 lobe motor section which offered improved performance in high torque (Figure 3.43) at lower rotational speeds (Figure 3.42 and Figure 3.45) more suited for coal drilling. When fitted with a 1.25° bend and combined with a 96.1 mm diameter PCD bit, surging, which had been attributed to in-hole friction, was greatly reduced and drilling rates improved. This DHM and configuration of bend and bit diameter was established as the industry standard for in-seam coal drilling.

High volumes of BQ sized directional drilling in Queensland in the early 1990’s saw the introduction of the 60.3 mm diameter, 5/6 lobe Drilex DHM. This DHM had very high rotational speeds at relatively low water flow rates (Figure 3.42). When used with electronic survey instruments, vibration caused damage to the down-hole component of the survey instrument. The stator was heat treated steel which tended to have an...
additional magnetic influence associated with it. Use of the Drilex DHM was phased out in favour of the 4/5 Accu-drill DHM.

As drilling activities in stone strata above and below the working seam has increased, DHMs with higher torque and lower rotational speed have been introduced in efforts to improve drilling performance. The 5/6 lobe DHM and the 7/8 lobe DHM (Figure 3.43 and Figure 3.45) from Accu-drill have been introduced for stone drilling with mixed results. Stone strength and hardness and drill bit design need to be considered in selecting the DHM for each stone drilling project. An even-walled 5/6 lobe DHM has been heavily promoted as providing higher torque (Figure 3.43). Higher torque capacity is only evident at higher water differential pressures outside the pressure capacity of the water pump. The increased rigidity of the stator has introduced flexing problems with thread and body failures in the vicinity of the stator/bent housing connection.

![Figure 3.45: Rotational speed versus water flow rate of N sized DHMs](image)

DHMs with the range of lobe configurations have now been supplied by alternative suppliers/manufacturers and tested in operational environments. The performance of power units has been inconsistent and unreliable in capacity and the materials used in universal joints and bearings have been questioned with numerous failures. Asahi supplied DHMs are still the preferred standard although alternative supplier products are still being assessed with caution.
3.4.3 Design and Performance of the Power Unit

The power unit of the helical-shaped stator and rotor (Figure 3.44) is a Moineau progressing cavity, positive displacement fluid motor which converts hydraulic energy of high pressure drilling fluid to mechanical energy in the form of torque output for the drill bit.

The pitch length and diameter of the stator and the number of lobes determine the rotational speed for a particular flow and the torque output for a particular operating pressure. The number of stages (or pitch lengths) in the stator also influences the torque capacity.

The standard rotor is chromed steel while chromed K-monel or BeCu are supplied for non-magnetic power units. The stator is a metal tube with a moulded elastomer lining. The standard tube is high tensile and heat treated steel while K-monel or BeCu tubes are supplied for non-magnetic power units for close proximity surveying. The higher strength BeCu is now preferred (over K-monel) although consideration has to be given to the work hardening characteristics of BeCu. Excess and repeated flex at the front end of the stator has seen brittle failure (Figure 3.46)

![Figure 3.46: Brittle failure of BeCu stator near the bent housing connection](image)

Heat treated steel stators are recommended for high volume stone drilling to limit erosive wear although surveying requirements and limited driller skills usually dictate
that BeCu stators are used for closer proximity surveying. When unusually rapid erosion of the BeCu stator is found, it is suspected that the BeCu is not the standard age-hardened Alloy 25 (C17200) BeCu.

The sleeve cast inside the stator case is now an elastomer material available in various forms suited to specific temperature ranges or oil environment. The material in the Slimdril DHM swelled and stalled the DHM when soluble oil was used in the flushing water. Pumping fresh water through the stator was sufficient to free the rotor but long term damage was usually experienced with early failure of the elastomer sleeve. Current elastomers are more resistant to the present of hydrocarbons although additives to drilling water are usually limited to drilling muds, polymers or detergents. When a DHM is to be put into storage, it is recommended to hand pump oil with a high aniline point (low in volatiles) through the DHM.

During rotation of the rotor, deformation of the elastomer lobes of the stator generates heat. An additional function of the flushing water is to provide cooling of the elastomer and stator case. When bogged in a fractured seam, a section of stator was encased in coal while still operating, resulting in heating of the outer case. The tensile loads applied trying to free the DHM resulted in significant stretching of the DHM (Figure 3.47). The inner sleeve was destroyed.

Figure 3.47: Stretching failure of a BeCu stator after internal heating and high tensile loading
3.4.4 Bend Deflection, Erosion and Wear Pads

The bent housing has been the preferred means to provide the deflection of the DHM for steering. The magnitude of the bend has been combined with the diameter of the bit to provide adequate deflection for steering. To control directional drilling, this combination must provide the ability to climb. With the bend orientated at 12 o’clock, the deflection must be sufficient so the lower edge of the bit is above a horizontal extension of the bottom of the stator (Figure 3.40).

With the directional drilling configuration standardised at a bit diameter of 96.1 mm with a 1.25° bend on the DHM, the deflection (B, Figure 3.40) provided by alternate bends and bit diameter combinations can be expressed as an effective bend (Table 3.6). The addition of wear pads has provided additional deflection which also has to be considered when determining the effective bend.

Table 3.6: Deflections created by DHM bent housings

<table>
<thead>
<tr>
<th>Bend</th>
<th>Wear Pad Thickness</th>
<th>Bent Housing Diameter</th>
<th>Offset of 73 OD at Bit Box</th>
<th>Bit Deflection</th>
<th>Equivalent Bend</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>degrees</td>
<td>mm</td>
<td>mm</td>
<td>mm</td>
<td>degrees</td>
</tr>
<tr>
<td>0.77</td>
<td>0.0</td>
<td>73.0</td>
<td>10.08</td>
<td>0.0</td>
<td>0.77</td>
</tr>
<tr>
<td>1.125</td>
<td>0.0</td>
<td>73.0</td>
<td>14.73</td>
<td>5.3</td>
<td>1.12</td>
</tr>
<tr>
<td>1.125</td>
<td>1.0</td>
<td>73.0</td>
<td>16.08</td>
<td>6.7</td>
<td>1.22</td>
</tr>
<tr>
<td>1.125</td>
<td>2.0</td>
<td>73.0</td>
<td>17.44</td>
<td>8.2</td>
<td>1.31</td>
</tr>
<tr>
<td>1.125</td>
<td>3.0</td>
<td>73.0</td>
<td>18.79</td>
<td>9.6</td>
<td>1.41</td>
</tr>
<tr>
<td>1.25</td>
<td>0.0</td>
<td>73.0</td>
<td>16.36</td>
<td>7.2</td>
<td>1.25</td>
</tr>
<tr>
<td>1.25</td>
<td>1.0</td>
<td>73.0</td>
<td>17.72</td>
<td>8.6</td>
<td>1.34</td>
</tr>
<tr>
<td>1.25</td>
<td>2.0</td>
<td>73.0</td>
<td>19.07</td>
<td>10.0</td>
<td>1.44</td>
</tr>
<tr>
<td>1.25</td>
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<td>20.43</td>
<td>11.4</td>
<td>1.53</td>
</tr>
<tr>
<td>1.25</td>
<td>0.0</td>
<td>74.6</td>
<td>17.45</td>
<td>8.3</td>
<td>1.33</td>
</tr>
<tr>
<td>1.25</td>
<td>1.0</td>
<td>74.6</td>
<td>18.80</td>
<td>9.7</td>
<td>1.42</td>
</tr>
<tr>
<td>1.25</td>
<td>1.5</td>
<td>74.6</td>
<td>19.48</td>
<td>10.4</td>
<td>1.47</td>
</tr>
<tr>
<td>1.25</td>
<td>2.0</td>
<td>74.6</td>
<td>20.16</td>
<td>11.2</td>
<td>1.51</td>
</tr>
<tr>
<td>1.25</td>
<td>3.0</td>
<td>74.6</td>
<td>21.51</td>
<td>12.6</td>
<td>1.61</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>mm</td>
<td></td>
<td>Stator Length</td>
<td>2110</td>
<td>Std for Aus</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Bend-Bit box</td>
<td>750</td>
<td>Std 1.25 deg</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Bit Length</td>
<td>110</td>
<td>Std for China</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Bit Diameter</td>
<td>96.1</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>
The point of deflection at the joint between the stator and the bent housing is the highest frictional contact with the borehole and subject to the highest rate of wear. If not regularly checked and replaced when necessary, the joint is eroded to expose the threads (Figure 3.48).

![Figure 3.48: Abrasive wear through stator box thread](image)

Asahi was approached to provide a Tungsten Carbide (TC) coating on the bent housing to reduce the effects of abrasion. The TC coating was tested but failed (Figure 3.49) due to brittleness, lack of bonding with the housing and flexing of the bent housing. Drilling Products Inc. (DPI) was approached to provide a laser applied pattern of TC wear pads. During the initial laser applications, the steel of the bent housing was tempered. On numerous occasions the bent housing straightened either during drilling or as the DHM was pulled out through the bends of the directionally drilled boreholes. The application was refined to prevent tempering problems with 1 mm and 3 mm thick wear pads produced (Figure 3.50). Extensive stone drilling exposed the wear pads to rapid wear (Figure 3.51) so Thermally Stabilised Polycrystalline (TSP) diamond inserts were incorporated in the design with improved wear resistance (Figure 3.52).
CHAPTER THREE
Drill Rigs, Rods and Down-hole Motors

Figure 3.49: Failed TC coating and buckling of Asahi bent housing

Figure 3.50: Laser-applied TC wear pads – 1 and 3 mm thick

Figure 3.51: Abrasive wear of laser-applied TC wear pads
PCD cutters had proven superior to TC matrix in wear resistance for drill bits so the supplier (DPI) was asked to design a wear pad incorporating PCD cutters (Figure 3.53). Due to the exposed mounting of the PCD cutters, they suffered fracture failure (Figure 3.53) from vibration but were still effective after more than 5000 m of drilling in highly abrasive sandstone. Design improvements are required to provide more anchorage stability of the PCD cutters to reduce susceptibility to vibration damage.

The introduction of the wear pads into Chinese operations was not controlled with wear pads of 3 mm thickness being ordered with 1.25º bends without regard to the 1.53º effective bend produced (Table 3.6). The resultant severe boreholes curves have led to either broken DHMs (Figure 3.46) or straightening out of the bend. It is now difficult to reduce the wear pad thickness used. For VLI’s drilling operations, the bend and wear pad combination in Australia has been limited to an effective bend of 1.25º and in operations in China to 1.34º (Table 3.6).
3.4.5 Bearing Pack

The bearing pack (Figure 3.54) is fitted with radial bearings at the front and behind to provide concentric rotation through radial bearing sleeves on the drive shaft (Figure 3.55). Axial loads from the bit and from the rotor are transmitted through axial bearings to the outer case of the DHM.

Brass bushes were used as radial bearings in the Slimdril DHM with sets of ball bearing races used for the axial bearings. The operational life was limited by wear of the bearing balls with 3000 m of drilling typical before the bearing balls had to be replaced. Approximately 10% of the flushing water flow was diverted through the bearing races to provide cooling and lubrication. The recommended practice was to hand pump rust protection oil through the bearing pack when not in use.

The Accu-drill DHMs have used sintered brass bushes impregnated with TC for the radial bearings while the axial bearings have been sintered brass rings with TC inserts (Figure 3.56).
Trials of DPI supplied axial bearings with stronger and harder matrix material and TC inserts have provided axial bearings with more resistance to wear but these bearings have been prone to fracturing due to vibration. The softer brass matrix (of the Asahi axial bearings) is thought to better absorb vibration and as such, is less prone to vibration damage. The brass axial bearings from Asahi are preferred.

### 3.4.6 Universal Joints

Rotational power and torque is transmitted from the eccentric rotation of the rotor to the concentric rotation of the drive shaft via either a flexy shaft or a set of universal joints. The design of the Drilex DHM with short motor section and a long bent housing with reduced deflection allowed for the use of a flexy shaft. This was the only DHM which used a flexy shaft with all others preferring a universal joint system.

In the original Slimdril DHM, universal joints were in the form of interlocking laser-cut lobes with axial load through a ball bearing between internal conical shafts. Later versions were fitted with grease nipple facilities and rubber sleeves to contain the grease. The interlocking lobes were thought to have limited torque capacity and not used in the higher torque rated DHMs.

The universal system of the Accu-dril DHM has two sets of opposing needle bearings (Figure 3.57) separated by a short shaft (Figure 3.54). The needle bearing pockets are heat treated to provide improved wear resistance. After repeated failures of the shaft
(Figure 3.58), the drive shaft was redesigned with sharp corners eliminated to reduce areas of concentrated stress.

![Figure 3.57: Needle bearings of a universal joint](image)

Universal joints from an alternative supplier have experienced failure through failed shafts, deformed bearing pockets and tripping of the bearing pockets (Figure 3.58). Asahi manufactured universal joints with more reliability are preferred.

![Figure 3.58: Broken, deformed and worn universal joint bearing heads](image)
3.5 SUMMARY

As in-seam drilling became an established practice for gas drainage to control outbursts, drilling was undertaken with a combination of small hand-held drill rigs and larger “home-made” units. As the requirement for longer boreholes increased, drill rigs used in the hard-rock industry were imported and modified for use in underground coal mines. In the late 1980’s, Longyear worked with mining personnel to produce a drill rig to suit their specific drilling requirements.

In the early 1990’s, Longyear worked with ACIRL personnel to produce an electro-hydraulic powered long-hole directional drill rig to specifications defined by earlier directional drilling research and development. Additional versions of that drill rig have been produced with upgraded capacity to suit a wider range of drilling applications.

Through exposure to drilling opportunities in the Chinese coal industry, VLD designed a drill rig using the key aspects of the various drill rigs being used for directional drilling in the Australian coal industry. This resulted in the production of the Series 1000 track-mounted drill rig which has allowed the successful introduction of directional drilling to the Chinese coal industry from which numerous copied versions have been produced. Both the Longyear and VLI directional drill rigs have proven capacity to produce in-seam boreholes to depths beyond 1500 m.

Although numerous drill rods have been used from the variety of drilling applications, NQ rods have been the favoured drill rod for directional drilling due to the wire-line capability required for the Eastman single-shot surveying commonly used before electronic surveying was established. As electronic surveying was developed and used successfully, CHD 76 drill rods with thicker wall thickness through the thread and deeper thread design offered greater strength and easier rod handling. These are now the preferred threads for underground directional drilling.

The available supply of DHMs from the USA has determined which DHMs have been used in underground directional drilling operations in Australia. The earlier DHMs allowed the characteristics of directional drilling to be established. As higher powered options became available, drilling performance improved. Problems were identified with the DHMs which started having an impact on drilling operations. Some were able to be rectified through interaction with the initial supplier. Alterations to components of
the DHMs by alternative supplier have not been conducive to good DHM operations and reliability with numerous break-downs and limited success with rectification. The initial DHM supplier has been re-established with access to reliable equipment.
CHAPTER FOUR – DESIGN AND DEVELOPMENT OF
PCD DRILL BITS

4.1 INTRODUCTION
One of the key components of any drilling system is the drill bit. A range of drill bits have been used from the initial in-seam rotary drilling where TC drill bits were home-made or imported from surface drilling applications through to modern PCD drill bits capable of drilling both coal and stone. This development is explained in this chapter.

PCD drill bit technology was introduced from the oil-field industry but developed for specific smaller diameter underground directional drilling applications. This chapter describes the development of PCD drill bits used in underground directional drilling applications.

Over-coring and reaming drill bits have been designed with PCD technology being a key to providing the capability to penetrate stone. The design and application of these bits is explained in this chapter.

4.2 ROTARY BITS
As the drilling of in-seam boreholes was identified as a key means to drain gas to reduce the occurrence of outbursts (Hargraves, 1983; Clarke, 1983). Drilling projects in the 1960’s and 1970’s employed bits fitted with TC cutters and trials by the USBM (Cervik, et al, 1975) demonstrated that TC drag bits (Figure 4.1) had superior drilling rates over roller cone and diamond “bull-nosed” bits. These bits were usually “home-made” or produced by bit manufacturers for the surface drilling industry rather than being specifically designed for in-seam drilling applications (Hungerford, 1995).

As gas drainage drilling projects were employed in the early 1980’s, the only bits readily available were three and four wing tungsten carbide blade bits (B-E, Figure 4.2, Beamish, et al, 1985) usually supplied to the surface water-bore drilling industry. Roller-cone bits (I, Figure 4.2) were also available for drilling harder strata. The flat-faced TC bits penetrated coal successfully but were prone to large lateral deviations and penetration rates diminished rapidly with any cutting of stone due to rapid loss of the sharp cutting edge. To attempt to deflect out of stone to continue drilling in coal, roller
cone bits were used. This practice was time-consuming (involving rod pulling and running to change bits) and roller cone bits were limited by the life of the bearings.

![Figure 4.1: Drill bits (A drag, B roller cone, C surface-set diamond) (Cervik, et al, 1975)](image1)

Figure 4.1: Drill bits (A drag, B roller cone, C surface-set diamond) (Cervik, et al, 1975)

Drill bit performance improved with the availability of the Krupp Widia TC system of 65 mm pilot bit and 80 mm reamer (A, Figure 4.2). The TC cutter inserts were more robust and the pilot/reamer configuration provided reduced lateral deviation while the 2° taper allowed responsiveness to vertical control drilling parameters. The configuration also promoted deflection out of low-angled intersections with stone roof or floor. But

![Figure 4.2: Assorted bits for BQ rotary drilling (Beamish, et al, 1985).](image2)

A  Widia bit  
B  4-wing TC blade bit  
C  4-wing TC blade bit  
D  3-wing TC blade bit  
E  4-wing TC blade bit  
F  Terratek PCD bit  
G  TC slotting bit  
H  TC HQ shoe bit  
I  Roller cone bit
with the success of the Widia bits deflecting off stone to remain in seam, maximizing penetration rates became the dominant drilling parameter. The 65 mm pilot bit was also used initially for cross-measure stone drilling for post-drainage but required regular sharpening.

For smaller projects with hand-held borers (Hungerford, 1981; Hanes, 1997b) for pressure monitoring or gas drainage, 2-winged tungsten carbide bits usually of 43 mm or 47 mm OD were employed.

With the development of light-weight compressed-air operated Ramtrak drill rigs using AW rods for compliance coring ahead of the development face, “pineapple” or “Drilltec” bits (Figure 4.3, Asahi, 1994) were popular. The design incorporates a 16 mm metric threaded 2-wing bit in front of a tapered steel body fitted with large irregular “Drilltec” pieces and available in 55, 65 and 80 mm OD with an AW box thread. These bits provided good penetration in coal (McKinnon, 2005), limiting lateral deviation while allowing deflections from low-angled intersections with stone roof or floor.

**4.3 INTRODUCTION OF PCD BITS**

Most development in drill bit technology has been driven and financed by the oil and gas industry with the flow on effect benefiting other drilling industries and with that technology being adapted to suit differing needs. A key development in bit technology in Australia was the availability of PCD bits through an experimental bit acquired by ACIRL from the USA in 1982. This 80 mm “Terratek” Stratapax bit (Figure 4.4, Allan, 1984) was an early version of the PCD bits being developed for the oil and gas drilling industry in the mid-1970’s with PCD cutters mounted on TC pillars secured into the face of a steel body. The PCD cutters penetrated both coal and stone without requiring the constant sharpening associated with TC bits. The slightly convex face allowed deflection out of stone intersections.
A concave faced version (F, Figure 4.2) was made available for the Collinsville drilling project. The initial design with parallel sides had a tendency to continually climb (Beamish, et al, 1985) similar to positioning a stabiliser directly behind the bit. After grinding the sides to provide a 2° taper to duplicate the responsive gauge angle of the Widia bit, the bit became responsive in vertical control through variations in the drilling parameters (Figure 4.5). The ability of the PCD cutters to maintain penetration in stone allowed drilling to continue and eventually deflect out of stone intersections. The relatively flat face of the PCD bit produced more lateral deviation than that with the Widia bit.
A PCD pilot/reamer configuration bit was available from Triefus (A, Figure 4.6). It was a good combination of both pilot/reamer and PCD technology but was not adopted by high volume rotary drilling projects due to the relative success and cost effectiveness of the TC Widia bit. In hindsight, it would have been ideal for limiting lateral deviation and also for deflecting off stone intersections to continue drilling. The 65 mm pilot bit was developed and modified to become a standard bit (B, Figure 4.6) for cross-measure stone drilling for post drainage.

![Figure 4.6: (A) Triefus 80 mm bit and (B) Asahi 65 mm bit](image)

### 4.4 PCD TECHNOLOGY AND CONSTRUCTION

The main parameters for PCD drill bit design includes the characteristics of the PCD cutters, face layout, number of cutters, the size of the cutters and back rake angle and these dictate the “drillability” of the bit (the rate the bit can penetrate a particular strata).

Figure 4.7 (Triefus, 1985) demonstrates the comparative cutting actions of the various bit types. The roller cone bit fractures rock with a crushing action, the natural diamond bit ploughs and grinds while the PCD cutter shears rock much like a lathe cutting action. Tungsten carbide drag bits (not shown in Figure 4.7) have a gouging cutting action similar to that of natural diamond.


4.4.1 PCD Construction

The PCD cutter (Figure 4.8) (alternatively referred to in the surface drilling industry as a Polycrystalline Diamond Compact - PDC) consists of a tungsten carbide blank or Substrate which has a thin layer of synthetic diamond matrix (Diamond Table) bonded to it through a sintering process under high pressure and temperature in a synthesis press (Baker-Hughes, 2008). The matrix is formed from a mix of synthetic diamonds and a Cobalt bonding agent with the diamond size, portions, temperature and pressure being regulated to produce improved diamond table. The cutters are generally cylindrical (diameter in the range from 8 mm up to 30 mm) but are available in other forms (oval, triangle or ground to a required shape).

Early PCD matrix was prone to flaking fracture failure (Figure 4.9) and occasional total failure of the bond with the TC substrate. The PCD cutters are also sensitive to temperatures over 800ºC and can be adversely affected from the process of silver soldering the cutters into the bit face. Development in PCD technology has improved the matrix to be more robust to manage abrasion and shock loading.
CHAPTER FOUR
Design and Development of PCD Drill Bits

Figure 4.9: Fractured PCD table with eroded tungsten carbide substrate

The PCD/TC interface has been modelled and analysed with various substrate profiles (Figure 4.10) to limit stresses in the PCD cutter and improve the bond strength. This has improved the design of the PCD cutter in resistance to chipping and fracturing.

The modelling also considered various table thicknesses, interface shape and grade of tungsten carbide in the substrate to provide PCD cutters suitable for specific load applications (Baker-Hughes, 2008). Further developments have included a chamfered edge to improve impact resistance and a polished face on the PCD table to enhance the removal of cuttings.

Figure 4.10: Substrate profile - radial symmetry, axial symmetry and ripple (Baker-Hughes, 2008)

The general rule for cutter size and number is that small cutters and a high cutter count are chosen for hard and abrasive rock formation, whereas large cutters and a reduced cutter count are preferred for soft to medium formation (Richards, 2013).

As the diamond table is much harder but relatively brittle compared to the TC substrate, the TC behind the diamond table is progressively eroded to present a leading cutting edge of PCD matrix (Figure 4.11, Baker-Hughes, 2008). When the diamond table
fractures, the TC substrate is rapidly eroded to expose a new PCD cutting edge (Figure 4.9).

4.4.2 PCD Drill Bit Structure

PCD bits are designed to hold a number of PCD cutters in specific locations and at prescribed angles across the face of the bit to allow efficient cutting during drilling operations. The body of PDC bits have been manufactured from two materials: steel faced and matrix faced bits. The steel drill bits, cast or machined with either recessed mounts in the steel body or TC post mounts (Figure 4.4 and A, Figure 4.12) for the cutters, were better at withstanding impact loading than matrix bodied bits. These bits were generally preferred for soft and nonabrasive formations and large diameter boreholes. The main disadvantage with steel was that it was less erosion resistant than matrix and, consequently, more susceptible to wear by abrasive fluids. To reduce the bit body erosion, sides of the bits were either “hard-faced” with a coating material (B, Figure 4.12) or had TC pieces inserted.

Figure 4.11: Abrasive wear of a PCD cutter (Baker-Hughes, 2008)

Figure 4.12: (A) Steel bodied bit with PCD cutters mounted on tungsten carbide pillars and (B) steel bodied PCD bit
CHAPTER FOUR
Design and Development of PCD Drill Bits

With cast matrix bits (Figure 4.13, Baker-Hughes, 2008), a mix of TC and cobalt binding agent is cast and bonded at high temperature and pressure to a steel body. The steel body is then welded or soldered to a shank machined with the desired thread (Figure 4.13) for connection to the drill string. The face is designed with mounting recesses (or pockets) which position each cutter in a precise location at a prescribed angle. The cast matrix offers more resistance to erosion than that of steel bodied bits and with improved technology, the matrix has been developed to be more robust with improved resistance to erosion and shock loading from vibration. In high erosion areas, TC blocks or strips, synthetic diamonds and/or Thermally Stablised Polycrystalline (TSP) diamond blocks are inserted in the cast to enhance erosion resistance.

![Figure 4.13: PCD bit structure (Baker-Hughes, 2008) and with cast matrix face removed from the shank](image)

With relatively small diameter bits (96 mm), there was limited space to design the usual blade face layout of larger diameter PCD bits (Figure 4.14) (Baker-Hughes, 2008). The smaller bits were designed with flat or relatively flat faces for enhanced branching characteristics. This profile has reduced stability in comparison to the convex profiles shown in Figure 4.15. The coverage of the cutters are overlapped to provide total coverage of the cutting face with the placement designed to generate balanced torque loading on the face. The cutters are arranged in a spiral from the centre outwards (Figure 4.16) which assists in reducing torque vibration and promoting the removal of cuttings towards the periphery of the bit. Cutter layout and coverage is less ‘sophisticated’ when compared to larger diameter bits where the designers are afforded more space (Richards, 2013).
The outer cutters provide both axial and gauge cutting with inserts providing passive gauge protection along the outside of the bit. Exposure of the outer cutters and a flat face are preferred elements with directional drilling for starting and propagating a branch.
4.4.3 Rake Angle Mounting of PCD Cutters

The PCD cutters are mounted at an angle relative to the axis of the bit and direction of drilling/penetration. This is referred to the back rake angle (A, Figure 4.17) (Triefus, 1985). This angle is required to present the cutting edge of the cutter in contact with the strata and controls how aggressively cutters engage the strata. The side rake angle (B, Figure 4.17) aligns the cutter so the back edge is within the cutting circumference of that cutter. It also enhances the shearing action of the cutter and helps to direct the cuttings towards the periphery of the bit.

