HYDROGEOLOGICAL ASPECTS OF ROCK MECHANICS AND MINING SUBSIDENCE AROUND LONGWALL EXTRACTIONS

by

Timothy Robert Colin Aston, B.Sc.

Thesis submitted to the University of Nottingham for the degree of Doctor of Philosophy.

August 1982

To my wife Rosemary whose

understanding and patience

have contributed so much.

"The thing can be done", said the Butcher, "I think.
The thing must be done, I am sure.
The thing shall be done! Bring me paper and ink,
The best there is time to procure."

The Beaver brought paper, portfolio, pens,
And ink in unfailing supplies:
While strange creepy creatures came out of their dens,
And watched them with wondering eyes.

So engrossed was the Butcher, he heeded them not,
As he wrote with a pen in each hand,
And explained all the while in a popular style
Which the Beaver could well understand.

"Taking Three as the subject to reason about -
A convenient number to state -
We add Seven and Ten, and then multipy out
By One Thousand diminished by Eight.

"The result we proceed to divide, as you see,
By Nine Hundred and Ninety and Two:
Then subtract Seventeen, and the answer must be
Exactly and perfectly true.

"The method employed I would gladly explain,
While I have it so clear in my head,
If I had but the time and you had but the brain -
But much yet remains to be said.

"In one moment I've seen what has hitherto been
Enveloped in absolute mystery
And without extra charge I will give you at large
A lesson in Natural History."

from The Hunting of the Snark

by Lewis Carroll.

ABSTRACT

A problem of potential water occurrences exists, whenever coal reserves are worked beneath either surface or sub-surface water bodies. The object of this work is to examine hydrogeological techniques and problems which can be associated with longwall coal extraction.

Current U.K. mine drainage problems are discussed along with the various means by which water can enter a working horizon. An area of principal concern is the potential access of water via fracture networks associated with subsidence profile formation. The role of subsidence development with respect to different geological conditions is considered. Similarly, parameters which control the potential yield of an aquifer, as well as the techniques available for assessing them, are also discussed, with particular reference to Coal Measures strata.

A reappraisal of existing test site data, collected by Nottingham University Mining Department is undertaken and concludes that permeability changes can be linked to subsidence profile development. Permeability monitoring techniques can be used for strata control investigations and reference is made to monitoring the caving characteristics of a massive sandstone roof. Similarly, problems connected with thick sandstone horizons and potential weight bump conditions are also examined.

The effects of subsidence profile formation on the undersea coalfield workings of North-East England are examined with respect to the NCB 1968 guidelines for working under the sea. A geological and hydrogeological reappraisal, concludes that water occurrences can be expected in any area where a potential aquifer horizon has been displaced by faulting into close proximity with a working horizon.

CONTENTS

Page No.

Page No.

Page No.

CHAPTER 6 CAVING CHARACTERISTICS ASSOCIATED WITH A MASSIVE SANDSTONE ROOF

Page No.

Page No.

Page No.

LIST OF FIGURES

Figure No.		Page No.
Chapter 6		
6.1	Plans of Barnborough Colliery Showing the Area Where the Weight Bump Occurred	192
6.2	Location Plan of the East Parkgate Workings Hickleton Colliery	195
6.3	Location of the Investigation Area (Hickleton) in Relation to the South Yorkshire Coalfield	204
6.4	Location of Hickleton Colliery	204
6.5	Geological Information available at the start of the Investigation, East Parkgate Area	206
6.6	Geological Section of the Parkgate Rock, East Parkgate Area	207
6.7	Extensometer Anchor Showing Attachement of Wire	210
6.8	Pulley System for Tensioning Downhole Wire	214
6.9	Calculation of Absolute Strata Movement	216
6.10	Absolute Movement, P30's Tailgate, Hickleton Colliery	218
6.11	Vertical Strain, P30's Tailgate, Hickleton Colliery	223
6.12	Vertical Roadway Closure, P30's Tailgate	226
6.13	Horizontal Roadway Closure, P30's Tailgate	226
6.14	System for Measuring Vertical Distance between the Up and Downhole Baseplates, P30's Maingate	232
6.15	Absolute Movement, Vertical Borehole, P30's Maingate, Hickleton Colliery	234
6.16	Absolute Movement, Inclined Borehole, P30's Maingate, Hickleton Colliery	236
6.17	Vertical Strain, Vertical Borehole P30's Maingate, Hickleton Colliery	242
6.18	Vertical Strain, Inclined Borehole P30's Maingate, Hickleton Colliery	243
6.19	Vertical Roadway Closure, P30's Maingate	226
Chapter 7		
7.1	Geological Information available at the start of the Investigation, East Parkgate Area	259

LIST OF TABLES

LIST OF PLATES

INTRODUCTION

INTRODUCTION

Since antiquity, miners have always encountered the four elements of Earth, Water, Fire and Air:

Earth - in the form of confined spaces, roof falls, variable strata and bad roadway and face conditions.

Water - in the form of inrushes from adjacent yet unrecorded workings and water occurrences from aquifer formations.

Fire - in the form of spontaneous combustion of coal, gas explosions and ever increasing temperatures as mines get deeper.

Air - in the form of asphyxiation due to lack of air, the presence of 'fire, black or stink damp' and the invisible yet explosive mixtures of methane and air.