![Figure 4.17: (A) Back and (B) side rake angles of mounted PCD cutters (Triefus, 1985)]
Generally, as the back rake is decreased, the cutting efficiency increases (more aggressive) but the cutter becomes more vulnerable to impact fracturing. The general trend is for the inner cutter to have a back rake angle of 20-30º with the back rake angle reducing to 10º for the outer cutters. A higher back rake angle for the inner cutter is also needed to ensure the back of the cutter is lowered so both it and the mount is inside the cutting coverage. Inadequate back rake angle and/or side rake angle exposes the back of the cutter and mount to contact with the face and erosion (Figure 4.18).

![Figure 4.18: Low back rake angle exposed cutter table and mount backing to erosion](image)

A large back rake angle results in a lower rate of penetration but gives a longer PCD bit life. Penetration rates reduce more rapidly as wear flats develop more rapidly for the smaller back rake angles (Figure 4.19).

![Figure 4.19: Wear flats for various back rake angles of mounted PCD cutters (Baker-Hughes, 2008)](image)
4.5 PCD BITS FOR DIRECTIONAL DRILLING

Steel bodied PCD bits (A, Figure 4.12) of similar design to the Terratek bit were available for DHM drilling. But these were regarded as relatively expensive and with long delivery times from the USA, emphasis was directed towards developing a locally available product. Longyear had advertised cast matrix technology for drill bit construction so were approached to design and manufacture a PCD bit with a cast matrix face.

Longyear produced an 80 mm PCD rotary bit and an 89 mm flat faced PCD bit (Figure 4.20) with exposed side cutting action for DHM drilling. The 80 mm design was used successfully for in-seam rotary drilling for exploration at John Darling colliery (Hungerford, 1986), German Creek mine (Hungerford, 1988b) and in Oceania mines (Ross 1991).

![Figure 4.20: Original Longyear 89 mm PCD bit with exposed outer cutters](image)

The initial design of 89 mm bits was found to have design flaws which had adverse influences on drilling performance and bit life (Hungerford, et al, 1988a). The main problems were with the security of the outer cutters and uniform water flushing:

- The steep angle of the back of the cutter mounts led to several break-out failures of the outer cutter mounts.
- Erosion under the outer mounts led to failure of one or more outer mounts (Figure 4.21).
• The single central flushing port was choked from a cylindrical cross-section to a half round section as it exited the face adjacent to the inner cutter mount. This directed high water flow at an angle to the alignment of the bit and the DHM shaft and caused severe vibration of the DHM.

• With one off-line flushing port and evidence of uneven wear distribution on the PCD cutters across the face, there was evidence of inconsistent flushing and cuttings removal across the face.

Figure 4.21: Original Longyear bit with eroded and failed outer mounts

Longyear were contacted and asked to make several modifications. These modifications included:

• To incorporate a TC reinforcement bar at the leading edge of the angle support under the mount of each outer cutter.

• Changing the mount to a swept back support behind each outer cutter. Plaster modelling was used to indicate the design change to Longyear (Figure 4.22).

• Changing the face shape to slightly convex so the outer mounts were provided more side support.

• Reducing the height the cutters stood above the face of the bit to reduce exposure and provide stronger mounting.

• Moving the choke in the central flushing port further back into the body of the bit so water is directed straight out the front of the face.

• Adding three flushing ports, one located in front of each outer cutter.
These modifications proved successful and were adopted by Longyear (Figure 4.23). They were also incorporated in their design of rotary drill bits with semi-parallel sided gauge protection. The problems solved during this evolution of bit design have since been established as key points for checking bit design, manufacture and performance.

Asahi produced a similar design in their PCD Claw bit series with semi-parallel side gauge protection with TC inserts (Figure 4.24). These bits are aggressive and have been used extensively in underground gas drainage drilling. They are available with a non-return valve incorporated inside the threaded section (Figure 4.25).

The introduction of the semi-parallel sided PCD bit from Asahi demonstrated that the drill bit did not need extremely exposed outer cutters to provide the directional control and the ability to branch required by directional drilling. The outer cutters were provided with more protection from erosion and vibration and were more securely mounted.
4.6 PCD BIT DESIGN FEATURES FOR DIRECTIONAL DRILLING

More bit manufacturers have entered the market in recent years and have provided a larger selection of bit designs. Some have copied established and proven designs while others have provided distinctly different bits in design and cutter configuration to provide cutting characteristics suited to differing drilling environments and conditions.

The 13 mm PCD cutter is the established standard in various forms of cutter number and layout. The same design principles of back rake and side rake angles, axially balanced flushing and cutter coverage apply (Richards, 2013). Smaller cutters (A, Figure 4.26) have been recommended for harder strata or to produce smaller
cuttings while recessed cutters (B, Figure 4.26) are more securely anchored and produce smaller cuttings. Both smaller cutters and recessed cutters provide slower penetration rates which are more suited for longer boreholes or drilling in boggy conditions.

![Figure 4.26: Flat-faced bits with (A) 8 mm cutters and (B) 50% recessed 13 mm cutters](image)

The convex bit (Figure 4.27) with fully exposed 13 mm cutters is a very aggressive bit with very good penetration rates but the convex face is not conducive to branching as it trends to deflect back into the original hole.

![Figure 4.27: DPI convex-faced bit with fully exposed 13 mm cutters](image)

To enhance the ability to drill backwards out of boggy conditions, back-cutting facilities have recently been added to bits in the form of TC pieces (A, Figure 4.28) or PCD cutters (B, Figure 4.28). These cutters have been proved successful in assisting the freeing of bogged equipment. The TC chips are favoured for coal while the PCD cutters are favoured in stone.
4.6.1 Cutter Coverage and Loading

The conventional understanding has been that the outer cutters are subjected to the most work and the inner cutters just remove the remnant core produced by the outer cutters. To cater for this, established designs increase the cutter count (or density) towards the outside of the face which reduces the exposure of these cutters (Figure 4.29). With wider spacing towards the centre, the inner cutters have a greater contact exposure.

![Figure 4.29: Cutter profile of a 6+3 cutter bit](image)

The radial distance of each cutter from the centre of the bit determines the travel distance around the circumference through each rotation. So with varying radii of rotation, the various cutters travel different distances each rotation. For the forward penetration achieved with each rotation, the inner cutters have to penetrate the depth over a shorter distance of travel, subjecting the inner cutters to a more gouging effect. The outer cutters are subjected to a more machining effect to achieve the same depth of
penetration over a longer travel in each rotation. The cutters immediately inside the outer cutters are subjected to longer travel with reduced penetration rate so are subjected to a longer abrasive exposure per rotation. The relative abrasive effects on cutters with longer travel at a reduced load compared to short travel at an increased load is difficult to access.

The replicated coverage of the outer cutters means these cutters share the penetration achieved during each rotation. Three outer cutters are each exposed to one third of the penetration while four cutters are each exposed to a quarter of the penetration and thus a quarter of the load.

The combined effect of the inner cutters having more exposure plus a faster penetration rate means the inner cutters are subjected to higher load. This has been evident with the fracture of both diamond table and TC substrate of the inner cutter (Figure 4.30) when bits were subjected to high torque loading when drilling stone. After the cutter fractured, a more stable cutter with a straight edge was created with lower cutting contact relative to the anchorage. The exposed TC matrix mount was worn back from the new straight edge. The more stable, reshaped cutter continued to cut for the life of the bit without further damage.

![Fractured inner cutters](image)

Figure 4.30: Fractured inner cutters

With the very limited exposure of the cutters immediately inside the outer cutters, it is likely little consideration has been given to balancing the torque loading around the face of these bits. Each supplier usually passes on a product from a manufacturer with the manufacturer having their own interpretation of efficient bit design. From a commercial
aspect, the manufacturer is reticent to go to the expense of a design change unless there is some technical justification likely to produce some financial gain.

With horizontal drilling in coal, low angled intersections with stone present the outer cutters in contact with the harder strata before that contact is detected through increased pressures at the drill site. Before the penetration rate is reduced to suit the harder strata, the outer cutters are forced into the harder strata at a penetration rate much higher than usually used for stone. This subjects each cutter to high loading which may fracture the cutters (Figure 4.31) or mount (A, Figure 4.32). This is mitigated by the load being shared between the outer cutters. Any erosion of the outer cutter mounts makes them more susceptible to failure (B, Figure 4.32).

Figure 4.31: Fractured outer cutter

Figure 4.32: Fractured of outer cutter mounts (A) before erosion and (B) after erosion
If the outer cutters survive the increased loading through shared loading, the next cutter towards the centre becomes the most susceptible as it does not share load with other cutters. The close coverage of the cutters towards the outside means a high load is only experienced over a small contact area. While the small exposed cutting edge may fail, the small contact area limits the load to which the whole cutter and thus the mount are subjected. As half the face engages with the harder strata, the inner cutter is in contact with stone and is subjected to the highest load due to the higher relative penetration rate.

4.6.2 Drill Bit Gauge Protection

From the introduction of TC inserts with the early Longyear bits, inserts have been an integral component to ensure erosion resistance and gauge protection, especially with semi-parallel sided bits being used. TC chips and bars were used initially to good effect. Technology development saw the introduction of Thermally Stabilised Polycrystalline (TSP) diamond inserts. These were combined in the form of bars with TC bar inserts (Figure 4.33) or as blocks and natural diamond inserts (Figure 4.27).

![Figure 4.33: Mixed T/C and TSP gauge protection inserts to base of outer cutters](image)

Although the gauge protection of Asahi bits is competent, the inserts are relatively shallow which leads to fracture and displacement of the inserts. From there, erosion removes the remaining inserts (A, Figure 4.34) and progressive erosion of the matrix occurs along the lines of flushing and cuttings flow from the face of the bit (B, Figure
The impact resistance characteristic of the matrix is evident in that the outer cutters have been extremely undercut without failing.

Manufacturers had trouble producing drill bits with exposed outer cutters and either parallel sides or with a 2° taper with gauge protection inserts. Bits were supplied with oversized gauge protection (A, Figure 4.35). The gauge protection inserts bore the brunt of the final gauge cutting. The result was very slow penetration rates in stone and appreciable wear on the matrix to expose the gauge protection inserts (B, Figure 4.35).

Bit manufacturers also supplied bits with parallel sides exactly matching the cutting diameter of the outer cutters (A, Figure 4.36). These caused surging DHM loading and feed due to jamming in the borehole. To ensure the cutters are exposed, bits are now
ordered specifying an undersized parallel gauge protection OD of 94.5 mm (B, Figure 4.36) with the cutting diameter of the back cutters further undersized. Gauges are available to check the supplied dimensions (Figure 4.37).

![Figure 4.36: Parallel gauge protection (A) full diameter and (B) recessed diameter](image)

An advantage with PCD bits is that they can be refurbished (A, Figure 4.38) if there has not been adverse erosion or failure of the face of the bit. The cost of refurbishment is a lot less than the cost of a new bit. Cutters which have suffered minor fractures or wear

![Figure 4.37: Gauge for measuring gauge protection diameter](image)
can be rotated to present a new cutting edge to the face (B, Figure 4.38). In some cases damaged mounts have been repaired by brazing in TC chips to form a new pocket. They are not usually as secure and cutters can be dislodged more easily as was the case in Figure 4.39.

![Figure 4.38: (A) Refurbished PCD bit, (B) rotated PCD cutter](image)

![Figure 4.39: Repaired cutter mount](image)

### 4.6.4 Drill Bit Face Flushing

With smaller size of drill bits compared to those in large oil-field projects, flow characteristics across the face are not as critical but still have an influence on drilling performance. Earlier studies identified the need for balance axial flow from the bit face to avoid lateral loading on the bit and cutters. A design aim has been to provide increased flushing port area across the face to achieve balanced but reduced flow velocity at the face. Most return water would then pass through the junk slots down the
sides of the bit with some flow (and cuttings) passing between the gauge protection and the sides of the borehole (Figure 4.40). Failure to provide some clearance has resulted in jamming of the bit. This flow (including cuttings) across the gauge protection increases the need for erosion resistant inserts.

![Flushing water flow from face of drill bit](image)

**Figure 4.40: Flushing water flow from face of drill bit**

Non-return valves have been employed in the bit (Figure 4.25) and/or at the back of the DHM to prevent flow back through the face of the bit when the DHM is not in operation.

Back flushing ports have been added to bits at a 45° angle (A, Figure 4.41) and at 90° (B, Figure 4.41) to provide flow back into potential blockages behind the bit to enhance the flushing clearance potential. These additions have not been analysed to determine flow rates through the back-flushing ports and their effectiveness at clearing blockages has not been proven. The right-angled ports may also be detrimental to borehole stability in unstable strata.
4.41  PCD back-cutters with (A) angled flushing ports and (B) flushing ports at right angles

The effectiveness of back-flushing ports may be in the reduction of flow velocity at the face of the bit when in unstable strata. High flows can be maintained to run the DHM while reducing the jetting effect of the flushing at the face.

The flushing characteristics at the face and around the cutters of large diameter bit in the oil field drilling industry have been analysed through computational studies of fluid dynamics (Figure 4.42) to provide the most efficient bit design and removal of cuttings from the face (Menand, et al., 2005). This not as critical with smaller diameter bits.

4.42  Computational fluid dynamics (CFD) modelling of flushing at the bit face (Menand, et al., 2005)

4.7  ALTERNATIVE BITS

Although PCD bits have made substantial inroads into the drilling industry, they were still limited in the hardness of stone they are capable of effectively penetrating. When strong, fine grained stone or stone subjected to igneous effects is encountered,
penetration rates have been reduced to levels which were unsustainable for economic drilling operations.

When this stone was encountered at the start of drilling, a slightly larger diameter (98 mm) roller cone bit with TC buttons (A, Figure 4.43) was introduced. This bit was used to drill the stone section and replaced when coal was intersected. Its use was limited and replaced regularly due to the complications in the hole if the bit failed.

This practice was not possible if the harder stone was intersected at depth. In cases of intersecting hard dyke material, the DHM is pulled and a core barrel inserted to core through the dyke before continuing directional drilling. In cases where there has been a requirement of maximum removal of cuttings from the borehole, high-speed DHMs have been used with impregnated diamond bits (B, Figure 4.43). To improve penetration rates, the triangular sections were removed to leave just the ‘Y’ section with the bit acting similar to a drag bit.

![Figure 4.43: (A) Roller cone bit with tungsten carbide buttons and (B) impregnated diamond bit](image)

### 4.8 OVER-CORING/SHOE BITS

Directional drilling involves expensive equipment down each borehole in potentially unstable environments. This puts the equipment at risk of being bogged and possibly not retrievable. The established practice of retrieval has been over-coring with larger diameter drill rods (or rod quality casing) fitted with a shoe bit. The shoe bits have been designed with the cutting capacity and diameter to suit the application. The main
characteristic of the shoe bit (as compared to a core bit) is the internal surface of the bit is equal to or larger than that of the over-coring rods and has no internal cutting components. The cutting components of the shoe bit cannot make contact with the inner rods being over-cored. A normal core bit with internal gauge cutters makes contact and can cut through the inner rods.

Early versions were home-made with TC cutters brazed to the end and sides of a short length of over-core rod (A, Figure 4.44). These were usually used to clean out a borehole up to and through a blockage without the need for enlarging the borehole diameter. Their failing was the progressive loss of external cutters (particularly if stone had been penetrated) which reduced the bit clearance and progressed towards the event of bogging of the over-core rods. Better designed and manufactured TC (Drilltec) shoe bits (B, Figure 4.44) were eventually provided by drill bit manufacturers. Impregnated diamond shoe bits (C, Figure 4.44) were also available for cleaning out blockages in stone sections without enlarging the original borehole.

![Figure 4.44: HQ shoe bits with (A) external TC cutters, (B) TC face cutters and (C) impregnated diamond face](image)

With increased borehole depths resulting in increased friction, the over-core process included increasing (reaming) the diameter of the borehole. This reduced the effects of bends in the borehole and reduced contact friction with the over-core rods. To provide a shoe bit capable of reaming both coal and stone strata intersected by the original borehole, a 135 mm OD Asahi PCD bit was modified by removing the pin thread, grinding the centre out to a diameter of 78 mm and welding on a HQ box thread (Figure
Specific shoe bits fitted with PCD cutters were then designed to suit various coal and stone over-coring applications.

Long and short bodied 105 mm OD shoe bits (A and B, Figure 4.46) were provided for relatively short depth recoveries while a stepped 125 mm OD shoe bit (C, Figure 4.46) was provided for longer borehole recoveries. Where hard stone has been intersected and must be reamed, a surface-set diamond shoe bit is available (Figure 4.47).
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Figure 4.47: 125 mm OD surface-set diamond HQ shoe bit

4.9 STANPIPEING AND REAMING BITS

Boreholes have had to be opened to a bigger diameter to allow standpipes to be installed to manage gas flows. In the days of open-holed drilling, an in-line reamer was installed in the drill string for the drilling of the last 3 m to enlarge to 80 mm diameter. As gas flows increased and standpiping was required to manage gas flows during drilling, 100 mm diameter standpipes were grouted into a 150 mm diameter hole drilled with a replaceable-blade drag bit followed by a cylindrical stabiliser (A, Figure 4.48, Cervik, et al., 1977) to attempt to provide a straight hole. This system was heavy so was modified with a lighter fluted stabiliser (B, Figure 4.48).

Figure 4.48: 150 mm replaceable blade bits with (A) cylindrical (Cervik, et al, 1977) and (B) fluted stabilisers

The Krupp Widia pilot bit and reamer system (Figure 4.49, Krupp Widia, 1982) was used where the coal seam was reasonable consistent. When the standpipe was to be
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installed into stone, reamers fitted with PCD cutters were available. Welding the reamer together to prevent unscrewing due to vibration was unsuccessful with the welds cracking and reamers being lost in-hole. The common practice developed was to clean the threads and apply high strength Loctite.

![Image of Krupp Widia multistage bit and reamer system](image)

**Figure 4.49:** Krupp Widia multistage bit and reamer system (Krupp Widia, 1982)

Each of these systems suffered with borehole deflection if stone bands were intersected resulting in difficulties in installing the standpipe. The TC cutters were not capable of penetrating stone roof or floor so PCD technology was introduced with a straight pilot borehole being drilling initially with the aid of a close fitting 94 mm stabiliser behind a standard sized 96 mm PCD bit (Figure 4.50). This ensured the pilot borehole was not deflected through the coal stone interface. The borehole was then reamed to 145 mm diameter by installing a PCD reamer behind the pilot bit and stabiliser configuration (Figure 4.51). Loctite was required to prevent the front stabiliser and pilot bit from vibrating loose. Although it is excessive for standpiping in coal, the PCD pilot bit, stabiliser and reamer system is capable of drilling straight boreholes for installing longer standpipes in both coal and stone. The 145 mm diameter was adopted so it could also be used through a 150 mm valve for borehole reaming.
With the use of cross-measure boreholes drilled in stone above a longwall for gas drainage, the boreholes are then reamed to 145 mm diameter to improve gas flow. The stone drilling standpipe configuration was used for this reaming but suffered thread failure from the reamer to the drill rods and the stabiliser and pilot bit vibrating loose even with Loctite applied. Heavier threaded NWJ rods are now used connected directly to the reamer to reduce thread failure. A single piece (stabiliser and reamer) has now been designed and manufactured (Figure 4.52, HMI, 2015) to eliminate the unscrewing problem. A worn or damaged bit is added to the front and sacrificed if it unscrews down the hole.
4.10 SUMMARY

Initial drilling practices in underground projects employed drill bits with tungsten carbide cutters which were competent in coal strata but blunted rapidly whenever stone was intersected. The introduction of bits with PCD cutters in the early 1980’s allowed drilling at reasonable penetrations rates in both coal and stone without the need to change bits. The early bits had steel bodies with PCD cutters mounted on tungsten carbide pillars set into the body.

The development of PCD technology was driven and financed by the oil and gas drilling industry. Bits were produced with cast matrix faces moulded pockets which positioned the cutters in specific location and angles to provide desired cutting characteristics. This matrix was more erosion resistant that steel bodied bits and was further reinforced by the addition of tungsten carbide and TSP inserts as gauge protection. The use of PCD bits has allowed improved drilling performance in an expanding number of drilling applications in both coal and stone.

Drill bit design has also provided smaller diameter bits for use in underground directional drilling applications although most bits used are as provided by the manufacturers. Studying the configuration and performance of these bits has allowed suggestions to be made to the some manufacturers to improve the design of their bits for general directional drilling as well as to provide specific characteristics for penetration rates, the ability to branch, size of cuttings, consistent torque loading and steering response.
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The key design aspects identified for PCD bits used for directional drilling applications were:

- Flat faced design with slightly exposed outer cutters to enhance the ability to branch.
- Complete overlapping coverage and balanced torque loading of PCD cutters on the face.
- Back rake angles determined to suit relative loading and frictional exposure.
- Side rake angles to position the back of the cutter within the cutting coverage, enhance the shearing action of the cutter and help to direct the cuttings toward the periphery of the bit.
- Gauge protection inserts to resist erosion of the sides of the bit and under-cutting of the outer cutter mounts.
- Axially balanced water flushing ports in the face to provide even flow distribution and avoid vibration during rotation.
- Back-cutters to assist recovery from boggy environments.

PCD cutters have been introduced to shoe bits for over-coring, bits for larger diameter drilling for standpiping and reamers for enlarging directionally drilled boreholes for gas or water drainage. This has allowed drilling through stone without the need for regular sharpening of cutters.
CHAPTER FIVE – Design and Development of Borehole Survey Systems for In-seam Drilling

5.1 INTRODUCTION

All boreholes are drilled from a specific location for a defined purpose. The vertical angle (pitch) and azimuth at entry can be measured with the aim of intersecting a target point or to position the borehole along a desired alignment. Only in ideal conditions might the path of the drilled borehole follow the original pitch and azimuth established at the start. It is more usual for a borehole to deviate from the original direction as a result of intersecting variable strata and the mechanics of the drilling action. To determine the position of a borehole, some form of borehole surveying is required.

This chapter describes the survey systems that have been used for drilling in the Australian coal industry and the development and use of the instruments which have been proven effective with the introduction of directional drilling. The sources of magnetic influences which are detrimental to survey accuracy are identified. Systems have been developed to plan and design borehole and define seam profiles, position boreholes in accordance with mine plan requirements and to monitor drilling progress relative to the design.

5.2 BOREHOLE SURVEYING

When in-seam drilling was developed as a means of gas drainage, whether or not boreholes were surveyed depended on the application. Various methods were employed to design borehole layouts and then confirm the location of the drilled boreholes with drilling patterns designed on the assumption that all boreholes would remain straight and consistent coverage would be achieved if some deviation occurred. One system involved surveying the intersections of boreholes by mining and a representative curvature established for drilling in that particular area. Subsequent designs allowed for that curvature and borehole intersections were surveyed to verify the design practice. The general location of boreholes which did not achieve design depth allowing them to be intersected by mining could be plotted reasonably accurately.

The alternative and more accepted practice is to survey a borehole along its length. At selected intervals along the borehole length, a survey instrument is used to measure the
pitch and azimuth and the position of each survey point (X, Y and Z co-ordinates) is determined through one of a variety of calculation systems. With the start point known, the borehole can then be plotted on a mine plan. This survey process can be undertaken progressively as drilling progresses or as a post-drilling activity. The interval distance is determined by the accuracy required with shorter intervals providing higher accuracy.

Some of the early rotary drilling for gas drainage required progressive surveying with an Eastman photographic single-shot survey instrument on wire-line. This was used to identify the optimum drilling parameters (penetration rate and rotational speed) needed to maintain vertical control to keep the borehole within the seam. Progressive single-shot surveying was the established practice to maintain vertical control and monitor direction for rotary exploration drilling. Due to the gradual and reasonably consistent curves usually produced with rotary drilling, the survey intervals can be lengthened to reduce delays to drilling.

The limitations of rotary drilling and associated surveying systems were exposed when gas drainage boreholes out of position were associated with an outburst which resulted in a fatality (Harvey and Singh, 1998). This resulted in the need for the location of all boreholes to be accurately confirmed through surveying. Following the event West Cliff instigated a policy that all holes had to be surveyed, a hole does not exist unless it has been surveyed (Tonogato, 1998). Two multi-shot instruments, the "Peewee" photographic instrument (Hungerford, 1995) and the Surtron "Champ" electronic instrument (Hungerford, 1995) were trialled with limited reliability and success in post-drilling surveying applications. After a rotary borehole was completed with non-magnetic rods behind the bit, the instrument could be pumped in (if it fits inside the drill rods being used) to survey the borehole as the rods are withdrawn. Alternatively, the instrument could be fed into the completed borehole either inside the drill string or manually on the end of conduit to survey the borehole.

Since directional drilling with its associated surveying was the only reliable means to accurately position boreholes, it became the accepted practice for in-seam gas drainage drilling. With that, surveying became an integral component of in-seam drilling and created pressure to improve survey technology. As well as measuring the pitch and azimuth at regular intervals during drilling, progressive surveying with down-hole motor drilling was required to monitor the down-hole motor orientation to maintain
control of borehole trajectory and direction. This was initially limited to the single-shot technology available at the time. But the need for fast accurate survey systems prompted the trial of several imported cable-less electronic survey instruments. Of these, one unit was developed to produce two effective, approved electronic survey instruments to suit the local conditions. These instruments have a down-hole unit installed in the drill string behind the DHM to, on request, measure borehole pitch, azimuth and DHM orientation and then transmit that survey data back to the drill site for processing. Both units are suited to 69.9 mm NQ size rods or larger.

Systems were developed to allow drillers to plot survey data on site to monitor and control the drilling process. These provide an aid to steer boreholes while defining the seam profile along the line of drilling. With access to the computing facilities incorporated in the electronic instruments, survey data is now processed on site. The drillers are provided with the X, Y and Z co-ordinates of each survey point to be plotted progressively to assist vertical and lateral steering control.

Rotary drilling which is still undertaken to provide compliance gas content coring is post-surveyed by pushing an Eastman single-shot instrument attached to the end of PVC tube into each borehole.

5.3 SURVEYING INSTRUMENTS

5.3.1 Eastman Single-Shot Survey Instrument

The Eastman Single-Shot Instrument (ESI) has been a reliable tool for borehole surveying. It had restricted Mines Department (MD) approval of a mechanical watch timer and two electronic timer versions (Figure 5.1) of the instrument. The “shot” is a negative photo of the gimbal compass with horizontal and vertical reference cross-hairs superimposed on the photo (Figure 5.2). The instrument has been used in progressive wire-line operations or fed manually into boreholes with conduit after drilling. The instrument suffers from increasing delays to drilling with increasing depth. Using longer intervals between survey points to improve drilling performance, increases the chances of errors in borehole positioning and control. It was the main instrument used in long-hole exploration drilling as it was the only instrument to operate reliably beyond depths of 700m before electronic survey instruments were developed to reliable standards.
5.3.2 Geoscience Model 24 Cableless Survey System (CSS)

The Geoscience Model 24 CSS was the first down-hole monitor used in Australian underground drilling (Hungerford, et al, 1988b) and was successful to a depth of 1004 m in 1986. The instrument was cableless; electromagnetic transmission along the drill string was used to communicate survey information to the borehole collar (Geoscience, 1986). Frequencies had to be progressively tuned to suit the borehole depth. The survey results were processed on site to provide the drilling with the X, Y and Z co-ordinates to allow make borehole plotting and steering easier.

5.3.3 Surtron “Drill Scout” Instrument

The Drill Scout instrument was installed behind the DHM for surveying and directional control. Communication was provided via a cable which was extended from a cartridge inside the drill string as each rod was added. This instrument had received MD approval status but cable problems limited its use in the industry.
5.3.4 Conoco/DuPont Directional Drill Monitor (DDM)

The electronic Conoco DDM was imported by DuPont in 1986. Although operated successfully to depths beyond 900 m in the USA, the method of acoustic signal transmission was unsuitable for Australian conditions. The signal was drowned out by the in-hole gas flow noises as depths increased. Various alterations in the form of acoustic signal transmission improved the depth capacity to approximately 600 m. The survey results were processed on site to provide the drilling with the X, Y and Z co-ordinates to allow make borehole plotting and steering easier.

5.3.5 AMT DDM / DDM Upgrade

Advanced Mining Technologies (AMT) has developed the DDM to the stage where it only vaguely resembles the original Conoco DDM. Two forms, the DDM and the DDM Upgrade were used in gas drainage drilling operations. The DDM was activated by rotating the drill string and then transmitted information by varying the frequency of a carrier signal along the drill string to the UHU (Figure 5.3, AMT, 1993) at the drill site. The instrument processed the raw survey data using the tangential method of survey calculations to provide X, Y and Z co-ordinates for each survey point. The raw and processed survey data was recorded on the drilling log sheets and plotted on vertical profile and lateral deviation plots. The depth capacity was 700 m with the DDM with one borehole being achieved to 1000 m depth with the DDM upgrade. These systems, although greatly improved, still suffered signal transmission problems from in-hole noise at depth or after drilling through “sticky” ground or stone.

Figure 5.3: DDM-Upgrade instrument (AMT, 1993)
5.3.6 AMT DDM-MECCA

Ongoing development by AMT produced the DDM-MECCA (Figure 5.4, AMT, 1995) which was an improved DDM combined with the MECCA system (Figure 5.5, AMT, 2000) for communication along the drill string. The MECCA system was a cable installed in each drill rod and provided automatic connection to the DDM with each rod added to the drill string. All outside influences on activation and signal transmission were effectively eliminated. Until recently, this survey instrument had provided the best depth capacity of all instruments.

![Figure 5.4: Schematic of DDM-MECCA for directional drilling (AMT, 1995)](image)

![Figure 5.5: Modular Electrical Connected Cable Assembly (MECCA) (AMT, 2000)](image)

5.3.7 AMT Drill Guidance System (DGS)

The AMT DGS (Figure 5.6, AMT, 2005) was developed to be powered from the drill rig to eliminate the need for charging the UHU on the surface. It has greater computer capacity than the DDM and offers the ability to support add-on geological tools. The upgraded DHU (Figure 5.7, AMT, 2004) has the capacity to support additional geological sensors. Although it utilises the MECCA system for communication, the DGS uses a different communication signal form. The depth capacity with this...
instrument had been limited to 1500 m before recent upgrades extended the depth capacity to beyond 2000 m.

Figure 5.6: Schematic of DGS for directional drilling (AMT, 2005)

Figure 5.7: DGS down-hole assembly (edited from AMT, 2004)

5.3.8 DPI Directional Drill Management System (DDMS)

The DPI DDMS (Figure 5.8, DPI, 2012) has been developed based on the design of the DGS with upgrades incorporated to improve depth capacity and provide an operational gamma function to assist strata location. The DDMS utilised the MECCA system for
communication under the label of Rod Communication System (RCS). The depth capacity with this instrument has been proven to beyond 1800 m on a stone drilling project. As with any prototype, the system is undergoing refinements as weakness are identified through exposure to the robust conditions of the operating environment.

![Figure 5.8: Schematic of DDMS cable connection to RCS (DPI, 2012)](image)

**5.4 SURVEY CALCULATIONS AND DATA PLOTTING**

**5.4.1 Survey Calculations**

Several different sets of formulas have been written to convert survey data of depth interval, pitch and azimuth to X, Y and Z co-ordinates either in the north-south and east-west grid system or relative to a specific target azimuth. Each used trigonometrical formulas and has been labelled based on the aspects of the survey data they use. The simplicity varies as does the relative accuracy of each. Three methods used in the industry are covered below although there are several others listed which are more complex and offer more accuracy when dealing with large changes in pitch and azimuth between surveys.

The model in Figure 5.9 (DRILLINGFORMULAS.com, 2009) depicts two consecutive survey points along a borehole.
Where:

\( L \) = measured distance between surveys
\( I_1 \) = inclination angle (or pitch) at the initial survey point in degrees
\( I_2 \) = inclination angle (or pitch) at the second survey point in degrees
\( Az_1 \) = azimuth at the initial survey point in degrees
\( Az_2 \) = azimuth at the second survey point in degrees

**Average Angle Method**

The average angle method uses a chord which is the average of the inclination and azimuth between the two survey points. This method is reasonably accurate with the formulas listed below:

\[
\begin{align*}
North &= L \times \sin \left( \frac{(I_1 + I_2)/2}{\sin([Az_1 + Az_2]/2)} \right) \\
East &= L \times \sin \left( \frac{(I_1 + I_2)/2}{\sin([Az_1 + Az_2]/2)} \right) \\
VD &= L \times \cos \left( \frac{(I_1 + I_2)/2}{\cos([Az_1 + Az_2]/2)} \right)
\end{align*}
\]  

[Eq. 5.1]

[Eq. 5.2]

[Eq. 5.3]

**Balanced Tangential Method**

The balanced tangential method used the tangent from each survey point over the half distance between the points. This method was reasonably accurate and has been
preferred for the survey data processing spreadsheet and electronic survey instruments in recent times. The formulas are listed below:

\[ \text{North} = \frac{L}{2} \times [\sin(I1) \times \cos(Az1) + \sin(I2) \times \cos(Az2)] \quad [\text{Eq. 5.4}] \]
\[ \text{East} = \frac{L}{2} \times [\sin(I1) \times \sin(Az1) + \sin(I2) \times \sin(Az2)] \quad [\text{Eq. 5.5}] \]
\[ \text{VD} = \frac{L}{2} \times [\cos(I1) + \cos(I2)] \quad [\text{Eq. 5.6}] \]

This method was most widely used in the underground directional system. As well as the survey data calculations in the mine grid co-ordinates, the parallel set of calculations determined the position of the borehole relative to a target azimuth (X and Y in the horizontal plane) (Figure 5.10, Hungerford, 1989) and Z in the vertical plane relative to the height of the standpipe.

Alternate survey calculation methods with more complex formulas which offer greater accuracy over longer survey intervals with large changes in inclination and azimuth include the minimum curvature method and the radius of curvature method.

Figure 5.10: Lateral positioning terms for a directional borehole (Hungerford, 1989)
5.4.2 Surveying Position

With the introduction of “non-magnetic” Accu-dril DHMs, drilling departments were able to move the survey point from 6m behind the bit up to the back of the DHM (3 m behind the bit) with the expectation of being in a non-magnetic survey. This allows for better directional control (being only 3 m behind a potential error).

All survey calculations are based on a consistent curve between survey points (6 m lead, Figure 5.11, Hungerford, et al, 2012). This occurs when a consistent orientation has been used between survey points as is the case with surveys 6 m behind the bit. With surveying 3 m behind the bit and using 6 m orientations intervals, the survey point is located at the mid-point of an orientation interval. The change in survey results represents 3 m of the previous orientation and 3 m of the current orientation. The resultant survey calculations produce a nominal location of the borehole which relies on a random average of which the accuracy is not known. An interval of “flip-flop” drilling each side would be represented by relatively consistent surveys which do not show what could be 2.5 - 3.0° changes if surveyed at the transition points.

![Figure 5.11: Survey positions with 6 m orientation changes](image)

When Eastman survey data with extended survey intervals are input into spreadsheets for processing, the established practice was for intermediate survey values at orientation change points to be interpolated (based on DHM response curves) and input for improved accuracy.

5.4.3 Hand Plotting of Survey Data Using a Protractor

Plotting the profile of a coal seam at normal scale over the length of a borehole produces a low angled plot with a 2.5 – 3.0 m thick seam depicted as a very narrow segment and small changes in angle are difficult to distinguish. Since coal seams do not
usually have large gradients or variations in gradient, the profile of a coal seam along a particular line can be plotted on graph paper on which the vertical axis has been exaggerated. With the A4 graph paper used at the time, a vertical exaggeration of 10 was convenient. This meant the angles of boreholes plotted on this profile were also exaggerated to suit the plot.

To allow a borehole survey data to be plotted progressively underground, a system was designed to plot a two-point average of pitch (Pa) [Eq. 5.7], a simplified form of the average angle method between surveyed depths on a vertical exaggeration plot.

\[
Pa = \frac{(P1 + P2)}{2} \quad \text{[Eq. 5.7]}
\]

For plotting in the horizontal plane, the difference between the measured azimuth and the target azimuth (TA) along the intended line of drilling is calculated and referred to as the Drift (Figure 5.10). The same average angle method was used in the horizontal plane to calculate the two-point average drift (Da) [Eq. 5.8], to be plotted between the surveyed depths on an exaggerated scale lateral deviation plot.

\[
Da = \frac{(Az1 + Az2)}{2} - TA \quad \text{[Eq. 5.8]}
\]

A vertical exaggeration protractor was provided and used to plot the two-point average on the vertical profile plot. The protractor (Figure 5.12) was a sheet of overhead transparency on which was plotted the exaggerated vertical displacement angles for each degree to 10°. The incremental degree lines were calculated as the vertical displacement achieved along the length of an angled borehole [Eq. 5.9].

\[
\text{Vertical Displacement} = (L2 - L1) \times \sin^{-1}(P2) \quad \text{[Eq. 5.9]}
\]

Thus the profile plot is of the vertical displacement on the y-axis against the borehole depth (as distinct from the horizontal advance of the borehole) on the x-axis.
The vertical displacement and lateral deviation plots were plotted on the standard A4 graph paper of the time (250 mm x 160 mm x 2 mm squares) with the longer 250 mm side represented 250 m of borehole depth while the shorter 160 mm side represented 16 m of vertical displacement (Figure 5.13). A similar arrangement was used for the lateral deviation plots (Figure 5.14).
To plot a two-point average value, the protractor is positioned with the apex at the previous survey point and adjusted so the average pitch (Pa) reading on the protractor is aligned with a horizontal line drawn from that survey point. A line is then drawn along the base line of the protractor from the previous point to the current survey point.

The lateral design of boreholes was limited to +/- 10° to remain within the limitations of the protractor. This particular protractor was suited only to plots with a vertical or lateral exaggeration of ten.

### 5.4.4 Borehole Design and Survey Data Processing

A spreadsheet (Figure 5.15) was developed originally to process the survey data and present that data in both table and graph form. Graph plots are of both profile and lateral deviation versus borehole depth over the length of the borehole and over 250 m intervals.

The spreadsheet has been developed into a borehole design system for each borehole. The lateral alignment of each borehole can be designed and incorporated on a mine plan with seam contours. Using the depths along the borehole at the contour intersection points, an estimate of the seam profile along the line of the proposed borehole can be established. Since the interpretation distances between surface boreholes are considerable and the contours not highly accurate, the profile plot provides a profile trend (maroon lines, Figure 5.16). The design information is presented as the...
spreadsheet data sheet (Figure 5.15) and in graph form over 250 m intervals (Figure 5.16 and Figure 5.17) so the borehole can be progressively plotted at the drill site.

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Figure 5.15: Survey data spreadsheet data page

The profile plot is adjusted as drilling progresses and the seam profile is defined (Figure 5.16). The lateral deviation of the borehole is plotted with drilling intended to follow the design line (red line, Figure 5.17)

Figure 5.16: Design and as drilled profile plot over 250 m interval

MG5, 1ct DH13 - Borehole Depth (m)
For boreholes with large lateral deviation and drift, the scale does not allow an acute awareness of the borehole’s azimuth relative to the design line. An additional plot of lateral deviation versus azimuth (Figure 5.18) is provided to show the azimuth required at a particular lateral deviation to curve the borehole adequately to stay on line. As an example, Figure 5.17 shows a borehole close to and inside the design line and very close to the required azimuth at a borehole depth of 320 m (-190 m Lateral Deviation). An incorrect DHM orientation at that point (Figure 5.18) caused the borehole to eventually over-shoot the target off-set (Figure 5.17).
5.4.5 Plotting of Processed Survey Data

With the development of readily available computers and printers, graph paper was printed to various scales. In these cases, trigonometry calculations are provided by the computer incorporated in electronic survey instruments. The vertical and lateral values for each survey point are calculated and plotted at the relevant borehole depth. The vertical scale is limited to a 24 m range for steeper seam profiles. If the seam extended outside the 24 m range over the length of the X-axis, the scale is reduced to 16 m and several plots used to cover the profile. Large lateral deviations with deviation angles of up to 90º (although logically limited to 54º for long-holes) are catered for with the scale changed to include a 50 to 60 m range on the lateral axis of each sheet.

5.5 MAGNETIC DECLINATION

For the sake of being able to determine the location and alignment of all features on the earth’s surface, the earth’s surface is divided into a north-south/east-west grid system from which northing and easting co-ordinates are devised. All plans of the earth’s surface use this grid system and the location on the earth’s surface can be defined by co-ordinates and any direction defined as a grid bearing. Plans of underground mines are included on the earth’s grid system. The angle between any line on the grid system and the north/south alignment is referred to as a grid bearing or azimuth.
Survey systems measure the alignment or angle of a borehole relative to the alignment of the magnetic field at that location (magnetic bearing). But the alignment of the earth’s magnetic field varies over the earth’s surface (and is continually changing) so the magnetic declination varies (Figure 5.19, NOAA/NGDC and CIRES, 2010).

The relationships between True North, Grid North and Magnetic North are shown in (Figure 5.20). With underground borehole survey systems providing magnetic measurements, all measurements have to be converted to the mine grid system for boreholes to be plotted on mine plans. To convert the magnetic bearing to a grid bearing or azimuth, the difference between the grid and the magnetic field has to be known or measured. Although the difference between grid north and magnetic north is listed as Grid-Magnetic Declination, this difference has commonly been incorrectly referred to as Magnetic Declination (MD). The term Magnetic Declination is now used to describe the Grid-Magnetic Declination.
Eastman instrument readings are converted to grid readings using the stated grid-magnetic declination from the local topographic map of the area of the mine.

This does not consider the accuracy of the compass angle card installed in the compass angle unit of each instrument. When designing a proposed borehole location and alignment, the desired grid bearing is determined and converted to a magnetic bearing for the driller to follow. At a later stage, the calibration sites for electronic instruments were used to check Eastman instrument readings.

As more accurate electronic survey systems became available, the trigonometrical calculations provided by the system could include a conversion of each magnetic reading to a grid reading. A standard procedure was established to calibrate each electronic instrument to determine the magnetic declination of that specific instrument at that location. With all instruments, there is variability in the mounting of the three magnetometers which read the angle of the magnetic field in which they are positioned. This gives slightly different readings depending on the instrument and the orientation of that instrument.

To determine the magnetic declination of that instrument at that site, a calibration site was established consisting of two posts with “V” notches cut in the top, cemented into
the ground with the “V” notches usually in a North-South alignment (Figure 5.21). Additional calibration posts can be provided in the East-West alignment (Figure 5.21). The grid bearing of the instrument mounted inside the non-magnetic BeCu rod lying across the posts in the “V” notches was measured by the mine surveyor. This grid bearing (Grid) was recorded for the calibration site and checked yearly or whenever the posts appear to have been disturbed. The selected location should ensure the site was free of any magnetic influences which would adversely affect any calibration results. An alternative to the posts cemented into the ground was a set of survey pins with a string line between the two. The instrument was then mounted in “V” blocks on the ground (Figure 5.21), with the instrument positioned parallel to the string line and the computer located a comfortable distance from the instrument (Figure 5.23) to avoid magnetic influences.

Figure 5.21: Calibration site using cemented in posts
Calibration of the instrument initially required setting the instrument’s magnetic declination value to zero so all measured readings were unconverted magnetic readings.
With the instrument installed on the calibration station, a disc (Figure 5.24, AMT, 2001) was attached to the BeCu rod. Surveys were then taken with the instrument rotated through the 24 positions indicated on the disc. There was usually a slight SIN curve variation in the readings (Figure 5.25) which varied between instruments and locations as indicated.

Figure 5.24: Magnetic calibration disc (AMT, 2001)

Figure 5.25: Variation in magnetic readings during the calibration procedure
CHAPTER FIVE
Design and Development of Borehole Survey Systems for In-seam Drilling

The average of these 24 readings (Av) was subtracted from the grid bearing (Grid) for the site [Eq. 5.10]. The resultant value (including the sign +/-) was the MD for the instrument at that site.

\[
\text{MD} = \text{Grid} - \text{Av} \quad \text{[Eq. 5.10]}
\]

For MD displayed on world maps (Figure 5.9), the MD has been determined by the equation [Eq. 5.11]:

\[
\text{MD} = \text{Av} - \text{Grid} \quad \text{[Eq. 5.11]}
\]

The sign of the MD displayed on the world map was opposite to that calculated for use by the software in the electronic survey system. The equation [Eq. 5.12] installed in the electronic instruments to calculate the azimuth (Grid Bearing) was:

\[
\text{Azimuth} = \text{MB} - \text{MD} \quad \text{[Eq. 5.12]}
\]

The preferred alignment of the calibration station is to be the same as that of the predominant direction of drilling underground at that mine. Common practice is to calibrate on a north-south alignment. Some sites also include an east-west alignment which has identified a slight magnetic influence problem with the instruments. With the slight SIN curve variation identified during the calibration process, the system relies on the random spread of instrument orientations to provide an averaging of the slight error. In cases where better accuracy is desired (for the intersection of a target point), the practice is suggested to take each survey to determine the orientation of the instrument then rotate the instrument to the position which corresponds to the most accurate survey on the SIN curve variation (60°, 180°, 240° or 300° for the lower example in Figure 5.25) to record the survey to be used in the calculations.

### 5.6 INFLUENCES OF BOREHOLE SURVEYING ACCURACY

#### 5.6.1 Magnetic Influences

The predominant form of directional surveying in underground drilling is by measuring the magnetic alignment (bearing) at points along the borehole. This is done by measuring the alignment of the survey instrument relative to the magnetic field in the area. These readings are then converted to grid bearings (azimuth) to be incorporated on mine plans.
The accuracy of these measurements relies on accurately measuring the difference between the magnetic field in the area and the grid system, referred to as MD. But the magnetic field at the calibration station or in the area of drilling may have been altered by magnetic influences. To improve the accuracy of borehole surveying and thus provide accurate positioning of the boreholes on mine plans, potential sources of magnetic influence should be identified and either eliminated, avoided or at least catered for.

Calibration sites should be established in areas free of magnetic influences that will adversely affect the MD process and lead to wrongly located boreholes. The preferred location for a calibration station is in relatively undisturbed bushland with vehicle access. A variety of environments have been identified as causing magnetic influence:

- Buildings, mesh or sheet fences, compressed air and water pipes which all contain steel.
- Car parks, which change in magnetic influence with the number and relative location of parked cars.
- Major roads particularly which may have cars or haul trucks passing during a calibration.
- Concrete structures, slabs and culverts usually contain steel reinforcement.
- Railway lines have a magnetic influence which increased as a train passes, particularly a slow moving coal train during loading.
- Overhead high tension power lines generate an electromagnetic field which altered the magnetic field in the area.
- Buried, armour protected power cables.
- Flat playing fields were convenient but may be reclaimed land. One particular site contained buried car bodies with substantial changes in magnetic field across the various locations tried on the field.
- Vehicles used to transport the instrument to the station need to be removed 20-30 metres from the site.
- Steel capped boots should not be near the front of the instrument.
- The UHU/laptop used for the survey needs to be removed to the limit of the cable being used.
5.6.2 Survey Instrument Accuracy

Most instruments were quoted as having an accuracy of +/- 0.5° which was equivalent to +/- 8.7 m over 1000 m. The horizontal location of any borehole plotted on a graph or mine plan, although represented by a line, can be anywhere within an area of accuracy 0.5° either side (Figure 5.26, Hungerford, 2003). When planning long boreholes in close proximity to roadways (or proposed roadways), recognising the zone of accuracy needs to be part of the design process.

![Figure 5.26: Zone of accuracy with lateral borehole location (Hungerford, 2003)](image)

5.6.3 Drill String Influences

Sources of magnetic influences were included in the DHM, drill rods and inherent in the survey instrument. The inherent sources were discussed in the previous section. The drill string (being steel) has always been known as a source of magnetism. But rather than the proximity of the nearest steel rod, the influence of the whole length of steel rods on the magnetic field has been regarded as more influential. With the sensors located at the front of the survey instrument in a 3 m BeCu rod with a 3 m BeCu rod behind, the steel rods are 6 m removed and regarded as having limited to no influence.

Non-magnetic DHMs have come in a variety of versions. The common configuration has been the rotor and stator made of non-magnetic material and the front section of bent housing, universal joints and bearing pack made of steel. K-monel was used
initially but suffers from limited strength, thread galling and has developed some magnetism. An analysis of the magnetic effect from a DHM was completed using a 73 mm diameter, BeCu Accu-dril DHM with a Surtron Champ instrument (Hungerford, 1995). The survey instrument was positioned at a full range of alignments around the 360° of the compass. At each position a survey was taken with the DHM removed, then located in-line at zero and then 3 m from the front of the survey instrument. The first survey (with steel removed) was regarded as the reference survey for the two surveys with magnetic influence (Figure 5.27).

![Graph](image)

Figure 5.27: Influence on magnetic readings behind a non-magnetic DHM (Hungerford, 1995)

When the exercise was repeated using a steel Accu-dril DHM and the DGS, the values were reversed (Figure 5.28). Again the influence was restricted in the north-south alignment and at the maximum in the east-west alignment. Influence was negligible between 3.0 m and 6.0 m so 3.0 m was taken as the base line. Although small, the influence at 2.0 m, which represents the positioning behind the steel front end of a non-magnetic DHM, was 0.1° in the east-west alignment and should not be discounted.
The current practice is for the stator case to be made of BeCu and the rotor chrome coated BeCu. The back sub was non-magnetic stainless steel but was found to develop slight magnetism through either the machining process or multiple uses. This was suspected through variable survey results with different DHMs and eventually identified by introducing the DHM to the front of the survey instrument during a calibration. The magnetic survey results varied by 2° without the thread being engaged (still 100 mm from the final joined position). This sub has now been replaced with a BeCu version.

Steel DHMs pose an obvious source of magnetic influence so surveying directly behind a steel DHM will produce inaccurate surveys. Common practice was to move the survey instrument back 3 m from the back of the DHM with the inclusion of a 3 m BeCu spacer rod. This also requires improved skills to steer the borehole with surveys 6 m behind the bit rather than the usual 3 m. At one mine, a steel DHM was introduced unknowingly with the resultant off-line drilling and the subsequent unexpected intersection by mining causing problems. From Figure 5.28, this error would have been greater than 7.5° in an East-West alignment. Steel DHMs are now banned from the mine through that incident and the limited driller skills available to operate with surveys 6 m behind the bit.
5.6.4 External Environmental Influences

The use of electronic surveys instruments has assisted in the ability to detect the presence of magnetic anomalies. The set of three magnetometers mounted in the instrument produced raw data lasted as Mx, My and Mz with the square root of the sum of squared M values listed as Mu. Mu was effectively the vector of the three M values. The Mu value had to be calculated by hand from the raw data of the second page of the DDM-MECCA instrument but is recorded by the DGS system and available to the driller on site. This value remained constant for the magnetic field. Slight but noticeable changes in the Mu value have been used to identify a change in the magnetic field had occurred, indicating the presence of a magnetic influence. No indication of this is possible by viewing the actual magnetic survey results.

If a copper or composite (non-magnetic) standpipe was used, the first survey at 6 m can be at the end of the standpipe but if a steel standpipe is used, the first survey should be with the survey instrument sensor located at least 6 m past the end of the standpipe. If the standpipe is extended or steel casing is to be installed, the pilot hole should be drilled and surveyed before reaming and installing the standpipe/casing. Drilling and surveying can continue from the pre-surveyed section of hole.

Starting a borehole at a low angle to the roadway should be avoided where possible as it puts an extended length of the start of the borehole within the influence of the steel support system.

This effect was evident with a flanking in-rush protection borehole drilled 23 m out from old workings and 23 m from the rib-line of the proposed development (Figure 5.29, Hungerford, 2008). The direction of the drilling was almost due south. The borehole was subsequently intersected by mining at a location 800 m from the standpipe at a point approximately 23 m off-line (Figure 5.30) representing an angular error of 1.65°. The effect must have been that the magnetic surveys had read further to the right than actual. The borehole was steered more to the left with the perception that it was being kept on line.
Igneous intrusions in the form of sills, dykes and siderite domes are known to have some level of magnetism (they are usually identified by airborne or ground magnetometer surveys over suspect areas) (Stanley, 1991). Dyke intersections are usually planned and anticipated prior to exploration drilling. Penetration is either straight through with directional drilling, using orientations which maintain pitch and azimuth. If azimuth readings are affected, drilling continues until the readings return to normal. The interval from the last good survey before the dyke to the first good survey after can be bridged before adding the borehole to the mine plan.
Ironstone bands and ironstone knobs can alter one or several magnetic readings if the instrument is in close proximity.