Even today with an energy hungry world equating coal reserves against those of oil, nuclear power and alternative energy forms, these same words hold true. Research over the past 150 years has solved many of the problems posed by Air and Fire - improved ventilation and gas monitoring techniques and the introduction of non-flammable and spark resistant materials. However, Earth and Water still pose many problems to the mining engineer, even with the advanced technology available today and the considerable resources of mining research establishments throughout the world. The short term roadway support system that will provide maximum support, restrict closure and eliminate 'dinting' is still eagerly awaited. Operational mining problems due to the affects of water have been even more underestimated by engineers. It is only the rapid exhaustion of inland reserves that has drawn attention to the potential problems associated with mining vast undersea reserves that exist not only in Great Britain, but also elsewhere in the world.

The extent to which water occurrences can vary in magnitude has been well summarised by Singh and Kendorski (1). But it was a historical realization that water could occur in any quantity from nuisance to catastrophic that led the National Coal Board during the late 1950's and early 60's to draw up a series of guidelines for working under aquifers and large surface water bodies. The guidelines can be found in the NCB Directive - PI/1968/8 'Working Under the Sea' (2). The directive contains guidelines for both longwall and room and pillar extraction methods. Derivation of the guidelines was based primarily on current experience of safe working conditions and predicted horizontal strain calculated at the base of the water body. Strain is calculated using the National Coal Board - Subsidence Engineers Handbook (3). A similar set of guidelines has been issued by the American Bureau of Mines (4) and deals with standards required for mining close to water bodies under American conditions.

In brief, the British guidelines state that for longwall workings:

'the thickness of strata between the seam top and seabed shall not be less than 105 metres, of which at least 60 metres must be Carboniferous strata. Similarly, the extraction will not endure at seabed, a tensile strain of more than 10 mm/m for both first and successive seam workings.'

To-date the guidelines have proved remarkably successful, but this is almost certainly due to a combinatation of experience and good luck at the time of drafting, rather than a comprehensive understanding of the mechanisms associated with mine water occurrences. The amount of strata cover defined in the guidelines is considered adequate and not open to dispute. However, at several collieries, the limit of 10 mm/m tensile strain at seabed is effectively sterilising large reserves of economic coal in the deeper seams of multi-seam undersea workings. Originally, the value of 10 mm/m tensile strain appears to have been derived in a pseudo arbitrary manner, rather than based on extensive empirical methods and a clear scientific understanding of the mechanisms involved. Certainly, the variation in Coal Measures strata and associated aquifer properties both prior to and post mining have not been fully taken into account. Similarly the presence of faults and fissure/joint networks in the adjacent strata and their behaviour under subsidence profile formation is also considered significant.

A primary objective of this research is therefore to determine using both field and theoretical studies, whether sufficient scope exists for relaxing the 10 mm/m guideline, without increasing the likely incidence of water in the mine workings. Also running parallel with this work is the need to fully understand the caving characteristics of strata associated with subsidence profile formation. This is particularly important when potential aquifer horizons exist above the mining horizon. In order to achieve these objectives, it is necessary to consider the structure and geology of Coal Measures strata, the mechanics of flow through aquifer mediums, development of subsidence profiles, assess fully existing work on the subject and conduct further investigations into the problems. During the investigation period, the author encountered several monitoring and instrumentation techniques, some of which were already well established and others which although relatively unknown, could if fully developed become essential tools for solving mine water problems. Descriptions, applications and limitations of these techniques along with the authors experience is therefore included whenever possible. In this manner, it is hoped that future researchers can develop further some of these techniques and avoid the general 'laws of universal cussedness' which are sometimes encountered.

CHAPTER 1

ORIGIN OF MINE WATER IN BRITISH COALFIELD WORKINGS

CHAPTER 1

ORIGIN OF MINE WATER IN BRITISH COALFIELD WORKINGS

1.1 Introduction

The hydrological cycle governs movement of all free water within the terrestrial environment and of its various components, the most important in relation to British Coal Mining activity is groundwater. Major aquifer systems can be related geologically to the Triassic and Cretaceous periods, although significant quantities can also be yielded from Carboniferous, Permian, Jurassic and Quaternary formations. It is therefore initially proposed to outline the hydrological cycle in order to appreciate the various sources from which groundwater can originate. Similarly, once this has been achieved, principal aquifer systems within Great Britain and their relation to broad geographical mining areas, as well as the occurrence of water at mines within these areas, can be identified.

1.2 Hydrological Cycle

Water within the terrestrial environment is found in three phases: solid, liquid and gas. Hail, snow and ice form the solid phase. Atmospheric rain and free water in streams, lakes and oceans on the surface constitute the liquid phase, while water vapour in the atmosphere and superficial layers of the earths crust form the gaseous phase.

The hydrological cycle envisages that all water is continually in cyclic movement about the Earth's surface, Figure 1.1.