Not all unexpected intersections of boreholes by mining are attributed to magnetic influences. When drilling flanking boreholes ahead of development with the occasional branch directed across the proposed development, an error in rod count when pulling back to branch can put the branch in the wrong location. In the case shown (Figure 5.31), the rods were pulled out an additional 159 m more than planned for the branch, putting the actual location shown in red where it was intersected by mining.

![Figure 5.31: Plan of incorrect location of borehole due to a rod count error.](image)

5.7 SUMMARY

To determine the location of boreholes drilled in and around coal seams, boreholes need to be surveyed either with intersection by mining or by surveying along the length of the borehole post drilling or progressively during drilling.

Surveying along the length of a borehole was originally by the Eastman single-shot instrument. With directional drilling, the orientation of the DHM was also provided for steering purposes. For surveying during drilling, the instrument was pumped down the drill string and retrieved on a wire-line. This process suffered from progressively increasingly delays to drilling as borehole depths increased.

Several multi-shot instruments were introduced with varying success at speeding the post drilling surveying process. The Conoco DDM was eventually modified and improved to suit the gassy conditions experienced in Australian coal mines. This was assisted by industry operators willing to use the developing instrument and suffer delays.
as problems were found and rectified. This led to the development of the DDM-MECCA capable of surveying to depths beyond 1500 m. The latest class of survey instruments include the AMT DGS and DPI DDMS powered from the electrics of the drill rig and offer a platform for additional geophysical tools.

Progressive plotting of surveyed boreholes was introduced with the use of vertical exaggerated scales and matching protractors using two-point averaging between survey locations. With electronic instruments, more accurate trigonometry calculations were available with the Tangential Method used initially before the more accurate Modified Tangential Method was adopted.

Surveys are measured as magnetic reading which must be converted to grid values for reference to the mine plan system. Before any surveying at a mine site, the electronic instruments must be calibrated to determine the variation (or magnetic declination) between magnetic and grid at that site and the conversion installed in the instrument. Then all readings are displayed as grid readings.

But being an instrument which measures magnetic alignment, it is exposed to magnetic influences which may alter the Earth’s magnetic field in the vicinity of the instrument; giving a false reading. Magnetic influence can also be present in the drilling environment through the proximity of steel in adjacent roadways, bogged rods, ironstone bands or magnetism from igneous intrusions. These influences are noticeable in some instances and can be catered for while others are only discovered through unexpected intersections of the borehole with mining.

The accuracy of electronic instruments is usually defined as +/- 0.1° in Pitch and +/- 0.5° in Azimuth. This accuracy relies on an averaging of positioning of the surveying instrument around its axis and the location of the survey point within the curves of a directional borehole.

A spreadsheet has been developed to calculate the X, Y and Z co-ordinates of boreholes for plotting on mine plans. This spreadsheet has become a tool to design boreholes which assist drillers steering the borehole along a planned path in the lateral plane while providing an estimate of the seam profile.
CHAPTER SIX – IN-SEAM DIRECTIONAL DRILLING PRACTICES

6.1 INTRODUCTION

Underground directional drilling was introduced to the Australian coal industry in the mid-1980’s with basic applications established through a research and development project (Allan, 1984). A subsequent project (Hungerford, et al, 1988a) described the development of directional drilling as an in-seam drilling application and defined the operational and steering capabilities of directional drilling which are presented in this chapter. These were incorporated in a training manual (Hungerford, 1989) and presented as classroom and mentoring training packages to the drilling personnel of mines undertaking gas drainage. The drilling practices at each mine then evolved to cater for local requirements and included in operational manuals (Appin Colliery, 1996).

The generic requirements of drill site layout have been defined with variations to suit the size and shape of equipment, ventilation and interaction with mine operations. Control of the borehole has been designed with the use of standpiping to manage expected pressures and rib conditions. These are presented in this chapter although each mine has developed their system to suit local requirements.

A variety of strata and conditions adverse to stable drilling have been encountered. Through direct involvement and in consultation with in-mine drilling operations, means of recognising, planning for, managing and/or avoiding unstable environments have been defined and are presented in this chapter.

6.2 OPERATION OF A DHM FOR DIRECTIONAL DRILLING

Very little was initially known in Australia on operating a DHM for underground directional drilling so it was a case of learning how a DHM operated and what the required parameters were in terms of water flow and pressure. Drilling started with slow feed rates to gain an understanding of the changes in water pressure as the bit cut the face and was fed in at varying rates. All these aspects were recorded to learn how to improve directional drilling and to form guidelines with which to train drillers in the future.
6.2.1 Water Idle Pressure

High pressure water was pumped down the drill rods and through the DHM to provide the rotation of the bit for drilling. At the control panel, the water pressure gauge registered the pressure of that water and was understood to represent the flow resistance through the fittings, water hose, water swivel, drill rods and through the DHM to provide rotation free of drilling load. In underground drilling operations in Australia this pressure was labelled the Idle Pressure since it represented the water pressure in the system when the bit was spinning but not cutting (idle). Surface drilling and US underground operations use the terminology “off-bottom pressure”.

This pressure has been recorded and was found to be an indicator of any changes in the state of the DHM as well as monitoring any developing in-hole blockages and the progressive increase due to length of drill string (Figure 6.1).

Before DHM test benches were readily available, each DHM was tested out of the borehole by pumping a water flow of 200 l/min through it and recording the idle pressure to assess the interference fit in the power section as an indication of the torque capacity of that DHM. Using a standard six metres length of one inch hydraulic hose to the water swivel, an idle pressure above 2 MPa indicated a good DHM which any idle pressure below 1.7 MPa indicated the DHM had limited torque capacity.
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When water flow is applied to the drill string, water pressure progressively increases as the drill string fills. As the water reaches and starts passing through the DHM, the water pressure spikes as the DHM starts running then settles down to a steady idle pressure. This spike is still evident but not as pronounced if the drill rods are already full of water. It is also not pronounced if the DHM has produced a relatively low idle pressure when tested out of the borehole.

6.2.2 Water Drilling Pressure

As the drill bit is advanced into the face of the borehole, the DHM produces torque for the bit to cut the face at the applied rate of penetration. The water pressure increases due to the additional load required to produce the drilling torque. The water pressure during drilling has been recorded (Figure 6.1) and represented the idle pressure plus the pressure required to drive the DHM during drilling. This difference has been referred to as the Differential Pressure.

Drilling pressures which resulted from a range of penetration rates in the coal (or stone strata) being drilled were established. Variations in pressure were identified as indicating intersections with strata of differing strength or hardness. As harder strata are intersected and the water pressure started to increase, the feed rate was reduced to maintain a steady water pressure. Continued drilling into harder strata without reducing the penetration rate results in an increase in the water pressure to the point where the DHM stalls and the relief valve on the water pump releases. The flushing water is monitored for a change in the colour to confirm a change in strata.

The introduction of DHMs with higher torque capacity has produced slightly higher idle pressures and a capacity to drill at higher drilling pressures without stalling. A drilling rate that produces a water drilling pressure 1.0 MPa above the idle pressure has been able to be maintained at shallow depths without experiencing substantial spikes in pressure and potential stalling of the DHM. In consistently strong strata, higher pressures have been maintained without substantial fluctuations. When drilling in coal of variable strength or fractured coal, it was found the DHM produced spikes in pressure which were attributed to rapid increases in torque loading. When these spikes were large enough to exceed the water pump pressure setting, the DHM stalled. This environment forced a reduction in penetration rate to limit the magnitude of the spikes to avoid stalling.
The same events applied as borehole depths increased. In the initial long borehole to 1000 m, surging started from a depth of 60 m and progressively increased, forcing the feed rate to be progressively reduced. The surging was attributed to the effects of friction in the borehole as the borehole depth increased and buckling of the rods through the bends in the borehole. This produced increased surging of the feed which was magnified over the length of the rod string resulting in fluctuating torque loading and eventual stalling. Reducing the penetration rate reduced the magnitude of the surges. The progressive reduction in penetration rate in a long borehole is evident in the reduction in the differential pressure in Figure 6.1. For some long boreholes, the surging has been so severe that repeated stalling even at very slow penetration rates prevented further advance.

### 6.3 STEERING A DHM

The steering of the DHM in the initial directional drilling was experimental and based on simple principles of orienting the bend on the motor in a direction and recording the survey results to determine the effects of each orientation. The electronic survey instrument was positioned 6 m behind the bit so drilling with specific orientations over 6 m intervals allowed assessment of the change in azimuth and trajectory produced with each orientation. These changes were referred to as the lateral and vertical responses and plotted against the orientation used as response curves. Drilling with a combination of a 73 mm diameter DHM, 0.75º bend and 89 mm diameter bit produced the response curves shown in Figure 6.2 and Figure 6.3 (Hungerford, et al, 1988a).

![Figure 6.2: Vertical response curve, 0.75º bend – 89 mm bit (Hungerford, et al, 1988a)](image-url)
It was evident that the plots were off-set approximately 30° (or one hour) clock-wise. This indicated a reactive torque from the DHM was being distributed back through the drill rods, producing an anti-clockwise elastic torsional flex of the rods. Increasing or decreasing the penetration rate would have a direct effect on the torque produced by the DHM and thus the magnitude of the elastic reactive flex in the drill rods. When drilling ceased, the torque was released and the orientation of the DHM returned to its static position before the subsequent survey. With this knowledge and the response curves, a clock-faced steering guide was produced (Figure 6.4, Hungerford, 1989) to assist in training drillers in steering the DHM. Both response curves and the “clock-face” allowed the drillers to predict what the survey results will be at the bit and plan what orientation to use to produce a desired result over the next 6 m of drilling.
With the introduction of the 4/5 lobed Accu-dril DHM, the bit diameter was increased to 96.1 mm in an attempt to reduce friction in the borehole which was causing surging, limiting drilling rates and borehole depths achieved. The magnitude of the bend was increased to 1.25° to provide an additional ability to climb. This was greater than the minimum required but provided additional deflection for control in broken or cleated environments where changes in either vertical or lateral direction were found to be reduced. Electronic surveying was not available to provide an indication of lateral and vertical response over 6 m intervals so indicative response curves (Figure 6.5 and Figure 6.6, Hungerford, 1989) were produced as guides for drillers to maintain directional control with wire-line surveying. The steering guide was adjusted to suit the greater deflections and increased reactive torque of the higher torque of the Accu-dril DHM. The response graphs and the steering guide were included in both the Directional Driller Training Manual (Hungerford, 1989) and the Appin drilling manual (Appin Colliery, 1996).
The change reduced in-hole friction and thus surging feed and allowed boreholes to be drilled to 1500 m at improved penetration rates.

The development and utilization of the DDM-MECCA survey instrument reintroduced 6 m survey intervals which allowed response curves to be produced for the 96.1 mm bit and 1.25° bend combination (Figure 6.7 and Figure 6.8, Hungerford, et al, 2012). The plots showed much more variation that initially expected. This was attributed to variations in coal strength due to banding which exists in coal seams, cleating of the coal, occasional broken coal or variations in torsional flex in the drill rods or a combination of any of these.
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Figure 6.7: Vertical response - 1.25º bend with 96 mm bit (Hungerford, et al, 2012)

Figure 6.8: Lateral response - 1.25º bend with 96 mm bit (Hungerford, et al, 2012)

The response curves provided an indication of the available curve rates which could be used in borehole design. Reduced curve rates could be used to control the borehole laterally while still having adequate vertical control of the borehole. A lateral curve rate of 1.5 deg/6 m was adopted as a general design limit although this was reduced when cleat direction was known to limit the rate at which boreholes could be turned laterally.

6.4 DEFINITION OF THE SEAM PROFILE

Although directional drilling allows control of the borehole in both vertical and lateral planes, it does not indicate where the borehole is located in the strata. To achieve this and be able to follow the seam, the profile of the seam along the line of drilling needs to be defined.
In the early days of in-seam directional drilling, only an indication of whether the seam dipped upwards or downwards and the expected dip were offered by mine surveyors or geologists. It was left to the driller to progressively define the seam profile as drilling advanced. This was achieved by intentionally directing the borehole upwards to intersect the seam roof after approximately 50 to 70 m of drilling (Figure 6.9). A line of the seam roof was then drawn from the drill site to the intersection point and extrapolated for another 70 to 100 m. Another line was then drawn parallel to the seam roof with the seam thickness drawn below to indicate the line of the seam floor. The drilling was then continued by pulling back 6 to 18 m to branch and drill forward assuming the extrapolated roof position. After drilling forward another 70 to 100 m, the borehole was again directed upwards towards another roof intersection and the process continued.

The preference is to regularly intersect the seam roof for seam definition as only 6 to 24 m is usually required to be redrilled after retreating for a branch. If the seam floor is intersected, the bit has to be pulled back further to reach a suitable branch point which will not direct the future drilling back into the floor.

When drilling adjacent to and in relatively close proximity to a previously defined seam profile, that profile can be used with slight adjustments for seam dip. Since the seam profile does not need to be progressively defined, branching is greatly reduced with improved drilling rates in subsequent boreholes drilled in the same direction.
When drilling in thick seams or in a seam with a distinctive band, alternative targets can be used for seam definition. With a 10 m thick seam, a 3 m mining section against the seam floor and complicated geology high in the seam, the seam floor can be targeted (Figure 6.10). A branch point is created before diving over towards the floor. The drilling can then continue from the branch point. In the thin Balgownie seam, a distinctive shale band in the middle of the seam was regularly penetrated to maintain the borehole within the seam. With low angle intersections, the bit can be deflected out of the stone and drilling continued to avoid branching if the stone/coal interface is known to be stable. If it was unsure whether the intersected stone was roof or floor, the “rule of thumb” on deciding which drilling attitude to take was “if in doubt, drill up”. If the stone was roof, continued drilling proved that and the bit was pulled back a short distance to branch. If the stone was floor, continued drilling deflected the bit back into the seam for continued drilling with no loss of distance.
If drilling in a mining section low in a thick seam with complicated or unstable strata higher in the seam, the floor of the seam has been targeted for seam definition. At regular intervals, a sharp rise is drilled to provide a branch point before diving over into the floor of the seam. Drilling continues after branching from the branch point provided (Figure 6.10). In most cases, the bit is withdrawn approximately 30 m to branch whereas that distance would have been much greater if the roof of the seam was targeted.

If a significant, consistent and well defined stone band has existed within a thick seam, this band has been used as the target for progressive profile definition.

6.5 BRANCHING OF BOREHOLES TO CONTINUE DRILLING

6.5.1 Branching Practices

As well as having the ability to steer the borehole with directional drilling, a DHM can cut into the side and bottom of a borehole to form a new drilling face. In the Australian underground coal drilling industry this is referred to as branching. In surface drilling and in the coal industry in the USA it is referred to as side-tracking. Branching gives the ability to continue drilling after the borehole has been stopped for some reason.
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The preferred point to start a branch is a section of borehole which is curving upwards or has a significant lateral curve. Initially, branching was achieved with the bend of the DHM facing down at six o’clock with a 3° drop experienced over 6 m from the branch point. Due to being forced to branch close to the seam floor, lateral curves were used to assist branching attempts and found to be as effective for general branching use.

Now branching utilises both vertical and lateral curves with gravity dictating that a branch will always start in the lower section of a borehole and be angled downwards initially and laterally to one side. As such, the DHM is orientated within the range of 3:30 to 8:30 on the clock-face (Figure 6.4).

The feed rate is limited to less than 0.2 m/min until fluctuations in water pressure and increased colour in the return water indicate that a lip has been cut and has been propagated to a new face. The initial lip is cut by elastic deformation of the DHM pushing the bit against the side of the borehole (Figure 2.8, Chapter 2). As the lip is advanced, the elastic deformation of the DHM is eased. Further propagation of the lip to an increasingly larger face (Figure 6.11 and Figure 6.12) relies on slow feed with the bend and weight of the DHM holding the leading edge of the face of the bit against the lower side of the borehole. After a full face is thought to be established, the orientation is set for ongoing directional control with care taken not to break back into the original borehole. Normal drilling can then be resumed at appropriate feed rates and orientations of the DHM.

Figure 6.11: Profile through a branch
The three-degree change in borehole direction is shared between vertical and lateral aspects depending on orientation of the bend. The vertical location in the seam and angle of the section of borehole chosen for the branch is chosen to allow the borehole to continue comfortably within the seam. Branching low in the seam or angled steeply down usually directs the borehole back into the floor stone. If branching from a section of borehole that has been following a lateral curve, the preferred branch direction is towards the inside of that curve to maintain lateral control.

6.5.2 Stability of Branching Points

The branch point has been regarded as an area of instability due to the increased cross-sectional area opened by the borehole (Figure 6.13 and Figure 6.14). This is likely in areas of broken coal where large pieces can drop in from the top of the borehole. In stable areas, the only likelihood of instability would be due to high vertical or lateral stresses. Some branch points when exposed by mining have been enlarged due to borehole collapse.
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Figure 6.14: Cross-sections showing rods in the borehole through a branch

With the increased cross-sectional area through the branch (Figure 6.14), the average flushing velocity is substantially reduced (Figure 6.15). This has been attributed to recurring blockages at a branch point when fractured coal is drilled further into the borehole. Larger pieces flushing along the borehole are settled out of the flow with the reduction in average velocity. When they accumulate and are eventually flushed back into the reducing section (or wedge) towards the one metre point of the branch (Figure 6.13), a blockage is formed. This is usually complicated when pulling the rods backwards, as the friction of the rods drags the larger pieces back into the wedge and tightens the blockage. The bit has to be worked back through the branch point to break up the larger pieces before continuing with the drilling.

Figure 6.15: Average velocity of flushing fluid through a branch with rods in place

Although the cross-sectional area is increased with the removal of the drill rods, the combined gas flow from each leg into a branch provides an increase in average velocity. This usually prevents settlement of debris and prevents blockages forming in the area of the branch after the drill rods have been removed.
6.6 DRILLING INFORMATION

6.6.1 Recording Drilling Information

With any drilling, the minimum data recorded is the rod numbers, borehole depth and some comment on the strata drilled. In the initial stages of directional drilling research, numerous forms of information on the drilling was recorded to enable directional drilling practices to be defined and the action and reaction behaviour of most aspects identified. With current directional drilling, the detail to which data is recorded is dependent on the application and how much particular information is required to be recorded to satisfy the project goals. In gas drainage applications associated with Outburst Management Plans (OMPs), the detail of record keeping is defined by what information is required to satisfy the OMPs. Current in-house gas drainage drilling projects may only record rod numbers, depth and survey results with comments made on the drilling environment and with stone intersections or boggy ground intersected (Figure 6.16) while some are more regimented with data collection on drilling parameters and conditions (Figure 6.17).

The comments of the drilling environment are added to mine plans by the mine surveyor when processing the survey data for approval-to-mine assessments. Recording the survey data provides the driller with a readily accessed record of the hole when determining branch points. The hard copy is a layover from the early days of development of the electronic survey instruments when digital data was occasionally lost if an instrument broke down or malfunctioned.

![Figure 6.16: Sample directional drilling log with specific OMP requirements](image-url)
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Figure 6.17: Sample directional drilling log with specific OMP requirements

With long-hole directional drilling, recording all pressures, flows and orientations associated with the drilling provides data which defines the performance of the equipment, and the state of the borehole and allows an assessment of depth capacity. These are recorded progressively with each survey at 6 m intervals on a DHM drilling log sheet (Figure 6.18) with experienced drillers noting variations of water or feed pressure and recording the depth stone was intersected. A line is left blank to more clearly define a branch. The notes column is used to record the orientation the drillers used with the DHM.

With the trend to survey only 3 m behind the bit, data has not been available to assess changes over 6 m intervals with the same orientation (Section 5.4.2). To gather that information when assessing in-hole friction effects, the drillers were asked to take and record an intermediate “check-shot” survey (Figure 6.19). The drillers also measured and recorded the feed rate in the comments column.
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6.6.2 Processing Drilling Information

The survey data is processed directly from the downloaded data from the survey instrument as explained in Section 5.4.4. The borehole location is plotted on the mine plan. The seam profile is defined with both survey data and drilling records to define
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when stone roof and floor have been intersected. Vertical and lateral response curves (Figure 6.7 and Figure 6.8) are provided from survey data to assess directional control in the drilling environment and with the particular bend configuration being used.

The two numbers recorded in the Feed Pressure column are the hydraulic Feed Pressures and the hydraulic Hold Back Pressures with each being required to calculate the thrust being used to feed in the rods. The thrust trend over the length of the borehole (Figure 6.20) shows the progressively increasing effects of in-hole friction and allows the maximum depth capacity to be estimated for each borehole (Hungerford, et al, 2012). In this case, repeated stalling from surging prevented the maximum pressure of the drill rig to be utilised and the estimated depth to be reached.

![Graph](image)

Figure 6.20: Plot of thrust pressure versus borehole depth relative to maximum capacity
(Hungerford, et al, 2012)

The numbers recorded in the Water columns are as discussed in Section 6.2. When plotted over the length of the borehole (Figure 6.1), the trend of increasing idle pressure as depth increases is evident. The DHM differential pressure is reduced progressively with depth as in-hole friction causes increased surging of the feed.
6.6.3 Shift Activities Recording, Processing and Presentation

With the requirements of OMP’s for data recording, each drilling department has developed their own way of reporting drilling parameters. To schedule and manage the maintenance of equipment and report on the time utilisation and delays incurred, drillers report on their activities throughout each shift as a shift report referred to as a Drill Operator’s Report (DOR) (Figure 6.21). The data from this shift report, with the addition of coding assigned to each activity, is collated and presented as a performance report (Figure 6.22) which can cover weekly, monthly or project time intervals.

![Valley Longwall Shift Report](image)

**Figure 6.21:** Drilling shift recording activities throughout the shift.
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The performance report has evolved with a more user friendly data input system and processing to include utilisation of drill rig, DHM, drill bit and survey instrument and also provide a record of driller experience.

6.7 MANAGING DIRECTIONAL DRILLING THROUGH UNSTABLE ENVIRONMENTS

6.7.1 Unstable Drilling Environment

Coal seams have been subjected to numerous and various forces which create areas of fractured coal and unstable strata. Even with precautions, caving can occur and bog the
drill string in the borehole. In the course of underground directional drilling, adverse and unstable environments have been encountered and must be identified and managed.

Different forms of unstable ground have been encountered and defined (Hungerford, 1995) with a variety of means developed to negotiate the zones:

- mylonite zones,
- fractured coal,
- high stress areas in coal,
- fault zones,
- cinder around dykes,
- soft clay dykes, sill and bands,
- flaking mudstone at seam roof or floor interfaces, and
- high gas content and pressure.

The instability associated with any of the unstable environments may have an immediate effect or may occur sometime after drilling has progressed through that area. These events may be detected by easier drilling parameters but are identified with increased water pressure and initial stages of bogging. If flushing circulation can be maintained, a blockage can usually be cleared and stabilised. If the drill string does become bogged, it requires an attempt to recover by over-coring or if accessible, eventual recovery by mining. This leads to a loss of drilling performance and equipment, project delays while attempts are made to retrieve the bogged drill string, and/or steel left in the coal seam to be later intersected by mining.

6.7.2 Drilling Practices to Negotiate Adverse Drilling Conditions

Before any drilling is undertaken, it is advisable to assess and define the potential drilling environment. Drilling projects can then be planned to manage the environment and to successfully complete the project while avoiding the adverse effects of intersecting unstable drilling conditions.

Driller experience and exposure of the drillers to the characteristics of unstable environments is the key to managing drilling in unstable environments. An experienced driller can then identify unstable zones and their nature and develop means to manage
any intersections. From that, early detection of potential blockages and bogging can avoid many bogging events.

Underground drilling does not have the luxury of head of water in the borehole that surface drilling does to maintain a constant pressure on the sides of the borehole so all underground drilling is hugely underbalanced as shown in Figure 6.23 (Thomson, 2009).

Steps developed to identify, manage and avoid bogging include:

- Limiting drilling rates in known unstable areas,
- Rotary/slide (slide one rod - rotate one rod) allows the rotation of the rods to break up larger pieces and also agitate cuttings in the borehole for easier removal,
- Clean out an unstable zone if possible with extended flushing and reaming until stable,
- Drill bits have been developed with a back-cutting component to enhance the ability to drill back out of boggy ground,
- If the area cannot be stabilised, design a branch to bypass the zone, and
- Revise the drilling project in line with the limitations created by the environment.
CHAPTER SIX
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When drilling has to negotiate unstable clay zones in the early stages of a borehole, casing can be installed through the zone to allow ongoing secure drilling. This has regularly been required when cross-measure drilling to the Wongawilli seam. Directional drilling produces a borehole through the unstable zone so the borehole is positioned to allow continued drilling (Figure 6.24, Hungerford and Ren, 2014). The borehole is then reamed to a larger diameter before installing casing. The process had to be completed in a 24 hour period before the clay swelled in contact with water.

![Diagram](image.png)

**Figure 6.24:** Cross-measure drilling to and within the Wongawilli seam (Hungerford and Ren, 2014)

This process was improved with the use of a casing advancer and reaming bit which allowed the casing to be used for the reaming with the bit retracted back inside the casing to allow the ongoing drilling. This process removed one of the installation steps and reduced reaming, installation and exposure time through the clay band.

6.7.3 The Use of Drilling Additives

The use of drilling additive is restricted in the underground environment without access to settling dams which allow easy recirculation for surface drilling projects. Application is usually through spot situations when “sticky” conditions are encountered. As mentioned earlier, underground drilling does not have the advantage of operating with a
static head of drilling fluid in the borehole and recirculated drill “mud” to install a “skin” to the wall of the borehole for stability (Thomson, 2009).

The use of additives in the underground environment is usually to increase the density and/or flow rate to enhance removal of cuttings and debris at a rate to avoid blockages. Some additive applications have been employed with a mixing tank underground. The application is single pass with the additives travelling into and out of the borehole then pumped away with the waste water. This becomes an expensive exercise and is only used in moderation.

6.7.4 Over-coring Recovery of Bogged Rods

Unfortunately, even with the best planning and prior knowledge, drilling equipment does get bogged with potential loss at high expense. With increased exposure to adverse drilling conditions and bogging events, over-coring to recover bogged equipment has become an established practice.

Experience, developed procedures and suitable equipment have been the key elements to successful over-coring recoveries. PCD over-core bits to 125 mm OD (Figure 4.46) have enabled over-coring to greater depths and through stone intersections.

A 150 mm diameter standpipe is installed if areas are prone to instability or for new long-hole projects where the stability is unknown. This avoids having to over-core and replace a 100 mm standpipe before over-coring can commence. Gas problems at the site are difficult to overcome if re-standpiping is required in very gassy conditions.

With a bogging event, the borehole depth, magnitude of bends in the borehole and strata penetrated are all assessed to determine the logistics of the operation, and the diameter and make of the shoe bit to provide adequate borehole diameter to reduce in-hole friction. In seams which do not have mining access to eventually recover bogged drill stings, borehole depths are limited to the over-coring capacity of 600 m.

6.8 DRILLING SITE MANAGEMENT

6.8.1 Drill Site Layout

The main components which make up a directional drilling system include an electro-hydraulic drill rig with 1000V, 75/90 kW power which can be either a single track
mounted unit or a three piece modular unit. The hydraulic system must be designed to suit if positioning the power pack some distance from the drill site. The drill rods are usually stored on a trailer or racks positioned behind the drill rig with the water/gas separator, fines tank and water pump located at the entrance to the site. Connection to the mine’s gas extraction system is by a manifold and water trap usually outside the site. All these components must be arranged to provide efficient rod handling with clear walkways about the site. The preferred option is to have clear line-of-sight about the site for the driller although ventilation requirements occasionally have the water/gas separator behind a brattice screen. In this case, the driller relies on good communication with the off-sider to inform him of changes in the colour and flow of the return water.

6.8.2 Drill Site Dimensions

The traditional width of a drill site has been equivalent to the standard roadway width of the mine. The depth of a drill site is usually equivalent to the depth of stub driven opposite a cut-through to house pumping, electrical equipment and support supplies.

Where drill sites are driven into longwall blocks, geotechnical consideration must be given to the lateral and vertical stress likely to develop around the opening. Penetration into the longwall block is critical where high loading has been experienced on longwall faces through stress concentrations across the face and at the intersections during longwall extraction. Rib failure at corners becomes influential especially on thin, acute angled corners of stubs mined at angles other than 90°. In all cases involving longer and angled stubs outside the norm, geotechnical approval is required.

As utilised by most high volume gas drainage drilling operations, the drill rig, control panel and water/gas management system are included within a shallow stub with the power unit and rods located in the cut-through opposite. This requires a modular system that can manage a long run of hydraulic hoses between the power unit and the control panel without suffering hydraulic response delays. Working across the roadway for rod handling must be managed safely with controls in place when mine vehicles are passing the drill site. In instances where no vehicle transport is using the roadway, the driller has the luxury of arranging the equipment in the stub, roadway and cut-through opposite as best suits the operational needs of the project.
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If all equipment is required to be included within the drill site to reduce interaction with mine vehicles, while within the geotechnical limitations, the site must be deep enough to contain all equipment arranged in an efficient operating layout (Figure 6.25). In all cases, some form of management needs to be in place to manage any interaction between drill site and mine transport (Figure 6.26, Tahmoor Colliery, 2004).

![Figure 6.25: Extended angled drill site stub](image1)

![Figure 6.26: Drill Site Setup – Interaction with mine transport (Tahmoor Colliery, 2004)](image2)

When a wide range of standpipe angles is required to achieve a particular drilling pattern, the options of site setup are a wider stub (Figure 6.27, Hungerford, 2008),
CHAPTER SIX
In-seam Directional Drilling Practices

angled roadway (Figure 6.28) or setting the drill rig at the entrance of the stub to utilize the roadway for rod handling. Wider roadways into drill stubs are usually not the norm but with approval after geotechnical assessment, wider stubs are possible in coal and more likely to be permitted if the site is in stone. When drilling is required to flank a roadway, it is difficult to align the rig to start the borehole adjacent and parallel to the roadway with the width limitations of the site. With stand piping and rod handling space behind the rig, a distance of 6.5 m to 7.0 m is required along the line of the standpipe. This can be provided in some cases by excavating a cut-out into the side of the stub to provide more clearance for rig alignment close to parallel with the roadway (Figure 6.28). The stub provided, which may be 19 m deep, can have the side cut-out at the end or at any appropriate location along the side rib but requires geotechnical approval due to the stress likely to be associated with longwall extraction.

Figure 6.27: Wide drill site for fan pattern (Hungerford, 2008)

Figure 6.28: Widened deep angled stub for flanking borehole drilling

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6.8.3 Ventilation of the Drill Site

The initial underground drilling undertaken for gas drainage was rotary drilling without the benefits of directional control. As such, boreholes were not guaranteed to successfully reach their desired depth so pre-installation of standpipes was potentially wasted if a short hole was abandoned and sealed with grout. Drilling proceeded directly into the face with all gas, return water and cuttings flowing from the open hole into the drill site (Figure 6.29, Kelly, 1983). Water and cuttings were managed through dams, pumping and shovelling. Gas from the borehole had to be diluted and vented from the drill site. For active boreholes with large regular gas bursts, the gas, water and cuttings ejected into the site had to be directed away from the driller with the use of a brattice or conveyor belt screen mounted between the borehole and the front of the drill rig. For very active boreholes, brattice screens were shredded and had to be replaced regularly.

Ventilation in the form of brattice screen and venturi was used to dilute and remove this gas from the drill site. Dilution was required for high flows of seam gas (100% concentration) at the collar of the borehole, through the explosive range to the acceptable concentration in the roadway. High ventilation flows were needed in the roadway to reduce the gas concentration below the 0.25% required in intake roadways of a production panel or the drilling would have been undertaken in a non-production panel or return roadway under special arrangements. Gas monitoring was by an automatic methane monitor hung from the roof above the borehole and drillers carried hand-held methane monitors for use as required. Most gas drainage drilling occurred
under close monitoring and inspections by Deputies. Integrated methanometers were not fitted to the electrics of drill rig power units.

As a result of experience from the early drilling systems, improved standards and operations included recognizing ventilation requirements and installing integrated methanometers (Figure 3.16, Section 3) on the electrics of drill rigs to “trip” off power if gas concentrations were over the set limit. The aim now is to control and manage all gas make on a site with increased awareness of the gas concentration limits for intake airways to in-by mining faces. The system of managing gas, water and cuttings from boreholes has evolved in different forms at the various mines which undertake a high volume of gas drainage drilling. With the enhanced awareness of gas management, a gas “trip” is now a “reportable incident”.

With the increase in the depth of drill sites to be able to contain all drill equipment, ventilation of the drill site has become more complicated. Deputies are usually assigned the task of ensuring the site is adequately ventilated or the designers of the site layout can supply a suggested ventilation layout. The ventilation layout can now be verified for a typical drill site layout (Figure 6.25) by computer modelling the ventilation effects and dilution of gas flows (Figure 6.30 and Figure 6.31, Ren, et al, 2015). This case showed ventilation was effective with a brattice screen only with the vent tube having little additional effect. Alternative ventilation by brattice screen and/or vent tubes can now be provided for any deep site layout.

![Figure 6.30: Velocity vectors distribution at 1.5 m above floor (Ren, et al, 2015)](image)

<table>
<thead>
<tr>
<th>Brattice screen only</th>
<th>Brattice screen and Vent tube</th>
</tr>
</thead>
<tbody>
<tr>
<td><img src="image" alt="Velocity vectors distribution at 1.5 m above floor" /></td>
<td><img src="image" alt="Velocity vectors distribution at 1.5 m above floor" /></td>
</tr>
</tbody>
</table>

Figure 6.30: Velocity vectors distribution at 1.5 m above floor (Ren, et al, 2015)
6.8.4 Standpiping

Standpiping is used to control and manage all returns (gas, water and cuttings) from the borehole during drilling and manage gas and/or water drainage on completion of drilling. The system should be designed to manage the worst case possibility of intersecting high pressure gas or water during the drilling operation. For this, the expected pressure is assumed as the static head of water from the surface.

The standard standpipe diameter used is 100 mm for drilling a 96 mm directional borehole. A 145-150 mm borehole is drilled into the face into which the standpipe is grouted (Figure 6.32, Hungerford, et al, 2013b) with fast curing, non-shrinking grout. Installing standpipes horizontally identified a problem with sealing along the top of the borehole due to either an air cavity, settlement or thinning of the grout. A practice was introduced to ensure the standpipe was installed with a conformed angle downwards or upwards similar to the dip of the seam and on the proposed line of drilling. Grout is then pumped to the lower end of the standpipe to fill the void from the lower to the higher points ensuring full displacement of air or water.
When adverse conditions are expected and over-coring a bogged drill string is likely, a 150 mm standpipe is installed. This diameter standpipe is also used if reaming is required for enhanced gas or water drainage, pumping, casing or installing services. The standard standpipe is 6 m long copper tube with some mines now using 4.5 m long composite standpipes. Where high pressure environments are likely to be encountered, longer standpipes of stronger material are installed. Heavy walled standpipes are used in high stress areas to prevent crushing or where standpipes may be worn through with rotary reaming.

The preferred environment is to have well anchored and sealed standpipes at 90° to the cleat direction (Figure 6.33, Hungerford, et al, 2013b) which limits gas flow through the coal. Angling the standpipe into the corner of a drill site provides maximum distance for a rib (Figure 6.34) but cleat direction can still be influential.

Figure 6.33: Low gas migration across cleat to rib with high shut-in pressure (Hungerford, et al, 2013b)
The characteristics of the coal, the state of the rib and the planned alignment of the borehole are assessed before installing the standpipe. High permeability, parallel cleating (Figure 6.35 and Figure 6.36, Hungerford, et al, 2013b), fractured rib and reduced distance of angled standpipes in from the rib (Figure 6.36, Hungerford, et al, 2013b) present problems with the effective seal of the standpipe. In each case, high suction pressures can draw air in through the rib and into the standpipe or with high shut-in gas pressures gas can migrate to and out of the rib into the drill site. To reduce and control gas emissions into a drill site, common practice is to seal the rib with pressure grouting (Figure 6.37, Hungerford, et al, 2013b) or polyurethane injection.
Figure 6.36: Low angled entry reduces effective sealed depth of standpipe (Hungerford, et al, 2013b)

Figure 6.37: Pressure grouting to seal the rib (Hungerford, et al, 2013b)

6.8.5 Management of Cuttings and Waste Water on Site

Effective management of the gas/water/cuttings system on the site reduces the instances of gas issues on the site. This includes regulating suction (if available) to remove gas or directing excess gas into the return airway under established controls which would include a diffuser at the exhaust end. Installing a U-tube to the fines bin, managing the water pump, removing waste water and emptying the water trap (either manually or via
automatic dewatering) are good practices which ensure effective management of gas make on site.

The first design of standpipe fittings originated from US coal drilling operations included a low pressure, in-line water/gas separator (Figure 6.38) with a rod wiper at the back so that gas was directed out of the top and water and cuttings dropped out below. With the need to be capable of managing high pressures, a gland stuffing box was introduced with the standpipe fitted with high pressure “T” piece and valves (Figure 6.39) to manage high pressures.

Hosing from the standpipe fittings carry the gas, water and cutting mix from the standpipe to a water gas separator (Figure 6.40, Touzell, 2014). Gas is removed by vacuum to the gas drainage range and the water/cuttings mix is dropped into a settling dam or bin. The cuttings are settled out of suspension and the waste water pumped into the mine’s waste water system. Cuttings are removed by either shovelling or a scroll mechanism. The additional valve on the side of the standpipe has been installed for suction to control gas flow when removing the DHM from an open standpipe.
This system is modified to suit various applications with or without access to vacuum drainage. Some drilling departments drilling in familiar conditions have reverted back to the in-line separator system for ease of water handling but rely on ventilation to manage gas on site when high gas flows are experienced.
6.9 SUMMARY

The operating parameters and steering characteristics are revised as new models of DHM or bend configurations are introduced. Borehole design parameters are modified to utilise the changing steering characteristics.

Progressive seam definition has become a standard practice with all in-seam directional drilling with standards established for regular seam or roof intersections and with well positioned branching used to continue drilling. The branching practices are well established as are the concerns of stability and flushing efficiency in the area of branches.

Data recording has been essential throughout to define the drilling parameters now in use. The ability to steer a DHM is basic to directional drilling and would not be possible without drilling and survey records. Management can monitor and assess the performance of drilling projects and the equipment through the shift reports provided by drillers. Project reporting to management and clients has become easier and more complete with the current performance reporting.

Site management has evolved from positioning all drilling components at an intersection and starting drilling. Efficient and safe positioning of all drill rig components is essential with due regard to people movements, moving machinery and managing all material flowing from the borehole. Standpiping has been established as the standard means of control with consideration needed to manage the conditions of the state of the rib and the expected in-hole pressures. A high pressure stuffing box is preferred to manage high pressure environments although some drilling operations have to rely on ventilation to manage gas releases into the drill site. With mines preferring all equipment off the travel roadways to limit interaction between drill site and transport, longer drill sites have been provided. With that, site ventilation becomes more critical with modelling able to provide confirmation ventilation systems are adequate.

Unstable environments have always caused problems for drilling operations. The nature of unstable environments encountered with underground drilling has been defined through experience and problems associated with intersections. Prior knowledge, planning and established drilling characteristics when intersected allows unstable zones to be defined, negotiated and/or avoided with limited loss of equipment or drilling
production. Even with experience with boggy zones, drill strings are still being bogged. Recovery procedures are improving with each recovery attempt and offer improved level of security for directional drilling practice in difficult environment.

Any new developments in directional drilling technology have been introduced to the coal industry through papers, presentation or direction consultation.
CHAPTER SEVEN
APPLICATION OF IN-SEAM DRILLING FOR GAS DRAINAGE IN COAL MINES

7.1 INTRODUCTION

Although the application of in-seam drilling has required the progressive definition of the seam profile to position boreholes within the seam for gas drainage, it was not until the late 1980’s that the technology was utilised for in-seam exploration. This chapter explains how direction drilling is utilised as a means of in-seam exploration. Drilling to identifying and defining the various geological structures or features which may be present differs from each case. The methods used and the planning required are explained with examples of how the drilling has been used to define a number of the geological features.

Directional drilling has also been used to provide water drainage of coal seams, drain water filled old workings and provide a means of water management in underground mine. The inrush event at Gretley colliery has led to the each mine having a responsibility to identify any possible inrush risk, to define that risk and to put plans in place to either avoid or mitigate that risk. The various applications of directional drilling to provide water management are described in this chapter with specific examples of each application. An example involving the use of several exploration and water management applications used in concert at the same mine is described.

7.2 IN-SEAM DIRECTIONAL DRILLING FOR GAS DRAINAGE

Although directional drilling was introduced for gas drainage to control outbursts, slower drilling rates, the requirement for higher driller skills and unreliable survey technology, caused many mines with gas drainage drilling programs to persist with rotary drilling as their preferred gas drainage method. Ongoing development of directional drilling continued through both gas drainage and exploration drilling projects. As the technology was developed, mines with drilling programs took on the training of their drillers to have the capability to provide gas drainage for specific areas adjacent to roadways and around structures.

The training of the drilling crews at mines explained in Chapter 6 was for gas drainage applications and using Eastman single-shot surveying. This gave the drillers an
understanding of directional control and of hand plotting both vertical and lateral aspects of each borehole as they were drilled.

The fatal outburst at Westcliff Colliery in 1994 (Walsh, 1997) was primarily due to rotary drilled drainage holes deviating off-line without surveying being conducted to verify their location (Tonegato, 1998). This led to the Inspectorate requiring OMPs to be developed at each mine. It became imperative to use a drilling system that was both accurate and steerable, thus being able to position boreholes consistently. Directional drilling was the only form of drilling which could provide accurate placement of boreholes with surveying to verify their location.

The key components of the OMP were:

- Drilling boreholes on a proposed design layout to provide gas drainage.
- Monitor gas flows from the boreholes to assess gas drainage effectiveness.
- Provide adequate drainage time to allow gas levels to be reduced to safe levels.
- Accurate positioning of boreholes and verification of their position through surveying.
- Coring for worst case environment (mid-point between gas drainage boreholes) to assess and determine gas content is at safe levels before issuing an Approval to Mine (ATM).

7.2.1 Borehole Spacing

Design of the appropriate borehole patterns and drilling the boreholes in the correct location are the domain of the drilling department. The eventual spacing of the boreholes is a function of the permeability of the coal and the time available for drainage determined by gas drainage personnel. Earlier borehole spacing was based on expected gas flows from reservoir analysis of gas content, gas pressure and seam permeability. From that point, borehole spacing was along the basis of trial and error with a particular borehole spacing being used, gas flow measurements recorded over the drainage life of the boreholes and gas content coring undertaken to assess the gas drainage effectiveness before mining. The spacing was then adjusted to suit the available drainage time for the particular drainage characteristics at that mine.
For longer boreholes, the resistance to gas flow creates a pressure in the boreholes which retards desorption of gas from the coal (Hungerford, et al, 1988a). Drainage from the end of a long borehole is going to take longer than from a shorter borehole. Borehole spacing then becomes a balance between the drainage effectiveness for the length of borehole and the length of time available to provide drainage before mining enters the area.

With borehole spacing determined, the design of borehole patterns becomes a process to provide coverage of the drainage area within the limitations of borehole curve rates and distance.

### 7.2.2 In-seam Borehole Patterns

The current standard of gas drainage drilling is of fan patterns across each longwall block from sites to provide drainage of the next gate-road development as well as the longwall block.

Each mine seemed to arrange their preferred drilling pattern along the lines of previous rotary drilling patterns. This was also defined by the level of driller skills developed by the mine training programs and the level of expertise developed by supervising management. The easiest and simplest drilling is of straight borehole fan patterns (Figure 7.1) with a Target Azimuth to suit each borehole and the driller required to drill a straight borehole along that alignment. Although borehole patterns can be duplicated, they are usually altered to suit the required coverage from each site. Borehole spacing varies throughout but the borehole layout provides coverage with no borehole spacing exceeding a designated maximum.
Although each mine has its characteristics of borehole patterns, mines progressively revised and modified their design to provide better drainage coverage as driller skills and directional control improved. Initial single-shot surveying limited the drift (angle from the Target Azimuth, Figure 5.10) and the lateral deviation that could be hand plotted. This limited curvature in the designed borehole patterns. The technology advances and improved performance as well as availability of electronic survey instruments provided on-site calculation of vertical and lateral displacement allowing more elaborate lateral designs with greater lateral displacement. Directional drilling progressively defined the rates at which boreholes could be deviated to follow design curves while still providing vertical control of boreholes within the coal seam. The curve rates were found to be reduced when affected by cleat direction, in areas of instability or when drilling in a particular direction. These factors defined the rates of curve used when designing borehole patterns to ensure the drillers were able to follow the design layout.

The progressive improvement in drilling techniques allowed branching and curved fan patterns to be utilised. As drilling and steering skills improved, drilling patterns became more refined. The candelabra pattern of boreholes, as shown in Figure 7.2, was developed and this allowed coverage by evenly spaced, parallel boreholes through the drainage area of an adjacent gate-road panel. As the drilling crews establish the ability to drill a particular pattern, this pattern was then duplicated to be repeated along each longwall block to adequately cover the next gate-road development. The completed drilling (in blue) also shows notations in green of soft coal from drilling records. A
bogged drill string is also shown so the contact with the steel components can be managed by the longwall operations on extraction. Additional boreholes are required to provide drainage of the longwall block in the area between in the candelabra patterns. The shorter curved boreholes shown adjacent to the panel are rotary boreholes for gas content determination to satisfy OMP requirements. The planned drilling to cover the next panel is shown in red. With the drainage time available before development of the next gate-road intersects the boreholes, the borehole spacing is usually 18 m. This spacing has been reduced to 15 m for shorted longwall blocks with shorter drainage times available.

A seam profile for each borehole is required to maintain the borehole within the seam. But for production drilling for gas drainage, the preference is to reduce the roof intersections and subsequent branching required. This is to increase drilling production which also reduces the number of branch points regarded as potential blockage points. Branching is also to be avoided if conduit needs to be installed. Various practices are utilised which involve using one borehole to define the seam profile then subsequent boreholes can use that profile with slight adjustments for the grade of the seam across the drill pattern. The centre borehole can be drilled first then adjusted for the profiles

Figure 7.2: Candelabra patterns of boreholes across longwall blocks
CHAPTER SEVEN
Application of In-seam Drilling for Gas Drainage in Coal Mines

each side or the boreholes at the extremities can be drilled first with an average grade used between the two profiles.

As adverse drilling and/or gas drainage environments are intersected and defined, the standard patterns can be modified to suit the specific requirements, accessing areas which haven’t been covered previously. Borehole patterns at 90° to the longwall block are the norm but the alignment can be altered to take advantage of better drainage when drilling relative to cleat direction. The alignment of borehole patterns at Grasstree colliery has been altered by 13.5° to take advantage of improved drainage characteristics (Figure 7.3, McInerney and Brown, 2016).

![Diagram](image)

Figure 7.3: Angled candelabra patterns of boreholes across longwall blocks (McInerney and Brown, 2016)

Each OMP includes a system of coring for gas content testing to assess the gas content before mining can proceed. An ATM is issued for the next section of mining. With the mine plans, this includes borehole locations, drilling anomalies and gas content and compositions results.

7.2.3 Long-hole In-seam Borehole Patterns

Where access from the adjacent gate-road is not available, modified designs allow drainage access from more remote drilling locations. When longer drainage times are available, long-hole drilling technology has been utilised to provide long boreholes
along the longwall block and adjacent gate-road (Figure 7.4, Hungerford, et al, 2013a). These require more detailed design and interaction/support with the drillers to explain the specifics of the design and drilling requirements and to manage the lateral control required with the large lateral deviations involved (Figure 7.5). Because of the long drainage time available, the borehole spacing was extended to 55 m. Due to borehole length restricting gas flow, the preferred option is to only have one long borehole from each standpipe. The seam profile (Figure 7.6, Hungerford, et al, 2013c) from each borehole provides the mine with exploration information for mine planning well before development commences. In the case in Figure 7.6, clear ground was proved until an anomaly was identified at 957 m. The mine had ample time to investigate how to negotiate the structure before development reached the area.

Figure 7.4: Long longitudinal boreholes for drainage of future longwall development (Hungerford, et al, 2013c)
CHAPTER SEVEN
Application of In-seam Drilling for Gas Drainage in Coal Mines

Major development panels are usually angled away from previously mined roadways which limit access to drill boreholes for gas drainage. Longitudinal borehole patterns have been designed to provide long term drainage ahead of development with suitable borehole spacing and drainage “lead time”. When drainage is effective with longer
boreholes, a longer length of uninterrupted development is available when mining commences. When mining reaches the extremity of the drilling, mining must stop until more drilling and drainage has been completed. The borehole pattern shown in Figure 7.7 involved 1400 m long boreholes from a remote site to run parallel to the proposed development with single boreholes running down the middle of each line of pillars and two boreholes flanking each side of the panel. Once a profile has been established with regular roof intersections, that profile can be used for adjacent boreholes with much wider spaced roof intersections required to confirm the seam profile. Close attention is required to lateral control so boreholes don’t run down a roadway and present problems with subsequent mining dealing with any residue gas flows from the borehole. The borehole spacing in this case was 40 m.

![Figure 7.7: Longitudinal boreholes for drainage of roadway development](image)

One consequence of drilling long boreholes is that mining has to deal with the prospect of crossing each borehole at some stage. Close attention to monitoring gas flows over the drainage life of each borehole may detect the possibility of a blockage but in each case, the mine must approach an intersection with the prospect that that hole may be pressurized due to a blockage. The mining sequence has to be altered to allow probe drilling to intersect each borehole ahead of mining and intersected boreholes can be hosed over each roadway intersection to continue drainage.
Profile definition is provided by the initial borehole along the panel with subsequent adjacent boreholes requiring fewer roof intersections to remain within the seam.

When access is restricted but the drainage time is limited, rather than have long parallel boreholes along the panel, a borehole pattern has been designed to provide multiple boreholes crossing the proposed development (Figure 7.8, Hungerford, et al, 2013c). Multiple boreholes (Figure 7.8) at even spacing provided a more concentrated coverage for faster drainage of the panel. This pattern also provided drainage of the longwall block to the extremity of the boreholes within the block.

![Figure 7.8: Combination pattern from remote access for short term drainage (Hungerford, et al, 2013c)](image)

With longer boreholes used for gas drainage, the standpiping and fittings need to be able to manage the removal of the DHM from the borehole with high gas flows present. Practices have been developed which allow this without exposing the borehole to the atmosphere at the drill site. If very high gas flows or pressure are anticipated, an extension is recommended which allows the DHM to be withdrawn without exposure at the site.

### 7.2.4 Gas Drainage Drilling of a Thick Seam

Gas drainage in a thick seam has to consider the drainage characteristics across the full section of the seam. With low permeability, impermeable bands within the seam or low flows of CO$_2$, drilling has to provide coverage of the whole seam section. In the Dartbrook mine for example, boreholes were directed up through the seam to the roof
before branching in the lower section of the seam to continue drilling (Figure 7.9). This is particularly suited if the upper seam strata are unstable so drilling is not exposed in the upper section for long periods. The presence of distinctive stone bands makes seam definition much easier.

![Figure 7.9: Thick seam drainage coverage with multiple branches up to the seam roof](image)

In the current Narrabri mine drilling practices (Baxter, 2015), boreholes along every alignment of a borehole pattern are duplicated with one borehole in the upper section of the seam and another drilled directly below (Figure 7.10). This latter case is suitable if drilling conditions are consistently good throughout the seam profile. In this case, the most efficient sequence would be to drill the upper borehole first with roof intersections for seam profile definition followed by the lower borehole not needing any further profile definition.
7.2.5 Installing Conduit into Boreholes

In low flow, down-hill boreholes, the lower end of boreholes may not be effectively drained due to build-up of water in the borehole. A practice has been established to install conduit (40 mm diameter PVC or polyurethane tube) to the end of the borehole and seal the annulus of the borehole. Gas pressure will eject water out through the conduit in the initial stages of drainage and dewater the seam in the vicinity of the borehole. When water flow has diminished, the annulus is opened to allow free flow. Some operations allow the installation of conduit by hand while others require hollow rods to be installed to clean the borehole before installing the conduit through the rods. A disposable non-return valve is used on the rods and released by an anchor pushed forward on the conduit. The anchor then secures the end of the conduit at the end of the borehole so the rods can be withdrawn, leaving the conduit in place. The conduit usually has 30 m of the bottom end perforated to provide multiple points of access for water flow.