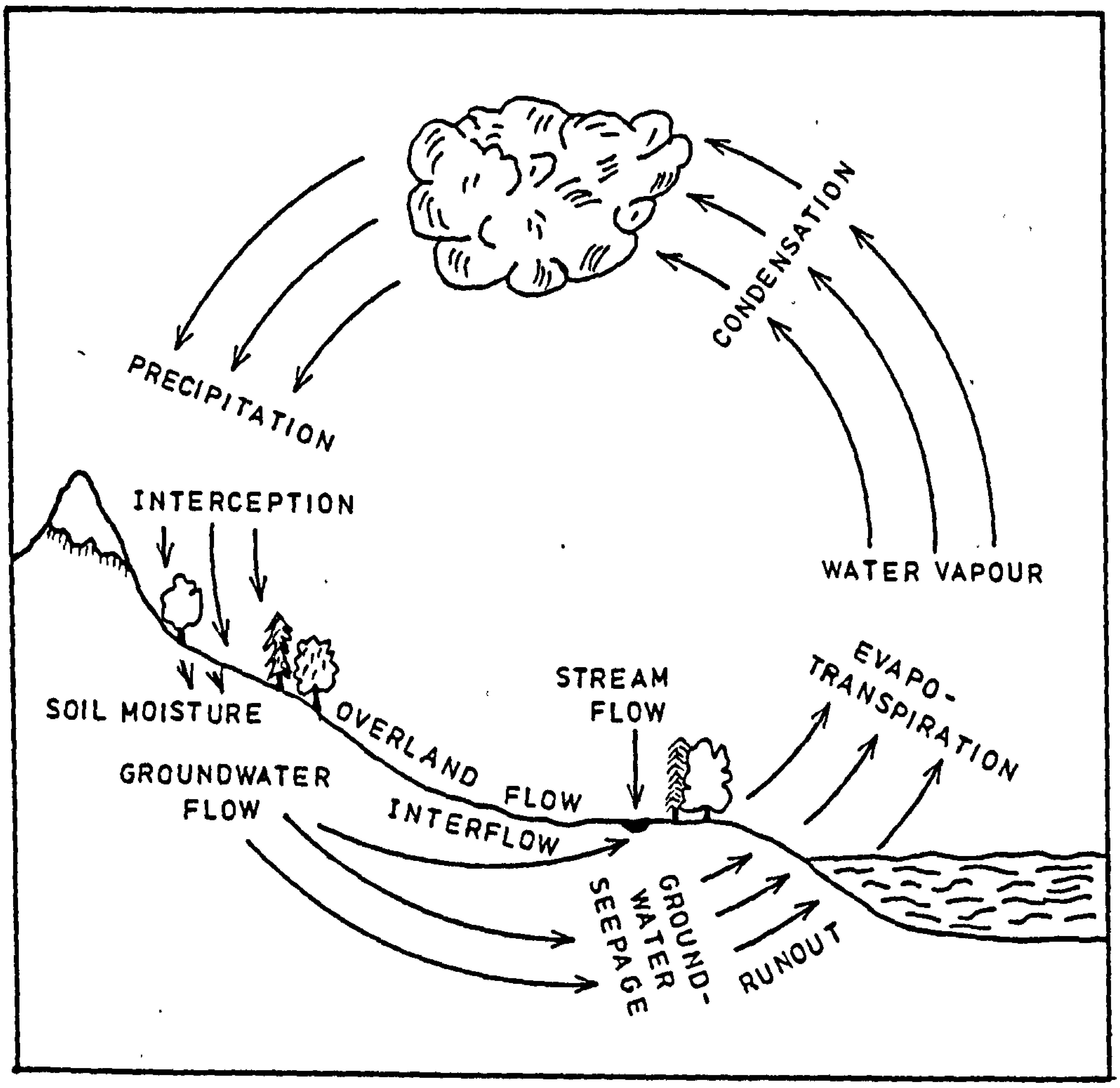


Figure 1.1 The Hydrological Cycle (after Ward (5))

Water vapour originally evaporated from the oceans, condenses and falls as precipitation over land. However, only a fraction of this total precipitation will eventually reach the ground. A proportion is evaporated while falling and yet more intercepted by either vegetation or various surface structures from which it is again returned to the atmosphere by evaporation. Of the precipitation which reaches surface, some will remain in surface storage for return to the atmosphere by evaporation and some will become surface runoff, flowing overland to eventually become streams where it will either evaporate, form seepage and become groundwater or continue as surface flow to the ocean. The final component of precipitation will infiltrate into the soil becoming soil moisture, where it is removed by evaporation and transpiration, interflow towards stream channels or by further downward percolation to become groundwater. The groundwater component is eventually removed by upward capillary motion (to be lost by soil evaporation and transpiration) or by seepage to surface streams and runout to the ocean, where it is again evaporated to the atmosphere.

Finally, while both surface and underground mining operations can short circuit the hydrological cycle, they cannot break it.

1.3 British Aquifer Formations

An aquifer can be conveniently defined as 'a groundwater bearing formation sufficiently permeable to transmit and yield water in usable quantities', Bouwer (6).

One of the most common types of aquifer is unconsolidated alluvial sands and gravels which are found in river valleys and around stream beds. Aquifers of consolidated material can be divided into one of two types:

1) Strata which is permeable and holds water more or less freely throughout its mass, such as sandstone, exhibits inter-granular or primary permeability.

2) Strata which 'en masse' is impermeable, but can hold water within interstitial joints, fissures, solution channels or other karstic developments, such as limestone, exhibits fracture or secondary permeability.

Sedimentary rocks such as chalk or shale often yield significant quantities of water when it is stored in intervening fissure systems. Similarly, many igneous and metamorphic rocks such as granite or gneiss, although practically impermeable 'en masse' can make excellent aquifers when sufficiently fractured or porous (lava when the interstitial vesicles are interconnected).

Table 1.1 shows British stratigraphical eras and geological periods with their main water bearing formations, after Woodward (7) and Bennison (8). The Cretaceous and Triassic periods contain many of the principal aquifer systems of British water supply. Similarly, the Jurassic, Permian and Carboniferous, while of secondary importance are still valuable resources. Generalisations concerning the yielding capacity of a particular sequence, let alone whole geological periods should be treated with extreme caution. Lias clays of the Jurassic and Keuper marls of the Triassic have almost zero yield and water bearing strata of any period can have distinctly variable yields depending upon changes in physical strata properties, lithology and the local hydrological cycle.