The same system is used when longer boreholes are drilled through unstable areas. Perforated conduit is installed to ensure a flow passage through unstable areas to avoid blockages from caving. The whole length of the conduit is perforated in this application.
When conduit is to be installed, the borehole patterns are altered to ensure single boreholes only in the pattern (Figure 7.11) and not multiple boreholes from each standpipe (Figure 7.2 and Figure 7.8). The middle borehole (803MG-4A-05) would be drilled first with regular roof intersections as shown for seam profile definition. The profiles for the adjacent boreholes would have been adjusted from the centre profile based on contour or roadway grades to reduce the need for branching.

![Figure 7.11: Borehole fan pattern of single boreholes for installing conduit](image)

### 7.3 CROSS-MEASURE DRILLING FOR GAS DRAINAGE

The purpose of cross-measure drilling is to access seams in close proximity to the mining seam from drill sites located in the mining seam. A drill site does not need to be located in the seam in which drilling is planned. This method is commonly practiced in mining multiple seams with close inter-burdens or where direct access to the target seam is not available.

#### 7.3.1 Cross-measure Drilling into the Balgownie Seam

The initial drilling into the Balgownie seam was planned as a post-drainage exercise with gas removed as the longwall passed over the borehole (Hungerford, *et al.*, 1988a).
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This was successful in this case but had mixed success in subsequent attempts. Most problems were due to the inability to extend boreholes to any length in the thin seam.

Subsequent drilling projects have aimed to provide both pre-drainage of the Balgownie seam then have boreholes in position to provide post-drainage as the longwall face passes over the boreholes (Figure 7.12, Hungerford, et al, 2013a). A project was attempted at Tahmoor to position boreholes within the Balgownie seam adjacent to the main-gate roadway in the expected stress relief fracture zone under the extracted longwall block. This project had limited success due to difficulties in trying to drill in the thin Balgownie in an area which was undulating. Further trials are required to improve drilling within the Balgownie seam and to identify the most effective positioning of the borehole for post drainage.

Figure 7.12: Cross-measure drilling pattern into the Balgownie seam (Hungerford, et al, 2013a)

7.3.2 Cross-measure Drilling into the Wongawilli Seam

Drilling has been conducted down into and within the Wongawilli seam from drill sites in the Bulli seam. In each case, drilling was to provide drainage of the seam prior to either development mining or installation of a drift and coal storage bin between the seams. As identified earlier in the 1980’s research project (Hungerford, et al, 1988a), the soft tuff band in the upper section of the Wongawilli seam presented a stability problem both during and after drilling. The initial drilling involved drilling a
directionally controlled down through the tuff band in the Wongawilli seam at a trajectory which could continue drilling within the seam (Figure 7.13, Hungerford, et al, 2013a). This borehole was then reamed to 145 mm diameter and a 100 mm standpipe installed and grouted before drilling continued. The tuff remained competent for only 24 hours after drilling so the drilling, reaming and standpipe installation was restricted to a 24 hour period. This process was improved with the reaming being done using HWT casing fitted with a reaming shell and retractable casing advancer (Figure 7.14, Atlas Copco, 2009). When the casing had been reamed through the tuff band, the “over-shot” was fed inside the casing on the end of the drill rods to release the internal casing advancer bit. After retrieving the casing advancer bit, the casing was grouted into position before directional drilling recommenced. This removed a step from the process, and allowed more consistent standpipe installation before the clay in the tuff band swelled.

Figure 7.13: Profile of cross-measure drilling into the Wongawilli seam
7.3.3 Cross-measure Boreholes above Seam for Goaf Gas Capture

A trial has been conducted at Ravensworth underground mine by positioning boreholes along and above the longwall block to develop a system of reliable goaf gas capture (Brown and Hobden, 2013). A system had been in place in China of driving a sewer roadway in the stone above the longwall block with this roadway located within the expected fracture zone and generally closer to the tail-gate side (Figure 7.15, Cheng, 2010). The trial was to replace the sewer roadway with several appropriately placed boreholes.

Figure 7.15: Sewer roadway above longwall (Cheng, 2010)
Boreholes were drilled initially to 800 m and 1000 m depths on each side of the longwall block and various heights above the seam and distances in from the rib-line of the gate-roads. Several were reamed out to the larger diameter of 145 mm to depths of 500-600 m. Due to the drill site width and alignment, the three boreholes drilled at each site along the longwall block had to be started 30.5 degrees off line and curved around to the proposed alignment (Figure 7.16, Hungerford, et al, 2013a). This became a standard design with the final alignment of the three boreholes 25, 30 and 35 m in from the rib-line.

Lateral control was required to provide a tight curvature in stone to get the borehole on-line as some as possible. Early boreholes in the trial were directed as high as 30 m above the coal seam. Boreholes drilled more than 20 m above the seam intersected very hard sandstone which resulted in very slow penetration rates. The final height of 15 m above the seam (Figure 7.17) was in sandstone in which reasonable penetration rates were possible. The preferred profile then followed the profile of the seam as defined by Reduced Levels (RLs) along the adjacent roadway. Although a steel stator was preferred on the DHM for better wear characteristics, a BeCu stator was required so surveying was possible directly behind the DHM. The gradients required were very close to horizontal and very good vertical control was required to avoid hollows which may become filled with water and restrict gas flow.
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The gas capture design required that all boreholes be reamed to 145 mm diameter to enhance gas flow. With the comfortable reaming depth limited to 600 m, the design length of all boreholes was limited to less than 600 m with an overlap of 80-100 m between sets of boreholes. Reaming in sandstone identified weaknesses in the reamer design and suitability of drill rods. The design of the reamer evolved to the very robust and effective model shown in Figure 4.52 used in conjunction with NWJ rods supplied with an OD of 69.9 mm to be compatible with the rod clamp and rotation unit jaws in the drill rig and the diameter of the stuffing box seal.

![Figure 7.17: Profile of cross-measure boreholes for goaf gas capture](image)

7.4 The Introduction of Directional Drilling to the Chinese Coal Industry

Initial attempts to introduce directional drilling technology to the Chinese coal industry through a UN project in 1997 were unsuccessful. These attempts were poorly managed and introduced into unstable and adverse environments in which directional drilling was never likely to be successful. When invited back in 2002, a systems approach was adopted to promoting directional drilling in China (Hungerford, et al, 2005). This included: (1) audit sites prior to committing to supply of drilling systems to ensure directional drilling objectives were achievable; (2) design and manufacture drill rigs and ancillary equipment based on the mine requirements and logistics; (3) supply the drill
rig, guidance system and all ancillary equipment as a matched package; and (4) offer training and ongoing technical support to ensure the success of these operations.

This system was first employed with the introduction of directional drilling at Daning mine in Shanxi province (Hungerford, et al, 2005). At total of 78484 m of in-seam directional drilling was completed in first full year of drilling (Figure 7.18, Hungerford, et al, 2005). The mine eventually acquired four directional drilling systems and is now drilling 20000 m per month. The average monthly methane production volume is 5400000 m$^3$ with the recovery rate increasing from 28% methane to more than 68%.

![Figure 7.18: Daning mine gas drainage drilling (Hungerford, et al, 2005)](image)

The Series 1000 drill rig mentioned earlier (Figure 3.14) was designed and manufactured specifically for this first project and has since been improved and upgraded through a rigorous operational and performance review process. More than 80 directional drilling systems have now been introduced to and are operating in the Chinese coal industry.
7.4.1 Directional Drilling and Gas Drainage – Baijigou Mine

Common practice in Chinese mines was to drive a stone roadway under future development roadways and undertake gas drainage through a fan of rotary drilled boreholes up into the seam (Figure 7.19, Hungerford, 2007b). Coverage in the seam was limited and a high percentage of stone was drilled compared to coal drilling. A demonstration project was undertaken in 2007 at Baijogou mine in the Ningxia region with both underground directional drilling and SIS drilling proposed to drain gas from a proposed longwall block (Figure 7.20, Hungerford, 2007a).

![Diagram](image_url)

**Figure 7.19:** Rotary cross-measure drilling for gas drainage – Baijigou mine (Hungerford, 2007b)
The project involved drilling cross-measure in-seam boreholes from a stone drive located 50-60 m below the seam (Figure 7.21, Hungerford, 2007a). A seam thickness of 20 m and reasonable stable drilling environment allowed good drilling performance in coal after the very hard stone strata (>100 MPa uniaxial compressive strength) had been penetrated. The standard PCD bits were unable to penetrate the stone strata so a tungsten carbide tipped roller cone bit was used through the stone then replaced with a PCD bit for drilling in coal. The stone drilling sections are shown in blue in Figure 7.20. Branching in the coal was utilised to provide two boreholes for each standpipe and stone drilling section (Figure 7.22, Hungerford, 2007b).

The project initially also involved SIS drilling to cover the northern end of the longwall block but when that proved unsuccessful, boreholes drilled from underground were used to cover that area (green boreholes, Figure 7.20, Hungerford, 2007a).
A modular rig was designed and manufactured specifically for the project allowing the vertical angles required (+15 degrees) for the initial cross-measure drilling of each borehole. With suitable site dimensions, a wider fan of boreholes was possible (Figure 6.27).
The project involved 41466 m drilling in the original design. The longest borehole was 1023 m in length with an average borehole length of 712 m. The drilling design offered 80% of borehole effectiveness (metres in coal versus overall borehole metres including stone) as compared to 40% usually achieved with the rotary drilling program (Figure 7.19, Hungerford, 2007b). A total of 490 million cubic metres of methane seam gas was drained at 85% purity. This was achieved over an eleven months drainage time compared to the usual four years required with the rotary program.

7.4.2 In-seam and Cross-measure Drilling for Gas Drainage - Wuhushan Mine

The preferred practice with in-seam drilling in China for gas drainage is to drill longitudinal boreholes from the end of longwall blocks (Figure 7.23, Hungerford, 2012). Access is from main panels and does not have to manage interaction between mining and drilling operations if drilling from gate-road development panels. Where conditions allow, boreholes are drilled in-seam to 1000 m. In the Wuhushan project, drilling was to 1000 m in the No. 9 seam (blue boreholes, Figure 7.23).

Boreholes were also drilled into the underlying No. 10 seam (pink boreholes, Figure 7.23) by access either directly from the drill site of with branches from the No. 9 seam boreholes (Figure 7.24). The logistics of accessing the 10 Seam included drilling cross-measure through a hard inter-burden to attempt to intersect the 10 Seam at a low angle. This was required to maintain the borehole within the upper section of the 10 Seam to avoid adverse drilling environment which exists in the fractured lower section that seam. This was complicated by the inter-burden varying in thickness preventing easy planning for vertical trajectories, the hard inter-burden requiring steep intersection angles to avoid deflection back into the 9 Seam and trying to also maintain directional control in the horizontal plane to match proposed drilling layouts.
Although drilling was achieved in both seams, varying stability problems presented challenges with borehole collapse and bogging of the drill string a common occurrence. A specific set of guide-lines were established to manage borehole stability problems. If
any abnormal conditions were experienced the drill string was to be pulled back to a
safe position and the supervisor contacted for instruction. Such conditions included:

- No water or reduced water return.
- Large amount of fines, or large fines particles.
- Rods becoming sticky or boggy.
- Higher than normal drilling pressure.
- Difficulty in steering or controlling the drill string.

For drilling cross-measure in the No. 10 seam where bogged equipment would not be
able to be retrieved by future mining, borehole depths were limited. Over-core recovery
was expected to be limited to 600 m (Section 6.7.4), so a process was established where
the stability of the borehole was assessed to 400 m and then to 500 m before approval
was given to progress with the drilling for the next 100 m. In stable conditions, the
boreholes were extended to a maximum of 600 m.

Even with precautions mentioned previously to manage drilling in adverse conditions,
some fractured coal seams are very difficult to drill in. At best, drilling can only
continue for short periods before the borehole starts to collapse. The option is to drill in
stone either below or above the seam and intersect the seam for short intervals until
stability becomes a problem. Branching in stone can then progress the borehole further
for another coal intersection. The designed borehole in Figure 7.25 is shown in red with
the anticipated seam profile in brown. The actual drilling had to contend with the seam
profile being appreciably different to that anticipated. Drilling rates suffered due to slow
penetration in stone when compared to that in coal. Also branching in stone is a time
consuming process compared to that of coal. Although drilling rates are slower with
reduced exposure to the coal seam, the gas content in the seam was effectively reduced
before mining progressed.
7.5 SUMMARY

Significant advances in directional drilling systems during the last two decades has led to the development of a technology that can currently provide the coal mining industry with a large range of drilling options for gas control. The equipment and practices employed in directional drilling have been developed to improve drilling rates, control, accuracy and the overall drilling depths achieved. These advances have effectively reduced directional drilling costs and increased the opportunities for use of this technique in the coal mining industry, including gas drainage to ensure safe working conditions. Gas drainage drilling is now practiced successfully as either high repetition cross-block drilling to the mine’s preferred borehole pattern or longer, specifically placed boreholes to provide long-term drainage of development panels well before mining commences.

The ability to drill cross-measure to seams or horizons above or below the current working seam has allowed boreholes to be placed in seams to provide gas drainage which would otherwise not have been possible. Each project has a unique environment which has to be identified before drilling projects can be designed to suit the project requirements and manage the challenges of the environment. This has been done with
successful drilling from the Bulli seam into the Balgownie and Wongawilli seams in the south coast of NSW.

More work is possible with cross-measure in-seam drilling for both pre-drainage and post drainage. The main areas would be in drilling skills and support to successfully drill in thin undulating seams and experiment on the suitable positioning of boreholes under longwall blocks to best access gas released due to stress relief and fracturing.

Ongoing trials are required to utilise boreholes above longwall blocks to their best effect to reduce gas concentrations on the longwall face and surrounds. Most effective positioning of boreholes for each is likely to be through a system of trial and error.

With in-seam directional drilling technology established as a standard practice in Australia, the challenge has been to introduce the technology to overseas environments. This technology has been successfully introduced to the Chinese coal industry. Although early successes have been achieved, each project introduces unique challenges with underground operational and drilling environments.
8.1 INTRODUCTION

Although the application of in-seam drilling has required the progressive definition of the seam profile to position boreholes within the seam for gas drainage, it was not until the late 1980’s that the technology was utilised for in-seam exploration. This chapter explains how direction drilling is utilised as a means of in-seam exploration. Drilling to identifying and defining the various geological structures or features which may be present differs from each case. The methods used and the planning required are explained with examples of how the drilling has been used to define a number of the geological features.

Directional drilling has also been used to provide water drainage of coal seams, drain water filled old workings and provide a means of water management in underground mine. The inrush event at Gretley colliery has led to the each mine having a responsibility to identify any possible inrush risk, to define that risk and to put plans in place to either avoid or mitigate that risk. The various applications of directional drilling to provide water management are described in this chapter with specific examples of each application. An example involving the use of several exploration and water management applications used in concert at the same mine is described.

8.2 IN-SEAM DRILLING FOR EXPLORATION

In-seam rotary drilling with 80 mm boreholes and BQ drill rods was established for exploration through a project at John Darling colliery (Hungerford, 1986; Williams and Hungerford, 1988). The requirement was to locate and identify a geological structure for mine planning. The drilling was a success with the drill bit preferentially following the seam to a fault/dyke intersection. BHP Macquarie Collieries subsequently purchased the drilling equipment and routinely engaged in long-hole drilling for locating a variety of geological structures.

In 1987, a long-hole drilling trial was started at German Creek mine, drilling in-seam boreholes from the open-cut high-wall at German Creek mine (Hungerford, 1988b,
Williams and Hungerford, 1988). The drill rig was bolted to the open-cut pit floor and a shed provided for protection to be able to operate in close proximity to the high wall. The mine persisted with the technique to define structures around the proposed new Southern colliery. A maximum depth of 664 m was achieved.

### 8.2.1 In-seam Directional Drilling for Exploration

With the success but known limitations of rotary drilling for exploration, in-seam directional drilling was identified as an effective exploration tool to define the seam profile, investigate structures ahead of mining or prove ground free of structures. Long-hole in-seam drilling was seen as the only method of exploration which directly tests the area prior to mining, hence providing a potentially high level of confidence and forewarning (Williams and Hungerford, 1988). The key aspects of in-seam exploration is defining the seam profile and identifying any geological anomaly that is defined by a displacement of that profile or identified by changes in the drilling parameters and the colour of the flushing water.

Directional drilling was initially used for exploration with an in-seam exploration borehole drilled at Ellalong colliery flanking proposed main development roadways to investigate a possible fault in the area (Hungerford, 1988b). The drilling proved seam profile consistency to the 589m depth of the borehole (Figure 8.1). With the technology and practice proven, subsequent boreholes were drilled at Ellalong colliery in 1991 to prove clear ground ahead of development (Thomson, 1991) and define faulting in a longwall block (Thomson and Hungerford, 1991).

An in-seam exploration drilling project was established in 1989 at Huntly West underground mine in NZ to successfully define the profile and thickness of the seam and identify fault displacements ahead of development (Figure 8.2) to allow mine profile planning within an undulating thick seam (Beamish, et al, 1991). An added benefit of gas drainage from the drilling was utilised until the shutting of the mine after a spontaneous combustion event and subsequent explosion.
The use of in-seam drilling for exploration became an established practice at West Cliff and North Cliff collieries with both BQ and NQ sized drilling. Surface seismic had been used as the primary method of detecting major structures (>5 m displacement) over large areas of the reserve. A borehole was drilled to a depth of 922 m adjacent to
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proposed development roadways at West Cliff colliery (Walsh, 1991). Significant faulting (>1 m) was detected at 354 m. A similar borehole was drilled to the west of North Cliff colliery to prove production ahead of 210 Panel. A fault of 4.6 m displacement was intersected 5 m past the overdriven heading before the seam was re-entered and continued to a final depth of 733 m. Minor faulting and seam rolls were also detected.

ACIRL then acquired a Longyear LMC75 drill rig to drill NQ sized, 96 mm diameter boreholes to depths of 1233 m and 1535 m (Figure 2.28) to prove the future longwall domain to the north of the colliery to be free of structures (Walsh and Hungerford, 1993).

With the standard practice of defining the seam profile as drilling progresses into an area, directional drilling was established as an exploration tool. Refined interpretation allowed areas to be proved clear of structures which could adversely affect mining operations (Williams and Hungerford, 1988). With improved development rates and pressure on providing longwall continuity, mines could not afford to unexpectedly intersect a geological structure which delays mine to any extent.

Research has been directed at developing geophysical tools for detection of seam boundaries (roof and/or floor) and mylonite structures (Hanes, 1997a). Gamma has been successful in some environments at detecting roof or floor by suffered in Australia with variably strata. Radiometric sensors were not successful and robust enough to be introduced into operational drilling programs. No geophysical tools have been successfully developed for underground in-seam drilling. Ultimately, the observation by experienced drillers of feed and water pressures and return water colour and adequate recording of that information has been the key to providing exploration information from each borehole.

8.2.2 Directional Drilling to Locate and Define a Fault

When stone is intersected during in-seam drilling, the position of the intersection has to be compared with the trend of the seam. If not consistent, the driller has to decide whether it is a change in dip or some fault displacement. If a stone face has been encountered by either a single intersection of a second intersection at the same depth, the usual practice is to drill downwards rapidly to attempt to find and re-enter the seam
(Figure 8.3, Walsh and Hungerford, 1993). If the displacement is downwards and the borehole trajectory is not too steep, the borehole can be continued in coal. If the borehole trajectory is too steep, it becomes sacrificial to find the seam. When the seam has been found, another branch can be planned from before the fault to intersect the seam beyond the fault at a more acceptable trajectory allowing drilling to continue within the seam.

![Figure 8.3: Identifying and defining a fault (Walsh and Hungerford, 1993)](image)

When larger displacement faults are intersected unexpectedly, study of surrounding surface boreholes and adjacent mine workings can confirm the likelihood of the fault and in which direction the displacement is. Mine geologists may be able to give an indication of likely displacement direction from knowledge of stress fields in the area. When exploring the other side of the fault, the lithology of the surrounding strata can be used as a guide to where the borehole is in that lithology.
When drilling to determine the displacement of a known up-thrown fault, concerns included the stability of the fault plane and whether more faulting existed beyond the initial fault. The fault face was intersected at 672 m (Figure 8.4, Hungerford, et al., 2013a). A “sacrificial” branch upwards to re-intersect the Bulli seam defined the upward displacement of 20 m. A second branch was designed and drilled to intersect and continue within the Balgownie seam. When drilling conditions deteriorated in the Balgownie seam, the borehole was directed down to intersect the Wongawilli seam. Intersections with known coal seams down to the intersection with the Wongawilli seam proved no substantial faulting was present beyond the initial fault plane.

![Figure 8.4: Intersection with a fault and definition of the displacement (Hungerford, et al., 2013a)](image.png)

### 8.2.3 Intersection and Definition of Igneous Intrusions with Directional Drilling

Directional drilling has been established as viable means for confirming the presence and location of igneous intrusions which may have been detected by surface or aeromagnetic surveys (Hatherly, 1988). With access to igneous intrusions through in-seam drilling, the nature, size and characteristics of the intrusions can be defined (Ross, et al., 1987; Thomson and Hungerford, 1992). The drilling differs in defining a dyke intrusion which is usually vertical through the seam while a sill intrusion is usually within and parallel to the seam.
Exploring dyke intrusions

With directional drilling, several intersections of the dyke can be completed (Figure 8.5, Grosse, 2005) to determine the alignment as well as the characteristics of the dyke. The location of the dyke can then be accurately plotted on the mine plan (Figure 8.6, Grosse, 2005) to allow nine planning.

Figure 8.5: Profile of dyke intersections with in-seam boreholes (Grosse, 2005)
When exploring for dykes, the preference is to drill towards the middle of the extrapolated seam profile to provide an assessment of the maximum thickness of any dyke intersected. Indications of a dyke can start with the presence of cinder detected a grey frothy flushing which can be associated with harder drilling. The flushing colour of dyke is usually white with an apparent green/blue tinge. If the dyke is found to be hard, the geologist can then plan a branch prior to the dyke to re-intersect the dyke. Directional drilling is halted for core sampling through the dyke for testing.

When a mine has identified dykes by intersection in gate-road development, the alignment of the dyke can be reasonably accurately predicted. The nature of the dyke through the longwall block can change in thickness and hardness to adversely affect the longwall operation. In Figure 8.7, a large igneous plug was not adequately defined by rotary drilling before intersected by the longwall face. If dykes are intersected by development mining, directional drilling is now planned to more accurately define the nature of the dyke within the block. This is to determine if the dyke can be extracted by the longwall or if alternate excavation is required before longwall operations intersect.
the dyke zone. Figure 8.8 shows the thickness of a dyke defined by drilling (blue lines) and the planned drilling (orange lines) for the next longwall block.

**Figure 8.7:** Large igneous plug not adequately defined by rotary drilling

**Figure 8.8:** Dyke definition with directional drilling

When drilling in an area where the presence of dykes is unknown but needs to be identified, there is a chance of drilling through a thin dyke without noticing a change in
drilling parameters or colour which would identify a dyke (Hungerford and Ren, 2013). In this case, drill cuttings are sampled every 6 m and labelled. The samples are dried and inspected with samples which are suspect (dull, vesicular or lacking banding) are sent to the laboratory for basic Proximate analysis to determine the percentage of volatiles present. This allows any dyke, cinder or heat affected coal to be detected and further investigated.

Exploring sill intrusions

Sill intrusions are usually identified through the regular intersections with seam roof or floor as part of the progressive seam profile definition. As with dykes, the presence of cinder is the first indication of the presence of a sill. This is followed by white returns usually with a green/blue tinge. When a sill has been encountered, additional instructions are supplied by the mine geologist on intersection intervals and sampling requirements. A sill was encountered in the upper section of the seam shown in Figure 8.9 and found to eventually replace the coal in the line of drilling.

![Figure 8.9: Identification of a sill in the roof of the mining section](image)

In-seam directional drilling of boreholes to flank proposed development roadways for inrush protection identified a sill on the seam floor (Figure 8.10). Additional drilling
proved the extent of the sill (Figure 8.11, Byrnes, 2007) which prompted a move of the longwall installation roadway to avoid dealing with a clay floor.

Figure 8.10: Identification of cinder associated with a sill

Figure 8.11: Plan of in-seam boreholes defining the extent of a sill and alignment of the dyke

(Byrnes, 2007)
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8.2.4 Exploration Techniques Provided by Seam Profile Definition

The standard progressive seam definition used in directional drilling has been adopted for number of exploration purposes. The following are examples of the use of seam definition used for exploration purposes.

Determination of seam thickness

Seam thickness over the length of a borehole can be defined for mine planning purposes. The seam profile is defined as the borehole is drilled to the maximum length. Before pulling out, the geologist defined depths from which to branch for floor intersections which, when combined with the roof intersections, provide the thickness of the seam (Figure 8.12). A mid-seam shale band was used as an additional identifier of seam profile as the borehole was advanced.

![Seam Thickness Profile](image)

Figure 8.12: Profile of seam thickness defined by directional drilling

Determining the location of a seam split

A split in the seam is detrimental to longwall operations designed to extract a certain seam thickness. The case in Figure 8.13 shows a seam thickness of 4 m reduced to 2.5 m with the retreat floor intersections (Hungerford, 2009). Intersections with the
seam split in two boreholes allowed the location and alignment of the split to be accurately plotted on the mine plan for gate-road development planning.