GEOLOGICAL ERA	GEOLOGICAL PERIOD	AGE MILLIONS OF YEARS	MAJOR OROGENIES	CAINOZOIC SUB-SYSTEMS	WATER BEARING FORMATIONS
CAINOZOIC	QUATERNARY		ALPINE OROGENY	HOLOCENE	
				PLEISTOCENE	Sands and Gravels
	TERTIARY			PLIOCENE	Crag deposits
				MIOCENE	
				OLIGOCENE	Headon Hill Sands
				EOCENE	Bagshot Sands, Oldhaven, Blackheath, Woolwich, Reading and Thanet Beds.
		70		PALEOCENE	
MESOZOIC	CRETACEOUS	136			Chalk, Upper and Lower Greensand Hastings Beds
	JURASSIC	195			Portland Beds, Corallian, Great and Inferior Oolites, Middle Lias
	TRIASSIC	225			Bunter and Keuper Sandstones Bunter pebble beds
UPPER PALEOZOIC	PERMIAN	280	HERCYNIAN OROGENY		Magnesium limestone Permian sandstones
	CARBONIFEROUS	345			Coal Measures sandstones Millstone grit Carboniferous Limestone series
	DEVONIAN	410			Old Red Sandstone series
LOWER PALEOZOIC	SILURIAN	440	CALEDONIAN OROGENY		
	ORDOVICIAN	530			
	CAMBRIAN	570			
PRECAMBRIAN	EOCAMBRIAN	700	PRECAMBRIAN OROGENIES		
	PROTEROZOIC				
	ARCHEAN	3500+			

Table 1.1 British Geological Periods and their Water Bearing Formations

(after Woodward (7) and Bennison (8))

It is therefore considered useful to identify potential aquifer horizons within various broad geographical locations of the British Isles, which can be associated with coal mining activity.

1.3.1 Scotland

Major aquifers are found in the Permian sandstone sequences around Dumfries, Lochmaben, Thornhill and Stranraer, but only in the area around Dumfries are they heavily exploited, where one well is recorded as yielding upto 150,000 litres per hour. Smaller and often localised supplies are obtained from sandstone sequences in the Old Red Sandstone (Devonian) and Carboniferous. Similar supplies are also obtained from superficial sands and gravels and near surface, well jointed greywacke and granite areas, Greig et al (9).

1.3.2 Northern England

In the Carboniferous, Table 1.2, some of the limestones (Visean), most of the thick grit (Namurian) and sandstone (Westphalian and Stephanian) sequences can yield supplies, but these are often unpredictable in location and quantity. In the Permian, both the Penrith and Yellow sands of Durham make excellent and reliable aquifers. Similarly, the Middle Magnesium limestone (Permian), particularly in Durham can yield large quantities of water, Taylor et al (10).

1.3.3 Pennines and Adjacent Areas

Good yields have been obtained from the coarser sandstone formations of the Carboniferous Millstone Grit and Coal Measures sequences (Table 1.2). However, these yields are again unpredictable in location and quantity. In the Triassic, the Bunter and Keuper sandstone are highly permeable and yield large quantities of water. In lowland areas, such as Cheshire and Nottinghamshire, these formations are used extensively for public supply. It is estimated that these aquifers yield 227 Mℓ/day (50 million gallons per day (50 mgpd)) in West Lancashire and Cheshire with a further 91 Mℓ/day (20 mgpd) in South Yorkshire and Nottinghamshire, Edwards and Trotter (11).

1.3.4 Central England

In the Midlands, Triassic pebble beds and sandstones of the Bunter and Keuper series provide extensive public water supplies. Much smaller, more localised supplies are also obtained from the limestone sequences along the Jurassic outcrop and alluvial sands and gravels, Haines and Horton (12).

1.3.5 Wealdon District

In the Cretaceous, the chalk sequences yield consistent quantities of good quality water. However, more prolific yields are obtained from the Lower Greensand and Hastings Beds. Similarly, small localised quantities can be derived from the Pleistocene and recent deposits in the area, Gallois (13).

Aquifer formations of the Wealdon are particularly important since beneath the area lies the concealed Kent Coalfield..

1.3.6 South Wales

Small localised supplies can be obtained from the sandstone sequences of the Old Red Sandstone, in the form of springs and wells.

Larger, but still localised yields are also obtained from the river gravels and drift deposits found in many areas. Further, large quantities are yielded from the Coal Measures sequences of the South Wales Coalfield. Estimates of 370 Mℓ/day (81.3 mgpd) (in 1956) have been obtained by Ineson (14), with 150 Mℓ/day (33.1 mgpd) drawn from the pre-Pennant sequences and a further 220 Mℓ/day (48.2 mgpd) from the Pennant. However, these values are estimated from the pumping records of both current and disused workings and should be treated with caution.

1.4 British Coal Measures

Location of the British Carboniferous Coal Measures in geological time is shown in Table 1.1. The Carboniferous period as shown in Table 1.2 can be divided into five stages which in turn can be sub-divided into three main sequences of limestones, grits and Coal Measures. However, it is only in the Westphalian and some isolated cases the Stephanian periods that economic coal seams occur. Details of the limestone (Tournaisian and Visean) and grit (Namurian) sequences are considered beyond the scope of this work and the reader is referred in the first instance to Bennison (8), for further details.

Historically, the Westphalian was better known as the Productive Coal Measures, which in turn was divided on a lithological basis into the Lower, Middle and Upper Coal Measures series. However, these divisions have subsequently been redefined in relation to palaeontological zonal sequences, Table 1.3.

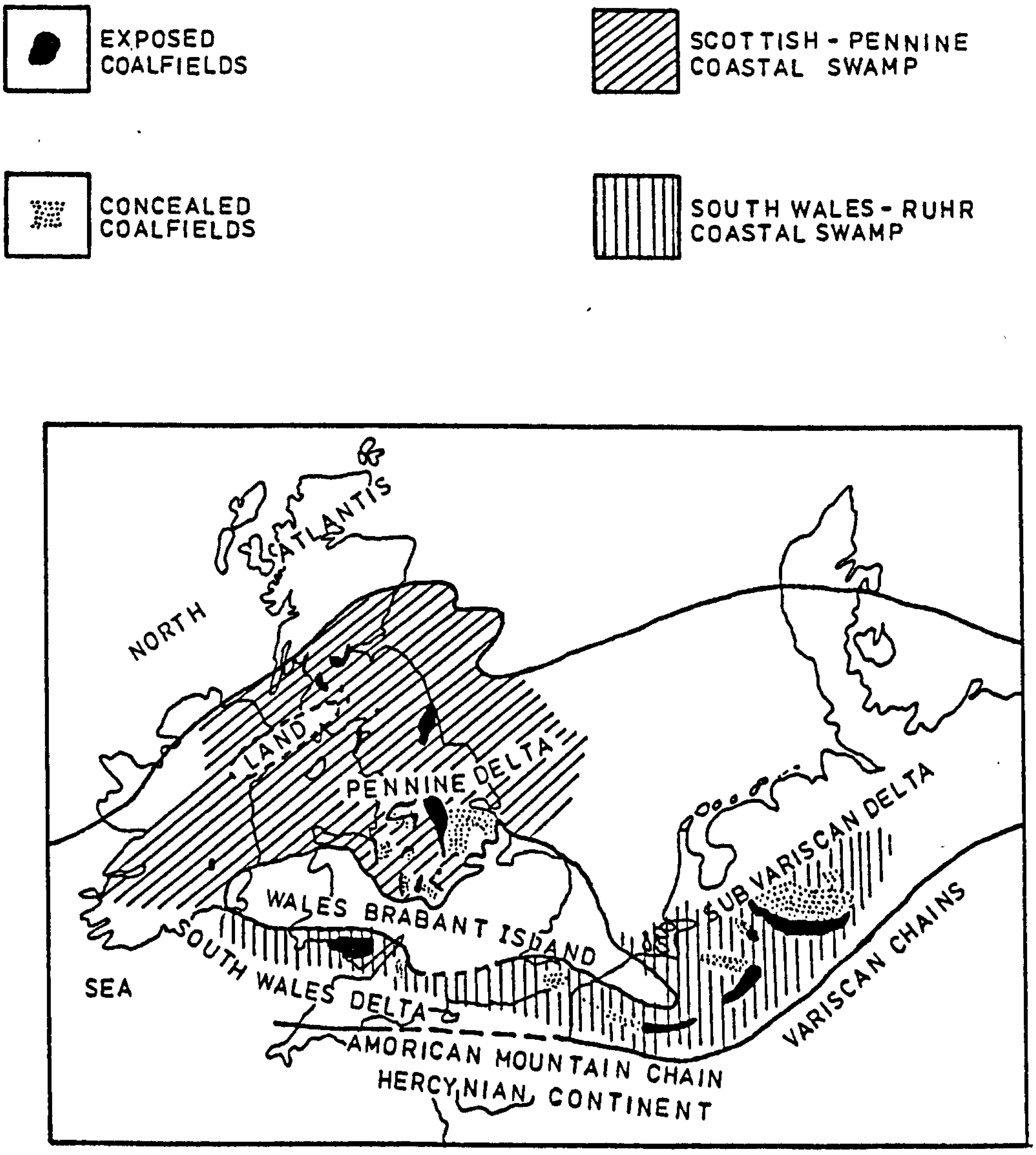


Figure 1.2 Palaeogeography of North-Western Europe during Westphalian Times
(after Bennison (8))

Europe	Lithological Divisions	Plant Divisions	Floral Zones	Goniatite Stages	Non-Marine Lamellibranch Zones	Chief Marine Bands
Stephanian		Radstockian	I		Prolifera	Morganian
Westphalian D	Upper Coal Measures	Staffordian	H		Tenuis	
			G		Phillipsi	Top M.B.
Westphalian C	Middle Coal Measures	Yorkian	F	Anthracoceras = A	Upper Similis-Pulchra	Mansfield M.B.
Westphalian B			E		Lower Similis-Pulchra	Ammanian
			D		Modiolaris	Claycross M.B.
Westphalian A	Lower Coal Measures	Lanarkian	C	Gastrioceras = B	Communis	
			B		Lenisulcata	
Namurian			A			

Table 1.3 Classification System for the Coal Measures - Palaeontological Zonal Sequences (After Bennison (8))

The base of the Carboniferous can be readily defined where marine Carboniferous sediments succeed land derived sediments of the Old Red Sandstone. Where marine Carboniferous succeeds marine Devonian the transition can only be determined by faunal changes. The top of the Carboniferous is usually defined by an unconformity, where deltaic sediments of the Coal Measures are overlain by red land derived sediments of the Permo-Trias.