![Diagram showing seam split](image)

**Figure 8.13: Definition of a seam split (Hungerford, 2009)**

**Proving adequate head of cover under an overlying lake**

When mining under a lake, there is a requirement to maintain a specific head of cover between the mining operations and the bottom of the lake. With a measurement of the Reduced Level (RL) at the standpipe, in-seam directional drilling provides the RL of the seam profile well ahead of the current mining operations (Figure 8.14). This provides mine planning with either assured access to the coal reserves or defines the limit to which mining can proceed. Prior to the requirements of inrush protection, the borehole offered proved the area to be clear of structures which may have some connection with the overlying lake.
8.3 DIRECTIONAL DRILLING FOR WATER MANAGEMENT

In-seam rotary drilling was used as an initial approach for drainage of flooded working and for pumping water across longwall blocks. Directional drilling progressively replaced rotary drilling with the ability to accurately place boreholes in designed locations and horizon.

8.3.1 Dewatering a Coal Seam

Directional drilling was used for water drainage of a coal seam in the early 1990’s in the same manner drilling is used for gas drainage. Borehole drilled within a seam with high water content and permeability can drain water from the seam to reduce problems with water in the development mining face. Drilling ahead of development at Gordonstone colliery was used to reduce the water content and thus reduce problems that the clay floor was presenting when wet (Hungerford, 1993).

Long in-seam boreholes flanking planned gate-road development have been used at Ravensworth underground mine to dewater the seam ahead of development mining (Figure 8.15). Lateral deviations of up to 290 m are required to run parallel and adjacent to proposed gate-roads. The boreholes were located 20 m away from the outer rib-line. Although seam contours provide an estimate of the seam profile (orange profile, Figure
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8.16), progressive definition of the seam profile is still required out to the depths of 1100 to 1300 m.

Figure 8.15: Plan layout of flanking in-seam borehole used for dewatering.

In a thick seam with high water content, roof conditions can deteriorate due to influx of water from the upper section of the seam. Boreholes drilled for water drainage need to cross the full section of the seam drain water from above impermeable layers. This can be done as the borehole advances or with branches on retreat (Figure 8.17).

Figure 8.16: Borehole and seam profile of a dewatering borehole
8.3.2 Intersection and Drainage of Flooded Old Mine Workings

Drilling to intersect flooded old mine workings has been used for several applications. The boreholes allow drainage of the old workings to remove the threat of high pressure water to allow development to progress safely in close proximity or to intersect the old workings. The pressure of the water in the old workings needs to be determined so adequate precautions can be identified in a risk assessment before drilling commences.

The rib support practices used in the old mine should be determined to define what rib bolts, straps or steel sets are likely to be encountered when “holing out” of the rib. All intersections are up from the seam floor to avoid bogging in the any rib spall that exists in the flooded workings. High angled intersections with rib-lines are preferred to avoid an unsupported DHM snapping off if unstable rib-line collapses.

If the borehole is to be reamed, this should be done prior to intersection then rotary drilling can continue the short distance to the intersection. While the borehole is “dead-ended”, flushing can be controlled and applied at high pressure rather than rely on the pressure and flow from the flooded old workings.

Gretley colliery was connected to an adjacent Wallsend Borehole colliery with a flooded section which was going to be re-entered to allow extraction of an adjacent
area. The flooded area was intersected by a 468 m long borehole which allowed drainage of the water (Figure 8.18 and Figure 8.19, Thomson and Hungerford, 1993). Intersection with the DHM with the borehole diameter of 96 mm was deemed sufficient. While drilling a third leg to intersection the flooded workings, debris has washed into one of the previous intersections and jammed the DHM (Thomson and Hungerford, 1993). The drill string was eventually freed with a lesson learnt for the future of a single intersection only per borehole.

Figure 8.18: Plan of water drainage borehole into flooded area of mine workings (Thomson and Hungerford, 1993)

Figure 8.19: Profile of drainage borehole at Gretley colliery (Thomson and Hungerford, 1993)
8.3.3 Drainage of a Water/Slurry Filled Shaft

With the accurate placement of boreholes, directional drilling is capable of intersecting voids filled with water under high pressure while maintaining control of any flows and pressure in the borehole. Directional drilling has been used to intersect water filled shafts to allow drainage of the shaft to an acceptable level before being intersected by mining. All aspects of dealing with high pressure are defined through a risk assessment process so the drilling and drainage process maintains control of the high pressure water.

With azimuth accuracy rated at +/- 0.50, the drill site should be within 330 m of the shaft to provide the confidence of intersecting the 6 m wide target.

The time available to drain the shaft usually dictates that the usual 96 mm diameter of a directionally drilled borehole is not adequate so reaming the borehole to a larger diameter is included in the process. For that, the diameter of the standpipe is increased to suit the reaming diameter. The length of the standpipe is increased to ensure adequate sealing to handle the pressure considering the state of the coal in the rib. The standpipe fittings are designed to handle the pressure and include an extension which allows the DHM to be extracted while maintaining a seal and control of the water. The use of DHM, non-return valve, standpipe and pressure testing and thrust capacity of the drill rig provided the security necessary to control the water pressure involved.

Unless accurate levels are available to design a vertical target, directional drilling usually includes a roof or floor intersection 20-30 m short of the planned intersection to confirm the profile of the seam and allow a specific horizon to be targeted at intersection. If the borehole is to be reamed, directional drilling is stopped 5 m short of intersection. The borehole is reamed before continuing rotary drilling through to the intersection. The reamer is fitted with a non-return valve to prevent water flowing back through the drill rods.

To test that the system can manage the expected pressure, after drilling out the end of the standpipe and at predetermined depths, the system is pressure tested to a defined pressure. This assesses the integrity of the standpipe grouting and checks for flow paths out of the face or into adjacent roadways.
8.3.4 Drainage of Shaft Involving High Pressure Water/Slurry

In the following example (Figure 8.20), a shaft was 260 m deep but the water/slurry level had been reduced to 160 m by bucketing from the surface. The slurry had a specific gravity of 1.3 and applying a safety factor of 50%, the pressure capacity of the system had to be 3.15 MPa. A 150 mm diameter standpipe was installed to 18 m depth to allow for eventual reaming of the borehole to 145 mm diameter. With the location and size restrictions at the drill site, the borehole design included a lateral curve and had to cross an up-thrown fault identified earlier by directional drilling (Figure 8.20).

The borehole (NWIS05) was drilled to 279 m with directional drilling with pressure testing undertaken at 18, 160, 240 and 279 m (Figure 8.21). The borehole was reamed out to a diameter of 145 mm before being drilled the remaining interval to intersection.

The slurry was drained down to the height of the standpipe, leaving a depth of 20 m remaining in the shaft which was further drained by another borehole from a site closer and at a level below that of the floor of the shaft/seam floor intersection.

The experience identified the need to consider clay bands as potential conduits and the need to consider pressure grouting of polyurethane injection of the face before drilling.
Figure 8.21: Profile of drilling to high pressure shaft intersection

8.3.5 Drilling an In-seam Borehole for Water Pumping

In-seam directional drilling has been utilised to provide boreholes across the longwall block for pumping waste water. Ulan introduced a system of drilling boreholes across the longwall block ahead of development mining (Allan, 1988). After development intersected each borehole, a temporary standpipe was installed and a diaphragm pump connected to pump high water make from the face. Water pumped through the longwall block was allowed to free flow down the adjacent main-gate roadway floor to a surface borehole pump station. This removed the need for pipework and staged pump stations to remove waste water from the panel.

Cordeaux colliery utilised directional drilling to provide a borehole which crossed three proposed longwall blocks at the low point along each gate-road. The seam roof was intersected before the line of the each proposed gate-road to ensure the borehole crossing was at a manageable height in the rib-line. A temporary standpipe was installed to allow pumping through the seam to the drill site where a pumping station had been installed. This avoided the need to install pipework and pumping stations in each panel to remove the waste water. As each panel intersected the borehole, a temporary standpipe was installed and pumping resumed.
A similar situation existed at Springvale mine in recent years. High influx of water into the mine workings required pumping through the longwall blocks to avoid high maintenance pumping systems. To ensure the security of each cross-block against the possibility of crushing, each borehole was reamed and a conduit pulled back through the borehole using the drill rig.

8.3.6 In-rush Protection Drilling

In 1996, a water inrush occurred at Gretley colliery (Staunton, 1997). Development at the mine was advancing towards old workings which were known to exist, the location of which had been inaccurately defined by old mine plans on record. The incident highlighted the dangers that inrush presents and the disruption it can cause to the mining operations. Legislation was introduced that requires all underground coal mines to put systems in place to ensure such an event is not repeated. All underground mines in NSW have a responsibility to identify any possible inrush risk, to define that risk and to put plans in place to either avoid or mitigate that risk.

Where mining is planned in the vicinity of old workings, in-seam directional drilling between the old workings and the proposed mining is required to provide an in-rush protection barrier (Figure 8.22, Hungerford, 2005). Subsequent drilling to intersect the old workings can provide boreholes to drain water and eliminate the risk due to high pressure water filled voids.

![Figure 8.22: In-seam borehole to provide a barrier for protection against inrush (Hungerford, 2005)](image-url)
8.4 COMBINED PROGRAM OF EXPLORATION, INRUSH PROTECTION AND WATER MANAGEMENT DRILLING

An example of the use of most aspects of exploration and water management with directional drilling is evident at Dendrobium colliery (McNaughton, 2002a). The mine was planned to extract the Wongawilli seam with inrush threats from the adjacent flooded Nebo and Kemira mine workings (Figure 8.23, McNaughton, 2002b). The first two longwalls would be under the Mt Kemba mine workings (Figure 8.24, McNaughton, 2002b) which were known to contain water likely to be connected to the flooded Kemira mine. Development of the Nebo Mains panel would also pass under the Cordeaux dam on the surface (Figure 8.25, McNaughton, 2002b).

![Figure 8.23: Mine plan between adjacent flooded mines (McNaughton, 2002b)](image)

![Figure 8.24: Mine plan with overlying flooded mine workings (McNaughton, 2002b)](image)
Drilling for inrush protection

An extensive drilling program was designed to provide inrush protection, ensure against geological features which may connect to any of the sources of water, assess the extent of the water sources and to drain water where applicable. All drilling was through standpipes grouted into the face to manage flushing returns and any intersection with high pressure water. With drilling in the lower mining section of the seam and with the complicated and unstable strata in the upper section of the seam, the drilling was designed to use the seam floor for seam profile definition (Figure 6.10).

Directional drilling provided in-seam flanking boreholes ahead of development adjacent to the flooded Nebo workings in the Wongawilli seam to provide an inrush protection barrier (Figure 8.26). These boreholes were continued ahead of development which passed under the Cordeaux dam. Cuttings samples were taken every 6 m to identify any dyke structure which may provide a water flow conduit from the surface dam, overlying workings or adjacent workings. Pumping boreholes were drilled from 17 and 19.5 cut-throughs and water drainage boreholes were drilled from 10.5 cut-through to intersect the old Nebo mine workings. The boreholes from 10.5 cut-through were reamed to 145 mm diameter for water supply.
Flanking in-seam boreholes also provided an inrush protection barrier between Kemira panel and the flooded workings of Kemira colliery (Figure 8.27). Boreholes were also drilled from 6.5 cut-through and the end of Kemira panel to intersect the old flooded workings. These boreholes were reamed to 145 mm diameter for water drainage.

No structures were found to have connectivity to known water bodies.
Drilling for water drainage

Directionally drill cross-measure boreholes were drilled to target expected voids in the old Mt Kembla workings at locations above the longwall blocks (Figure 8.28).

The cross-measure boreholes into the overlying workings allowed assessment of the flooding in those workings. Dry boreholes and boreholes which intersected water allowed the pressure head to be measured. This defined the level of water to be at an RL of 198m with an estimated stored capacity of 256 Megalitres (ML).

Water inflows into the current mine and in the old workings have a salt content which precluded it from being disposed of in the local fresh water creeks. As part of the planned water management scheme, a 300 mm pipeline was installed from the mine 7 km down to the nearest tidal creek to allow disposal of the water.
Water from the overlying workings was removed at a rate of approximately 1.2 ML/day until relatively dry before the extraction of the first longwall. Water was also drained from the Kemira workings and all fed by gravity down to the salt water outlet.

**Utilising the boreholes into the flooded old workings for water management**

Although the adjacent flooded workings presented an early hazard to be managed, they also presented a storage capacity to allow management of water inflows into the mine. The mine has approval to dispose of salty water through its salt water outlet but this water must be free of solid contaminants. To ensure this, waste water from the mine is pumped through the boreholes into the Kemira workings (Figure 8.27) and clear water removed through the boreholes from the end of the panel some 500 m from the pump-in point. The flooded old workings act as a settlement system. The system has a current capacity to pump in and remove 8 ML/day but is regulated to maintain a desired head of water in the old workings.

Boreholes drilled to intersect the Nebo workings (Figure 8.26) are being used to provide clean water for the fire-fighting line and used for dust suppression.
8.5 SUMMARY

Directional drilling is now well established as an in-seam exploration tool used for the identification, definition and/or confirmation of structures which may have adverse effects on the economic operations of a mine. To have either, early knowledge of such structures allows mine planning to cater for the structure or make adjustments to manage it. Proving an area of ground clear of structures and providing proof of seam thickness with directional drilling can now provide a mine with an extent of economic security.

Through experience and planning, drilling methods have been developed to adequately and appropriately identify and define most of the structures experienced in underground coal mines.

Through the development of directional drilling in underground coal mines, several research projects were funded to develop down-hole geophysical tools. No geological tools have been developed successfully to the extent of being installed on an operational survey system to provide “real time” readings and assessment. A much promoted seam “profiler” has not been developed to the state where it can be used effectively by drillers on site.

Legislation in Australia now requires that each mine assesses the mining environment to identify possible inrush threats, define their extent and take appropriate actions to remove the threat. Directional drilling is used extensively to provide inrush protection barrier pillars between old workings and future development and has been used to intersect water filled voids such as old mine workings and new ventilation shafts for drainage or recirculation of water.

Drilled intersections with adjacent old workings can provide access to a water storage capacity to recirculate water for mine use or manage the disposal of waste water.
CHAPTER NINE – PRACTICES OF SLIDE AND ROTARY DRILLING FOR EXTRA-LONG IN-SEAM BOREHOLES

9.1 INTRODUCTION

Directional drilling in Australian coal mining has been developed to a stage where standard directional drilling practices allow boreholes to 1400 m to be drilled regularly using slide drilling mode. The record depth for in-seam boreholes was 1761 m in 2002 in Australia (Valley Longwall, 2002). The effects of friction in repeated opposed curved boreholes drilled using the “flip-flop” method for directional control are explained in this chapter. The limitation to borehole depth has been identified as surging feed due to flexing of the drill string through each curve in the borehole and the eventual release as borehole friction is overcome. Variations in the length of each interval drilled with each DHM orientation produced changes to the in-hole friction effects which influenced the maximum borehole depth achieved. These are displayed as trends of feed pressure over borehole depth.

A new longwall domain was to be developed without access to pre-drain the proposed development with the usual medium to long in-seam boreholes. An opportunity was offered to trial a combination of slide and rotary drilling to attempt to extend the length of in-seam underground boreholes to depths of 2000 m and beyond to cover the length of the proposed longwall development and block. Prospective limitations in surveying and water pump capacity identified and equipment and drilling practices to be used and defined prior to the project are described in this chapter.

Boreholes depths were progressively extended beyond that previous achieved underground as understanding and drilling practices improved with experience. The effects of borehole friction are assessed in this chapter through trends of thrust demands, borehole deviations and relative slide/rotary mode usage. The performance of the potential equipment limitations is also presented.

9.2 RESTRICTIONS IN BOREHOLE DEPTH WITH DIRECTIONAL DRILLING USING SLIDE MODE

In achieving a depth of 1005 m with the earliest NQ rod size configuration, the feed started surging beyond 60 m so the rate was progressively reduced as borehole depths
increased to prevent stalling of the DHM (Hungerford, *et al*., 1988b). The surging was attributed to flexing and buckling of the rods through each opposing curve of the “flip-flop” for of directional drilling used. As friction between the drill rod string and the side of the borehole was overcome, the rods would side to the under-side of the borehole with a resultant surging forward of the drill it. This surge forward would cause increased torque loading on the DHM and a spike in the water pressure. Higher penetration rates and greater borehole depths resulted in larger spikes in pressure. As the water pressure increased with borehole depth, this spike would take to water pressure past the maximum pressure setting of the high pressure water pump. Reduction in water flow reduced the rotational speed of the DHM and with the increased torque loading, would stall the DHM. To avoid stalling the DHM, the feed rate would be reduced with a resulting reduction of the fluctuations in water pressure. But eventually even with very slow feed rates, surging and the resultant stalling would stop the drilling and limit the depth of long-holes.

A 73 mm diameter Accu-dril DHM was offered to the industry in 1992 through Asahi (Walsh and Hungerford, 1993.). This unit had a non-magnetic, high-torque, low-speed 4-5 lobe motor section (Hungerford, 1995) which, when fitted with a 1.25º bend and combined with a 96.1 mm diameter PCD bit, greatly reduced surging (which had been attributed to in-hole friction) and drilling rates improved. In 1993 and 1994, the first two boreholes drilled with this configuration achieved lengths of 1233 m and 1535 m (Walsh and Hungerford, 1993). This configuration was established as the standard for in-seam drilling in Australia and eventually the world. The response curves displaying the angular changes in both vertical and lateral planes achieved by this bit and bend configuration are displayed in Figures 6.7 and 6.8.

With the higher thrust loading involved, the capacity of in-hole equipment was being tested. Analysis of drilling data collected from long-holes allowed torque/drag models to be established, showing the NQ drill rod strength was adequate for the depths achieved to that point (Gray, 1991), but that borehole depth would eventually be limited due to helical buckling with the current drilling techniques (Gray, 1992). Tests of rod strength had proven that the preferred drill rod joint known as CHD 76 being adopted by the industry was the superior rod in strength and ease of handling in joining (Gray and Daniel, 2000). Withdrawal friction and rotation were also thought to be limiting factors from these analyses.
CHAPTER NINE
Practices of Slide and Rotary Drilling for Extra-long In-seam Boreholes

Another source of drag in the borehole has been the effect of larger cuttings lying on the bottom and up the sides of the borehole (Figure 9.1, Thomson, 2007).

![Figure 9.1: Cuttings bed in borehole section (modified from Thomson, 2007)](image)

Modelling of water velocity profile in a borehole indicates the highest velocity and the majority of fluid flow in the larger section above the rods lying in the borehole (Figure 9.2, Wang, 2013). The much lower flow velocity down the sides of the rods is not adequate to move heavier cuttings which accumulate and add to the drag. Additives such as drilling muds and foams are only used in spot applications to either increase the viscosity of the flushing fluid or increase the velocity, and these are effective when used. Bits with smaller cutters have been tried to produce smaller cuttings but the results were inconclusive and a perceived reduction in penetration rate does not appeal to the drillers.

In the late 1990’s, an in-seam drilling project for US Steel in the USA involved drilling 1500 m long boreholes (Kravits, et al, 1999). For that project, impregnated bits were used with a high speed DHMs fitted with a 1.125° bend. The smaller bend provided reduced curvature in the borehole while the impregnated bit produced very fine cuttings easily removed from the borehole. This would have reduced drag from accumulated cuttings with the added benefit of leaving the finished borehole relatively free of cuttings. The flat-faced impregnated bit did not gouge into the face when cutting so was not prone to stalling the DHM when pushed into the face by the inevitable surging at depth. The combination ensured 1500 m was achieved in each borehole but the much slower drilling rates involved were not readily acceptable for the project.
CHAPTER NINE
Practices of Slide and Rotary Drilling for Extra-long In-seam Boreholes

9.3 THRUST TRENDS

As longer boreholes beyond 1000 m were being drilled, data for analysis of thrust requirements was more readily available although the steering parameters were unregimented with drillers using DHM orientations as they wished to achieve directional control. “Flip-flopping” at 6 m intervals was the established slide drilling control mode for drilling along a desired heading. The drilling in one borehole could extend to a depth several hundred metres deeper than a previous borehole in the same environment without any apparent changes in drilling parameters.

Plotting of thrust versus borehole depth identified a trend of progressively increasing thrust as the borehole depth increased (Figure 9.3, Hungerford, et al, 2012). This was attributed to progressively increased friction through bends in the borehole as thrust increased. Most drilling with orientation changes at 6 m intervals reached depths beyond 1000 m before thrust requirements increased rapidly. Only the occasional borehole reaching beyond 1300 m before surging stopped drilling short of the maximum thrust capacity of the drill rig (Section 6.2.2).
Two boreholes have been drilled with the driller using changes in orientation at 3 m intervals. The boreholes depths were greatly reduced and inadvertently provided further proof of shorter interval changes in curve increases in-hole friction. As displayed in Figure 9.3, the thrust requirements in Borehole MG40-30-1 increased more rapidly from 350 m with a final depth of only 800 m achieved. On site, the drillers thought they were encountering very hard strata with regular stalling of the DHM and stopped drilling. When the thrust was plotted against borehole depth, it was apparent that in-hole friction had caused surging and stalling of the DHM. This was similar to the regular stalling of the DHM beyond 1200 m but at a shallower depth. In both cases (816D40 and MG40-30-1), the Fletcher drill rig was used with a maximum capacity of 114 kN which was not reached.

When having difficulties achieving the 1400 m design depth on a project, 12 m intervals at each orientation was used instead of the usual 6 m. Boreholes were successfully extended to beyond 1400 m. The thrust trend (Figure 9.3) indicated higher loads than that with 6 m intervals as the drilling progressed but did not experience as rapid increase in thrust loading to stalling. Stalling did eventually stop drilling at 1400 m at thrust pressures well short of the 140 kN capacity of the Series 1000 drill rig being used.

In most cases, where only directional slide drilling is being used, it is likely that the effects of in-hole friction (on curves) causing surging determine the maximum depth
that can be achieved. The ability to drill further is restricted by surging in the borehole due to friction rather than the maximum feed capacity of the rig being reached. It is evident that the severity and frequency of bends, particularly in the initial stages of a borehole, influences the rate and final depth achieved with drilling. Any attempt to drilling long boreholes would require limiting the bends in the borehole, particularly in the initial stages.

9.4 TRIAL ROTARY/SLIDE DRILLING PROJECT

Metropolitan colliery was developing into a new area of the Bulli seam which had high gas content with carbon dioxide being the dominant gas. Limited access did not allow the standard gas drainage drilling program to be employed to drain the gas prior to mining. Drilling shorter boreholes would necessitate a staged and disrupted development to allow progressive drilling and gas drainage. With the proposed gate-roads being 2000 m long, the colliery approached VLI to attempt drilling long, in-seam boreholes to 2000 m and beyond to provide gas drainage.

To extend borehole depths to 2000 m, a combination of directional slide and rotary drilling was proposed to be applied from the start. Slide drilling mode involves feeding the DHM into a borehole using “flip-flopping” orientations to provide directional control. The rotary drilling mode involves rotating the drill string including the DHM over extended lengths while the desired trajectory and alignment are maintained.

9.4.1 Potential Surveying Depth Limitations

The survey systems were thought to be a limiting factor with the DDM-MECCA initially preferred over the DGS due to signal strength. Previous boreholes drilled to and beyond 1500 m had suffered from poor signal strength problems when using the DGS. Subsequent development of the DGS (McCabe and Hellyer, 2013) had apparently improved signal strength and transmission but this had yet to be proven over the longer lengths.

To enhance the chances of successful signal transmission at depth, relatively new CHD 76 rods were used with the DPI Rod Communication System (RCS) – similar to the AMT MECCA installed.
9.4.2 Configuration of the Down-hole Drilling Components

The DPI supplied equivalent of the non-magnetic 4/5 Accu-Dril DHM was used with a 1.125° bent housing fitted with a 1 mm thick wear pad. A standard Asahi 96.1 mm diameter PCD bit was used which combined for an off-set at the bit (B, Figure 3.40) of 6.7 mm. This was equivalent to a bend of 1.22°. In initial rotation of the DHM, this would cause the heel of the bend to be flexed 2.9 mm to fit within the 96.1 mm diameter until the borehole diameter was increased by the rotation.

After the first hole, the bit diameter was increased to 99 mm by repositioning the outer cutters outward. This reduced the DHM deflection off-set at the bit to 5.3 mm and thus reduced the effective bend to 1.12°. In rotating the DHM, the heel of the bend fits within the 99 mm diameter thus avoiding any flexing of the DHM.

Directional control had been identified as a problem in previously drilling the longest borehole to 1761 m. At depths greater than 1500 m, the loading and release of torque in the drill string caused wildly inaccurate orientation settings of the DHM which made directional control very difficult.

9.4.3 Drill Rig Thrust Capacity

Due to a combination of size limitations in getting equipment into the mine and availability of drill rigs, the initial drill rig supplied for the project was a modular VLI Series 1000. This drill rig had a thrust capacity of 104.6 kN compared to 140 kN of the track mounted Series 1000. The track mounted drill rig was used for the last three boreholes.

9.4.4 Defining the Drilling Practice to be employed

Several methods had been employed previously or proposed to extend underground boreholes depths. These included:

- Reaming sections of the borehole to a larger diameter (Valley Longwall, 2002),
- Reducing the bend on the DHM, using an impregnated bit and high speed 1/2 lobe DHM (Kravits, et al, 1999),
- Employing a rotary/slide method of drilling commonly used by SIS drilling and previously in some underground drilling operations (Eade, 2002).
CHAPTER NINE
Practices of Slide and Rotary Drilling for Extra-long In-seam Boreholes

Before drilling commenced, the drillers were instructed on the drilling practices required for the project. Most drillers had used rotary/slide for short sections of drilling on previous projects so were comfortable with the practice. Rotary drilling was to be used whenever the drillers were comfortable with the position and alignment of the borehole and limited to rotational speeds of 30 – 60 rpm to limit damage and wear to the DHM. In addition to usual data recording with intermediate check-shot surveying, the drillers were asked to record their drilling mode and main pump pressure when in rotary mode.

To manage all returns from the borehole, drilling was through a 150 mm standpipe grouted into the face with suitable valves and fittings attached. With high gas flows expected from the boreholes, the rig was set back from the face to allow a 3 m enclosure (Figure 9.4) for withdrawing the DHM from the hole.

![Figure 9.4: Standpipe configuration with 3 m enclosure](image)

### 9.4.5 Drilling Conditions

Ultimately, drilling conditions have an influence on the borehole depths achieved. Good intact coal conditions allow for easy directional drilling with DHMs with minimum problems with in-hole collapse and bogging. If any unstable conditions are experienced, ongoing drilling beyond that point will always be exposed, with loss of expensive equipment being the main concern for the drillers. Blocked boreholes were a real concern for the mine. Over-coring has not commonly been successful beyond 700 m depths so drilling to extreme depths beyond the 600-700 m limit eliminates over-coring
as insurance. Plans need to be in place to eventually recover any bogged equipment when intersected by mining.