Figure 1.2 shows the palaeogeography of North-West Europe during Westphalian times. The most important feature is the Wales-Brabant island, which is a remnant of land originally elevated during the Hercynian Orogeny. Surrounding this island was a vast deltaic area which was periodically inundated by shallow sea conditions. When subsidence of the surrounding area exceeded sedimentation, the formation of vast swamps has ultimately resulted in the coal seams of today. Many of the European Coalfields are contemporaneous with British fields surrounding the Wales-Brabant Island, Trueman (15), Bennison (8).

The Coal Measures are essentially cyclic sediments the complete cyclotherm of which is as follows:

Coal
Seatearth
Sandy Shale
Siltstone
Shale
Shale with non-marine lamellibranches
Shale with marine lamellibranches.

(after Bennison (8)).

Upper Carboniferous	Silesian	Stephanian	Coal Measures
		Westphalian	
		Namurian	Millstone Grit
Lower Carboniferous	Dinantian = Avonian	Visean	Carboniferous Limestone and Limestone Shale
		Tournaisian	

Table 1.2 Main Divisions of the Carboniferous
(after Bennison (8))

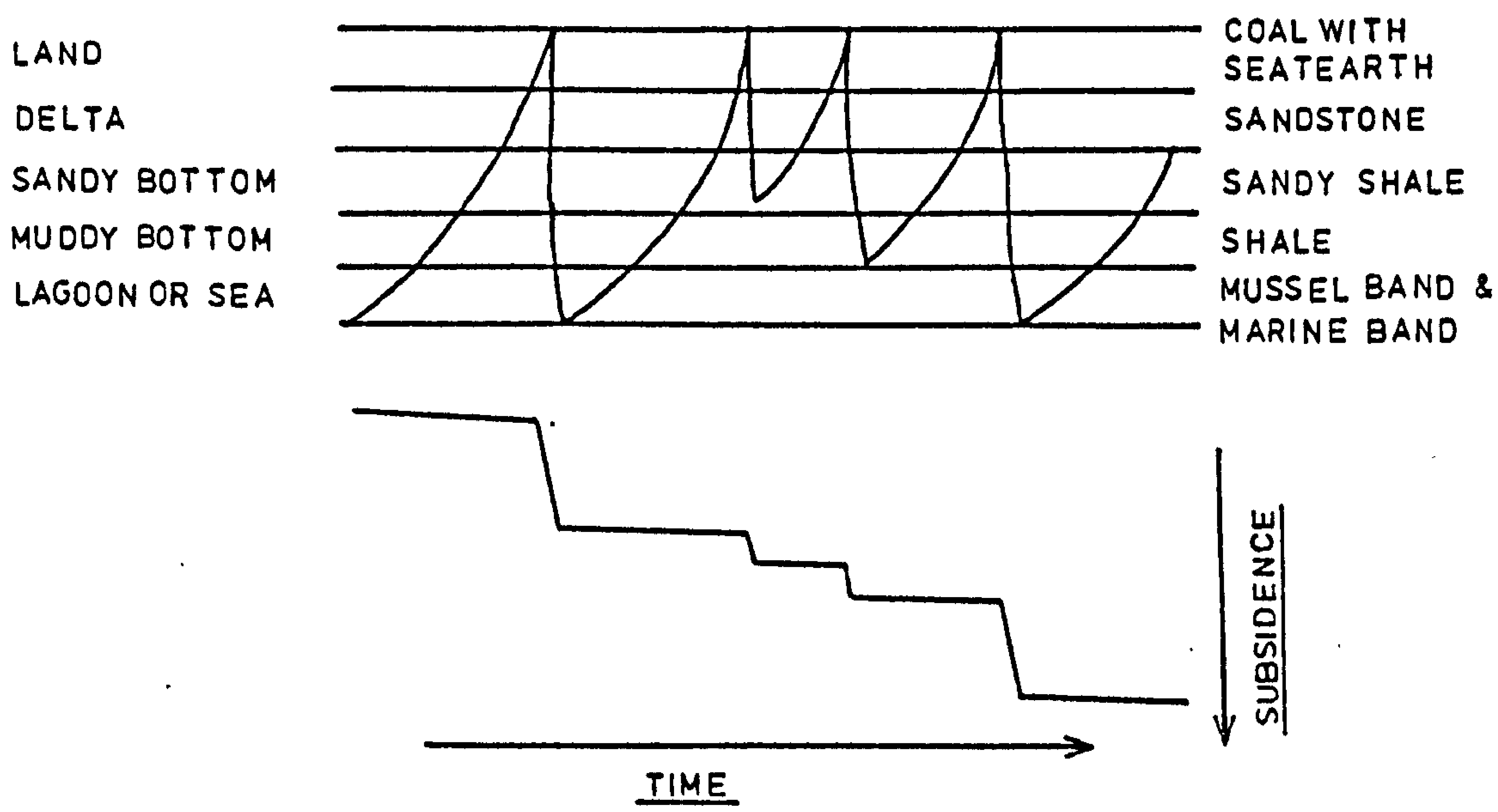


Figure 1.3 Formation of the Coal Measures-Cyclic Sedimentation
(after Bennison (8))

However, in the authors experience a slightly simpler cyclotherm classification can be used, which avoids the use of the rather vague term 'shale':

Coal
Seatearth
Mudstone
Siltstone
Sandstone

Additional terms such as carbonaceous mudstone, muddy siltstone and silty sandstone can also be included when considered academically.

Coal Measures cyclotherms are notoriously variable not only in sequence but also spatially around any given area. Figure 1.3 shows the concept of cyclic sedimentation in diagrammatic form and includes the presence of incomplete cycles of deposition. Zoning of the different Coal Measures divisions is achieved by examination of macro fossils, marine and non-marine lamellibranchs, while for coal seams microspore assemblages are used (Palynology).

Stephanian strata is generally devoid of economic coal seams and composed of interbedded breccias, conglomerates, marls and assorted assemblages of Coal Measures strata.

Hydrogeologically, Coal Measures sequences can be divided into four main groups:

1) Aquifer (intergranular permeability) - Sandstone, Siltstone
2) Aquitard - Siltstones, Mudstones
3) Aquiclude - Siltstones, Mudstones, Coal
4) Impermeable - Mudstones, Seatearth.

In each case, the location of any given strata type in a category is purely arbitrary, since aquifer properties are dependent primarily on the local strata lithology and hydrological conditions. Similarly, the existence of a well developed fracture/fissure network in any given strata type could result in a potential aquifer horizon exhibiting fracture rather than intergranular permeability.

Finally, it is worth noting that coal seams can be found in other geological periods such as the Devonian and Jurassic, but are generally less well preserved and uneconomical. A notable exception is the small Scottish coalfield at Brora, which is Jurassic in age, Macgregor (16).

1.5 Coalfield Formation and Structure

During late Carboniferous and early Permian times, folding of the accumulated sediments began due to a series of complex structural earth movements associated with the Variscan Orogeny in North-East Europe. The net result was formation of regional synclinal or basin structures with a general axial trend of north-south or east-west, in which the coalfields of today are found. It was only formation of these synclinal structures which has protected the Coal Measures from subsequent erosion and denudation. In addition, the pattern was also set for the determination of those areas now devoid of the Carboniferous completely (Central Mendips), the Coal Measures only (Pennines) or the Stephanian formations (as in numerous anticlines within the coal basins themselves).

It should be noted that this is a very simplified description of events and in reality a far more complex series of structural movements occurred which resulted in both regional and localised folding and faulting. A full description of the Variscan Orogeny is considered beyond the scope of this work and the interested reader is referred in the first instance to Bennison (8).

The structural environment of British Coalfield basins can be divided into one of three types:

1) Exposed.
2) Exposed but with a concealed extension.
3) Concealed.

1.5.1 Exposed Coalfields

These are areas of Coal Measures strata which have been protected from the processes of erosion and denudation by their synclinal structure. They do not possess an overlying cover of consolidated strata dating from a subsequent geological period. The Coal Measures are exposed to surface, but generally have a superficial covering of such recent deposits as sands, gravels, boulder clay or peat. Examples include South Wales, the Forest of Dean and most of the Scottish coalfields.

In each type of coalfield the extremities transgress either conformably or unconformably into strata of adjacent geological periods. However, in many cases one or more boundaries are formed where the Coal Measures have been thrown against either older or younger strata by faulting.

1.5.2 Exposed Coalfields with a Concealed Extension

These coalfields exhibit both an exposed outcrop of Coal Measures strata and a concealed extension which dips and extends beneath a cover of generally younger geological strata. The younger strata may transgress the concealed Coal Measures either conformably or unconformably. Examples include Nottinghamshire and South Yorkshire, Northumberland and Durham, Lancashire, North and South Staffordshire, Warwickshire, Leicestershire and Cumberland.

1.5.3 Concealed Coalfields

This type of coalfield is rare because most of the concealed coalfields can be directly associated with an extension of one of the exposed/concealed types already mentioned. However, one notable exception is the Kent Coalfield, where a thick sequence of Coal Measures is overlain unconformably by a thick series of Jurassic and Cretaceous strata. The coalfield exhibits a classic synclinal structure which protected the sediments from subsequent erosion and denudation.

1.6 Origin of Mine Water

The Coal Measures are poor to very poor aquifer systems with only the larger sandstone horizons providing any significant quantities of water. In general, when a mine experiences water problems it is not usually attributable to the Coal Measures, but some external source such as adjacent post-Carboniferous water bearing strata. The mechanism by which this water enters the mine is of particular importance to the mining engineer, especially in the design of dewatering schemes and the feasibility of new working areas.

1.6.1 Surface Recharge to Shallow Workings

Surface runoff due to rainfall can penetrate to shallow working horizons of less than 100 metres. The rate of percolation through consolidated Coal Measures strata is normally low, due to the presence of numerous semi-permeable horizons of mudstone and siltstone. However, when disruption of this stratà occurs due to working (development of the subsidence profile), formation of fissure networks and large cracks can result in more direct flow paths to the mining horizon. Net result, is an increased yield of water to the workings which has to be disposed of. A seasonal variation in the amount of water yielded is normally discernable between the summer and winter months.

1.6.2 Groundwater Recharge to Workings

When Coal Measures outcrop at surface, direct recharge of the strata may occur. If the strata or particular horizons are sufficiently permeable, water can be yielded to the workings from this source. A concealed outcrop of Coal Measures abutting heavily water bearing strata (Triassic sandstones) may similarly allow direct yielding of water to the workings.

Fossil water contained within more porous lithological units of the Coal Measures can also be liberated to a mining horizon, when disrupted by the workings. However, the quantity yielded will be finite unless recharge is occurring from an external source.

The presence of large faults or significant joint/fissure zones may allow the passage of water to workings from adjacent water bearing strata, even though the host strata is normally impermeable.