The Bulli seam had an average thickness of 3.0 m and although surface boreholes were limited, no geological structures were expected in the area of the proposed drilling.

9.5 DRILLING RESULTS

Eleven boreholes were completed from two drill sites (Table 9.1). All boreholes were drilled with a combination of slide and rotary drilling. The use of slide and rotary modes was unregimented with each borehole having different applications of rotary drilling, off-set entry angle, lateral curve and eventual lateral deviation. That delivered a different depth in each borehole from which slide drilling could no longer continue and drilling was continued with rotary drilling only. The table also indicates the depth to which slide drilling was possible before drilling continued with rotary drilling only, the lateral deviation and the reason for stopping the drilling each borehole. Figure 9.5 shows the location of the eleven boreholes on the mine plan relative to the proposed gate-road development.

<table>
<thead>
<tr>
<th>Borehole</th>
<th>Date</th>
<th>Depth (m)</th>
<th>Slide to (m)</th>
<th>Lat Dev (m)</th>
<th>Terminated</th>
</tr>
</thead>
<tbody>
<tr>
<td>EX03</td>
<td>15/06/2015</td>
<td>1779</td>
<td>1746</td>
<td>116L</td>
<td>No signal</td>
</tr>
<tr>
<td>EX02</td>
<td>19/07/2015</td>
<td>1875</td>
<td>1851</td>
<td>58L</td>
<td>Floor</td>
</tr>
<tr>
<td>DH01</td>
<td>28/07/2015</td>
<td>1971</td>
<td>1803</td>
<td>31L</td>
<td>Floor</td>
</tr>
<tr>
<td>DH04</td>
<td>09/08/2015</td>
<td>2001</td>
<td>1821</td>
<td>129L</td>
<td>Floor</td>
</tr>
<tr>
<td>DH05</td>
<td>31/08/2015</td>
<td>2007</td>
<td>1653</td>
<td>78R</td>
<td>To design</td>
</tr>
<tr>
<td><strong>DH08</strong></td>
<td>23/09/2015</td>
<td><strong>2151</strong></td>
<td><strong>1743</strong></td>
<td><strong>40L</strong></td>
<td>No rods</td>
</tr>
<tr>
<td>DH09</td>
<td>07/10/2015</td>
<td>2103</td>
<td>1761</td>
<td>83L</td>
<td>No rods</td>
</tr>
<tr>
<td>DH10</td>
<td>27/10/2015</td>
<td>2007</td>
<td>1761</td>
<td>121L</td>
<td>To design</td>
</tr>
<tr>
<td>DH11</td>
<td>17/11/2015</td>
<td>2016</td>
<td>1920</td>
<td>166L</td>
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<tr>
<td><strong>DH06</strong></td>
<td>08/12/2015</td>
<td><strong>2007</strong></td>
<td><strong>1923</strong></td>
<td><strong>50R</strong></td>
<td>No rods</td>
</tr>
<tr>
<td>DH07</td>
<td>22/12/2015</td>
<td>2013</td>
<td>1884</td>
<td>0 L/R</td>
<td>No rods</td>
</tr>
</tbody>
</table>
Figure 9.5: Plan of long-hole coverage of proposed longwall gate-roads

Because of limited access from two drill sites, the boreholes could not be designed as straight holes along their target azimuth with a zero lateral deviation. They were designed with off-set entry angles and lateral curves to provide the required drainage coverage and not set up specifically to create depth records. The lateral deviation is plotted for the boreholes from the two sites (Figure 9.6 and Figure 9.7) with reduced lateral control evident towards the end of each borehole due to rotary drilling extending the boreholes.

Figure 9.6: Lateral deviation of boreholes from 6c/t site
Figure 9.7: Lateral deviation of boreholes from 9c/t site

With little geological and RL information in the area of the proposed drilling, the initial borehole (EX03) served as an exploration hole with regular roof intersections to define the seam profile (Figure 9.8). The borehole and seam profile also shows boreholes crossed and the expected location of cut-throughs in the future gate-road development. To make each borehole more manageable when intersected by mining, the preference was for the boreholes to be located comfortably within the seam section at each proposed crossing. This borehole, which had established a new record for underground drilling in coal mines, was terminated with survey signal problems at 1716 m.

Each subsequent borehole increased that record until DH08 established the world record at 2151 m (Table 9.1). Being the first borehole to the left from the 9 c/t site, regular roof intersections were completed for seam profile definition (Figure 9.9).

The drilling provided the mine with good seam profile definition over the area. Drilling conditions were found to be stable with no structures or boggy conditions encountered.
Borehole DH08 was drilled with a combination of slide and rotary out to 1743 m (Table 9.1, Figure 9.10); at which point 45% had been slide mode with the balance of 55% in rotary mode. The drilling mode has been averaged both over 30 m intervals and as a progressive average. When slide drilling could not continue, the continued rotary drilling reduced the average in slide mode to 37%. The calculation of drilling mode
percentage includes drilling only in the trunk hole in which the rods are laying as the borehole progresses. Additional slide drilling involved for steering the borehole up to profile defining roof intersections is not included in the averages.

With each borehole, the portion of slide drilling starts high as the drillers establish directional control and lateral positioning before introducing rotary drilling. The portion of slide drilling increases as the borehole is steered laterally around to the target line and vertically negotiating changes in seam profile. Then it reduces progressively as rotary drilling is used more often.

The plot of thrust on the drill string for slide drilling (Figure 9.11) displays the usual trend of increased drilling rate increase with depth, indicating the frictional effect of curves earlier in the borehole (Hungerford, et al, 2012). The maximum thrust capacity was reached while still using the slide mode of drilling. Prior to introducing the larger capacity track mounted drill rig, the thrust trend from most boreholes (when extrapolated to 140 kN) indicated the greater capacity track mounted Series 1000 drill rig would probably manage slide mode drilling to 1900 m.
In rotary drilling mode, in-hole friction is greatly reduced (Figure 9.11) and only starts to increase gradually from the 1400 m depth. This reduction in friction also provided consistent feed at the bit compared to the surging feed experienced in slide mode.

Surging was still present in slide mode and indicated by the progressive slowing of the drilling rate. The magnitude of the surging was significantly reduced, allowing drilling to progress to greater depths with the maximum thrust capacity being reached rather than stalling causing a stop to drilling. A comparison of the thrust trends from the first two boreholes is provided with that from 12 m interval slide drilling (MG29-A3-DH01, Figure 9.3) in Figure 9.12. Slide drilling was still possible in boreholes EX02 and EX03 but the maximum thrust capacity of the drill had been reached. Surging had stopped drilling in borehole MG29-A3-DH01 well short of the maximum capacity of 140 kN of the drill rig being used.
Figure 9.12: Thrust comparison with slide drilling mode only

With consistent loading on the bit and DHM, drilling rates were more consistent and nominally higher with rotation compared to the rapid reduction in the slide drilling rate to avoid stalling the DHM (Figure 9.13).

Figure 9.13: Drilling rate – slide and rotary drilling
The pressure capacity of the water pump was a concern before drilling commenced with the increase in idle pressure with drill string length likely to approach the maximum available pressure of the pump. The drilling commenced using 200 litres/min water flow to assess the progressive increase with depth and identify any potential problems by extrapolating that trend to beyond 2000 m. As shown in Figure 9.14, the idle pressure increased from 1.5 MPa (at the start) at a rate of 0.15 MPa/100 m to reach 4.8 MPa at 2200 m.

Although the idle pressure was more than 2 MPa below the maximum available pump pressure at depths beyond 2000 m, problems were encountered when starting the DHM. The pressure spike (on starting) occasionally took the water pressure to 7 MPa and stalled the pump before the DHM could start. Lower water flows of 150-170 l/min, which generate lower pressure, were required to start the DHM. Once the DHM was running, the water flow was increased to 200 l/min for drilling. Drilling at flows higher than 200 l/min would require a higher flow and pressure rated water pump. The decreasing drilling rates are shown in the gradually reduction in the differential pressure over the depth of the borehole.
9.6 DIRECTIONAL CONTROL

The usual assessment of DHM steering response has been over 6 m intervals to match the usual 6 m flip-flop drilling method (Figure 6.7 and Figure 6.8). With 6 m intervals unlikely to be used with regular use of rotary drilling, the intermediate 3 m surveys allowed the vertical and lateral response curves to be assessed over 3 m intervals (Figure 9.15 and Figure 9.16). These plots showed response deviations approximately 50% that established for 6 m intervals with a standard 1.25° bend and 96.1 mm bit configuration. The magnitude of deviations with a 1.12° equivalent bend is slightly less than half over 3 m. The scatter of results is indicative of drilling in strata which is not homogenous with the varying strengths of coal and stone bands causing variations in the vertical and lateral responses.

![Graph](image)

**Figure 9.15: Vertical response curve over 3 m intervals**

The deviations over 3 m when rotary drilling after a prior 3 m interval of slide drilling were analysed to determine if the previous slide drilling deflection affected the rotary drilling deflection. The vertical and lateral responses in rotary mode were plotted relative to the preceding responses in directional drilling slide mode (Figure 9.17). No relationships were apparent. The same was done for rotary drilling response versus depth but again no relationships were evident.
Figure 9.16: Lateral response curve over 3 m intervals

Figure 9.17: Rotary drilling responses following slide drilling

The vertical and lateral deviations over each 3 m interval were plotted for both slide and rotary drilling (Figure 9.18 and Figure 9.19). The rotary drilling did not create straight boreholes with some deviations as big as with slide drilling. On average the deviations were reduced as seen by the tighter grouping with most within 0.5 deg/3 m vertically and 1.0 deg/3 m laterally.
Although the drilling was initially limited to rotational speeds below 60 rpm in the pre-drilling planning, traditional rotary drilling variations were tried with increased rotational speed (to 180 rpm) and reduced drilling rate to curve the borehole downwards. Conversely, the rotational speed was reduced and drilling rate increased to curve the borehole upwards. The variations in drilling rate are evident in the scatter of the red values in Figure 9.13. These variations were used successfully to extend all
boreholes past the depth beyond which slide drilling could not continue. The most effective was in borehole DH08 with 408 m being rotary drilled to the final record depth of 2151 m (Table 9.1).

Some lateral control was possible with the vertical control parameters used with rotary drilling. Drilling parameters used for climbing also deflected the borehole to the right and dropping parameters curved the borehole to the left.

9.7 EFFECTS ON DRILLING PARAMETERS

The first two boreholes (DH11 and DH6) drilled using the track mounted drill rig with greater thrust capacity were the most successful with depth drilled before slide drilling could not continue. Borehole DH11 had the greatest off-set angle (at -31.6°) to the target azimuth and has the largest lateral deviation of 166 m to the left. This was an indication of improvements in drilling skills through exposure and experience in the drilling practice and innovation on the drillers’ part.

9.7.1 Hydraulic Capacity for Rotation

Although the thrust loading was only at approximately 25% capacity while rotary drilling, the rotation pump pressure (Figure 9.20) was at approximately 85% capacity at depths beyond 2000 m. This is likely to be a limiting factor in determining maximum depth capacity with the current equipment. Extrapolation of the pressure trend indicates a maximum capacity of approximately 2800 m but some margin of safety would be required when approaching maximum capacity to ensure having the ability to retrieve the drill string from the borehole.
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9.7.2 DHM Performance

The initial DHM completed six boreholes before being replaced due to bearing pack failure. Wear at the bend has always been a problem with DHM drilling but the presence of the 1 mm thick wear pad prevented adverse wear. Erosion was evident to the wear pad and the adjacent steel on the shoulder of the pin thread of the bent housing but had not started to penetrate through to the thread at that joint. The PCD drill bit with the repositioned outer cutters that was used over that period showed limited abrasive wear. Some chipping had occurred on most cutters but not enough to affect the cutting characteristics of the bit.

9.7.3 In-hole Friction When Pulling Drill Rods

Concern had been expressed that tensile loading to withdraw the drill string from the borehole may be higher than compression loading to drill or feed rods in. Figure 9.21 shows the tensile load required to pull the drill string out without rotation. The rapid increase beyond 1700 m would indicate problems may occur with boreholes in excess of 2100 m depth. This was not apparent with DH08 to 2151 m. Pressure readings were not recorded when pulling drill rods from each borehole.
9.7.4 Performance of the Survey System

The survey instrument signal strength reduced rapidly over the first 400 m after which the rate of decay diminished (Figure 9.22). From 1200 m, the signal strength remained reasonably constant at 0.5%. The first borehole (EX03) was terminated with survey signal problems which could not be defined. After that, survey signal problems were insignificant with some difficulties only experienced at depths beyond 2000 m. In those cases, several surveying attempts were occasionally required before a signal was received.
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For this project, DRTG ZN50 grease with 50% metallic zinc within its formulation was introduced. The grease had good adhesion and anti-galling characteristics and with the metallic content, may have contributed to the improved signal transmission.

9.8 SUMMARY

Drilling with a combination of slide and rotary drilling was successful in producing a series of boreholes to and beyond 2000 m and located in positions which should provide adequate gas drainage of the gate-roads before development mining commences. The off-set angle at entry and the eventual lateral deviation had no apparent influence on the final borehole depth achieved.

The components of drilling practice employed to improve the borehole depths were:

- Using a larger diameter bit,
- Reduction in the effective bend,
- Drilling longer intervals with each tool-face setting when slide drilling, and
- The application of rotary and slide drilling.

The straighter drilling provided in the rotary drilled sections reduced in-hole friction and extended the depth to which slide drilling was possible. The reduction in the magnitude of the bend reduced the surging feed usually experienced with slide drilling.
in long boreholes. Thrust capacity to overcome friction was the limiting factor on the slide drilling depth rather than repeated stalling of the DHM through surging.

The increase in the bit (and borehole) diameter from the standard 96.1 mm after the first borehole to 99 mm for the remaining drilling reduced in-hole friction. Rotary drilling dramatically reduced the thrust friction on rods sliding in a borehole.

When slide drilling capacity was reached, rotary drilling effectively continued the drilling with some lateral and vertical control. More understanding of the effects variations in penetration rate of rotation speed have of directional control would be beneficial to future long-hole drilling projects.

Rotation capacity and water pump pressure capacity were within the requirements of drilling in-seam borehole to 2000 m but will be the limiting factors of drilling depth capacity when in rotary drilling mode.

Good profile definition from most boreholes provided the mine with definitive RL information over the area of drilling. No adverse drilling conditions were experienced which may adversely affect gas drainage from the area.
10.1 CONCLUSIONS

In-seam drilling for gas drainage was identified in the 1970’s as the most effective means to mitigate the risk of outburst. Rotary drilling programs were progressively introduced at coal mines as higher gas content coal was encountered. The introduction of directional drilling in the mid-1980’s offered more accurate positioning of boreholes but was limited by the state of the technology of the time. The high volume of in-seam drilling required in Australian coal mines to manage outburst risk meant there has been a great need for technical development and training to improve drilling performance. This was evident in the development of the DDM survey instrument which transformed directional drilling.

The fatal outburst in 1994 which was not managed by the precautions in place at the time prompted the introduction of OMPs. From the drilling aspect, these required that all boreholes be positioned as planned and be surveyed to verify their location. Directional drilling was the only form of drilling which could provide accurate placement of boreholes with surveying to verify their location. Such technology allowed for practical compliance with OMPs and became an integral part of each mine’s OMP.

Drill rigs were imported and modified for use for underground directional drilling applications. A variety of drill rigs, drill rods and DHMs were used as the directional drilling technology was developed. Specifications for underground drill rigs were defined for in-seam directional drilling. These included flameproof electrics, 75 to 90 kW flameproof motor to drive the hydraulic system, a high pressure water pump to deliver 250 l/min @ 10 MPa, a facility to lock the rotation, HQ rod capacity jaws and robust mobility with secure anchoring.

NQ size rods were chosen for strength and rigidity for drilling boreholes to depths beyond 1000 m. Drill rigs are designed with HQ rod capacity to allow over-coring recovery of rods bogged in adverse conditions. CHD 76 drill rods with greater thread strength, longer life and easier handling than NQ rods are now preferred for underground directional drilling.
The range of DHMs is available with higher torque and lower rotational speed ranges with the non-magnetic 4/5 lobe Accu-dril DHM found to be the most versatile DHM available and preferred when both coal and stone are to be drilled. Abrasive wear at the bend of the DHM exposed the thread of the stator to reduce the life of DHMs. Wear pads have been developed including laser applied TC pads, HSP inserts and PCD blanks and are required to reduce the rate of wear. The addition of thick wear pads introduced a problem of excessive flex of the DHM at the bend leading to straightening or failure of the DHM. The magnitude of the bend has to be reduced and matched with the thickness of the wear pad to provide DHM deflections within acceptable ranges.

PCD drill bits capable of drilling both coal and stone were introduced through prototype designs initially for rotary drilling and then directional drilling. Design improvements were made as defects were identified. The key design aspects were identified for PCD bits used for directional drilling applications. Studying the configuration and performance of these bits has allowed suggestions to be made to the manufacturers to improve the design of their bits for directional drilling as well as to provide specific characteristics for penetration rates, the ability to branch, the size of cuttings, consistent torque loading and steering response.

The Asahi PCD bit with a flat faced design of 4 outer cutters and three inner cutters has been established as the best performing bit in both coal and stone drilling. A standard was established for parallel gauge protection with a diameter of 94.5 mm on a 96.1 mm OD bit. PCD cutters have been introduced to shoe bits for over-coring, bits for larger diameter drilling for standpiping and reamers for enlarging directionally drilling boreholes for gas or water drainage.

Surveying is a key element providing both borehole surveying and directional control. Eastman single-shot wire-line surveying was the initial industry standard for directional drilling but was time dependent on borehole depth.

The development of reliable real-time electronic survey instruments revolutionised in-seam directional drilling. A borehole design system was established to provide an indication of the seam profile along the line of drilling. A lateral design is now provided to accurately place each borehole in the correct design position. A system of
exaggerated vertical scale plotting of the borehole was developed to allow progressive plotting of the vertical profile as well as the lateral position of each borehole on site.

During the development of directional drilling technology, drilling practices were established to record drilling data and understand the requirements of operating DHMs, surveying, steering a DHM and managing an underground drill site. Data recording systems for drilling and shift activities have been established. An industry standard of 96.1 mm diameter bit combined with a 1.25 degree bend on the DHM was established to allow consistent drilling to depths beyond 1300 m. Vertical and lateral response curves have been established for drilling 6 m intervals at orientations of the DHM for the standard configuration of bit and bend size.

Directional drilling has been established for gas drainage to satisfy OMP requirements of accurately positioning boreholes and confirming their position with surveying. Directional drilling of in-seam borehole patterns has been established as a common practice across longwall blocks to provide gas drainage ahead of future gate-road development. Directional drilling has provided longer in-seam boreholes regularly to beyond 1300 m to provide long term drainage of development panels when drilling access is not readily available.

Cross-measure drilling has positioned boreholes in underlying seams and overlying strata to provide both pre and post drainage around a longwall operation to improve gas capture and reduce gas concentrations in the ventilation.

Directional drilling technology has been introduced to the Chinese, Russian and Polish coal industries for gas drainage through a practice of auditing, design, supply, training and ongoing support.

Directional drilling has been established as an effective in-seam exploration tool. Directional drilling has the ability to target specific areas of investigation and provide geological information through drilling parameters and flushing returns. Seam profile definition can prove development panels and longwall domains clear of structures. A process has been developed to define the displacement of any faults identified as a discontinuity of regular seam profile definition. The location and thickness of igneous intrusions can be defined by in-seam directional drilling. Drilling plans have been established to adequately define geological anomalies when intersected.
The ability of directional drilling to intersect specific targets or to position boreholes along planned alignments while having control of any pressure in the borehole has realised applications of water drainage, water management and in-rush protection. In-seam directional drilling has been utilised to dewater saturated seams ahead of mining. Boreholes have been drilled through longwall blocks to allow pumping of water.

As a result of a fatal in-rush event, all mines are required to ensure against intersection of water filled void by mining operations. When mining in the vicinity of flooded old mine workings or other water sources, directional drilling is required to position a borehole between the water source and the proposed workings to provide an in-rush protection barrier. High pressure water filled voids such as shafts or old mine workings have been intersected to allow controlled drainage of that water. Specific controls and procedures have been established to manage the pressures involved.

In-hole friction has been identified as the factor responsible for limiting the depth of boreholes drilled with the conventional directional drilling methods. Surging feed and stalling of the DHM stop drilling before the thrust capacity of the drill rig is reached.

A combination of slide and rotary drilling was employed to reduce the effects of borehole curvature on in-hole friction and allow longer boreholes to be achieved. Boreholes depths were achieved to beyond 2000 m with slide drilling being achieved to beyond 1800 m. Rotary drilling continued each borehole beyond the slide drilling limit with a maximum depth of 2151 m being achieved.
10.2 RECOMMENDATIONS

Although drill rig designs have been established and the drill rigs are operating successfully, the design parameters based on mobile vehicles is excessive, leading to very heavy units. While drilling, the drill rig is a stationary unit. The design parameters should be reassessed with a reduced loading multiplier which would produce lighter and more manageable units.

CHD 76 rods have been established as the preferred rod for in-seam directional drilling but the supply has been variable in thread machining and material quality. Supply quality cannot be assumed so rods should be checked against specification on delivery and the manufacturers and suppliers informed of any discrepancies.

Recent supply of DHMs have had limited interference fit between rotor and stator providing reduce torque capacity. Standard facilities are required to test the torque capacity and rotational speeds of DHMs on delivery to ensure they are supplied to specifications. The matching of stator and rotor sizes from the 1990’s to ensure rated torque capacity has not been apparent recently.

The deflection provided by both bend and wear pad on the DHMs needs to be limited to avoid excessive bend loading through in-hole curves and premature fracture failure or straightening of DHMs.

Standard PCD bit design has aggressive low rake angle mounting of the outer cutters which rapidly produces wear flats on the lead cutting edge when cutting stone. The performance of PCD bits in stone should be assessed with increased rake angle across all cutters on the face.

At present, all small diameter PCD bits in use have a single line of exposure of the inner cutters. To attempt to reduce the load per revolution and improve penetration rates in stone, it is recommended that a manufacturer is asked to produce a bit which duplicates the contact of inner cutters (similar to the outer cutters). The effects on penetration rates and torque loading on the DHM would need to be assessed. The Asahi bit with only three inner cutters would be suited to this modification.
It is recommended Asahi be approached to increase the depth of the lead TC gauge protection insert on the PCD bit to improve erosion resistance. Erosion of the gauge protection rapidly increases after the lead insert has been dislodged.

There is still no operational instrument which accurately locates the seam roof of floor relative to the borehole position. The initial gamma tools were ineffective in the Australian coal environment and research projects to develop such an instrument have been unsuccessful or tools that have been developed are not proven and/or user friendly.

Water usage is high for an underground directional drilling operation with water flows of 200 l/min required. The use of clear water also limits the use of drilling additives. Although recirculation systems have been used in limited cases, further research and development are recommended.
REFERENCES


REFERENCES


REFERENCES

the European Community, Published by the Coal Directorate of the Commission of the European Communities, Verlag Gluckauf GmbH, Essen, Germany.


REFERENCES


http://www.targetdrilling.com/In-Seam-Degasification-Short-Holes.pdf


REFERENCES


Gray I, 1991. Limits of directional drilling for coal exploration, Australian Coal Association Underground Coal Mining Exploration Techniques, Brisbane, 8-9 November.


Grosse, D, 2005. NW longwall blocks, Boreholes WW32-1 and WW32-2, West Wallsend colliery, Unpublished report to Oceanic Coal Pty Ltd, October.

Hadden, J and Cervik, J, 1969. Design and development of drill equipment, TPR 11, USBM.


REFERENCES


Highton, W, 1982. Experience with pre-drainage of seam gas in the western area of the National Coal Board, in *Proceedings of the Symposium on Seam Gas Drainage with particular reference to the Working Seam*, University of Wollongong, 11-14 May.

HMI, 2015. 145 mm hole opener + 94 mm stabiliser assembly drawing.


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Hungerford, F and Thomson, S, 1996. The application of drilling and surveying technology to Australian coal mining, in *Proceedings of the Symposium on Geology in Longwall Mining*, University of New South Wales, Sydney, 12-13 November.


REFERENCES


Knight, S, 1991. The directional drill monitor for long hole gas drainage in coal mines, Underground Coal Mining Exploration Techniques, Australian Coal Association, Brisbane, 8-9 November.

REFERENCES


Longyear, 1984a. Q and CHD wireline coring systems, Diamond core drilling equipment, Information sheet, Reference No. 5MBP 1/84.


REFERENCES


286
REFERENCES


REI, 2009. Cutter coverage of 89 mm PCD bits, Information sheet.


Robertson, B, 2005b. German Creek gas management, *ACARP Gas and Outburst Workshop*, Mackay, 16 September. 


http://www.drillingsupplystore.com/media/Sandvik/coring/wireline_drl_rods_thread.jpg


Tahmoor Colliery, 2004. Set up of stub and c/t for Down-hole motor drilling, Standard work procedure, Tahmoor Colliery, Xstrata Coal NSW.

Taiheiyo Coal Mining Company Ltd, 1996. Outline of Kushiro Colliery, Taiheiyo Coal Mining Company, Ltd.

Target Drilling, 2006. Plugging In-Mine Boreholes and Surface CBM Well Laterals with Reformulated Polymer Gel and TDI’s New Gel Unit, Target Drilling Inc.  
http://targetdrilling.com/plugging-bore.html

Technidrill, 2012. 70HD heavy duty wire line drill rods, Technidrill Eurofor Group.  


REFERENCES


Walsh, R, 1991. Long-hole drilling experiences at West Cliff and North Cliff mines, Underground Coal Mining Exploration Techniques, Australian Coal Association, Brisbane, 8-9 November.


REFERENCES


Yuanping, Cheng, 2010. Theories and Engineering Applications on Coal Mine Gas Control, Xuzhou, China University of Mining and Technology Press.

Zabetakis, M, Moore, T, Nagel, A and Carpetto, J, 1972. Methane emission in coal mines: effects of oil and gas wells, RI 7658, USBM.