Investigations by Orchard (17) in the South Derbyshire/ Leicestershire coalfield concluded that a zone of increased permeability could extend to a height of 60 m above a conventional longwall face. The study analysed over 50 cases in which Coal Measures strata was overlain by Triassic sandstones, but only in half was water encountered in any form.

1.6.3 Shafts

Many shafts have been sunk through heavily water bearing strata, most notable of which is the Permian of North-East England and the Trias of Nottinghamshire and South Yorkshire. This has resulted in the use of water tight materials for shaft linings. However, seepage usually occurs along either the joints or from behind the lining and requires provision of collection and disposal facilities at the shaft bottom. In certain cases, the provision of surface boreholes is also required in order to create an artificially dewatered zone around the shaft.

A particularly novel and interesting shaft dewatering scheme is quoted by Levitt (18),

'At Manton Mine in Nottinghamshire the No. 1 Shaft was sunk to a depth of 108 m and connected at the 82 m and 108 m levels to the deeper No. 2 and No. 3 Shafts. Throughout the life of the mine, the Bunter and Permian water from around the No. 2 and 3 shafts is drained to No. 1 shaft and pumped to surface. About 17 million litres per day are pumped from No. 1 shaft, of which two thirds is used by the local Water Authority for public supply'.

1.6.4 Disused Mine Workings

When a mine is operating, there are usually no drainage problems in adjacent mine workings. However, if the mine is abandoned, flooding usually ensues and unless pumping is continued serious problems can occur in adjacent mining horizons. Provision of adequate coal barrier pillars both sequentially and laterally between workings, prevents the flow of both water and gas which might otherwise accumulate in the disused workings. The pumping of old mines is widely undertaken in the Scottish, South Wales and North-East coal-fields in order to protect current workings. A good example is found in South Durham, where pumping at the disused Castle Eden and Blackhall Collieries is continued in order to protect Horden Colliery.

Estimation of the water quantity stored in old workings is usually complicated by an inadequate knowledge of the area strata stability. Old goafs can store large quantities of water where incomplete consolidation has resulted in a marked porosity and permeability increase of the caved material. Rogoz (19) gives several methods for estimating the quantity of stored water for a variety of mining methods. In areas where ancient but unrecorded workings are suspected, routine exploratory drilling is undertaken in advance of the face. If water is encountered the face is either abandoned or a comprehensive dewatering scheme undertaken before further face advance.

1.6.5 Geological-Structural Features

Structural features associated with the geological environment of a mine can act as mechanisms for the access of water to working horizons. Faults which intersect the workings and also penetrate water bearing horizons can act as pathways along which flow can occur. Similarly, if faulting has brought a heavily water bearing horizon into close proximity with a mining horizon, then water can be encountered. A good example of this is found at Westoe Colliery, NCB North East Area, where a drivage from the Main seam across the Ninety Fathom fault encountered a heavily water bearing sandstone formation. Cement injection sealing techniques were used along this section of roadway in order to stem the flow and permit further advance.

Dyke structures which penetrate water bearing horizons can also act as pathways for the flow of water to working areas. At Westoe Colliery, a series of late Carboniferous dykes intersect the workings and penetrate to the base of Permian. Composition of the dykes is extremely variable and in many cases when intersected, have been found to be dry. However, in several cases, intersection has resulted in large quantities of water being yielded for a considerable length of time. A dyke intercepted by a drivage in the region of Main seam face F11 produced a maximum yield of 4.5 m^3/min (1000 gpm) which eventually decreased to zero. Exploratory drilling should therefore be undertaken in all areas where structural features can be associated with water occurrences.

1.6.6 Operational Water

In daily underground mining operations water is used for dust suppression and fire-fighting purposes. Dust suppression water is used on the coalface and at strategic points on the coal conveyor system. Levitt (18), gives a value of between 100 and 160 litres of water used for every tonne of coal produced in Great Britain. Fire-fighting water remains in a closed circuit pipe range until required.

The bulk of dust suppression water is removed from the coal face or transport system by absorption onto the coal, dust or other waste materials or as atmospheric moisture in the ventilation system. However, small quantities particularly in the face region can remain and be included in the total quantities pumped from the mine.

1.7 Dewatering in British Coalfields

Table 1.4 shows daily quantities of water pumped by each of the 12 NCB Areas during 1977 and 1980. Levitt (18) states an average figure of 2.5 tonnes of water pumped for each tonne of saleable output coal.

The total volume of water pumped by the 12 areas has increased by 24%, but only in 3 are actual volumes seen to decrease. This suggests that the overall quantities of pumped mine water are increasing for a variety of reasons, which are usually geological. Similarly, as more mines are closed, the number at which pumping has to continue increases, in order to prevent inundation of adjacent working areas.

NCB Area	Volume in Mℓ per day 1977	1980
Scottish	140	146
North East	210	305
North Yorkshire	17	13
Barnsley	60	98
South Yorkshire	44	57
Doncaster	7	11
North Derbyshire	15	20
North Nottinghamshire	3	3
South Nottinghamshire	7	34
South Midlands	37	18
Western	36	39
South Wales	213	231
Total	789	977

Table 1.4 Mine Water Abstractions from NCB Areas (after Levitt (18)).

Over 70% of the total volume pumped can be associated with the Scottish, South Wales and North-East areas. All three have a large exposed outcrop of Coal Measures strata which allows direct and groundwater recharge to current and abandoned workings. In addition, the North-East also has a large concealed extension which is overlain by a thick sequence of Permian aquifer systems. A long historical record of coal mining activity is found in each area, which starts on or near the outcrop and becomes progressively deeper. Consequently, pumping has to be maintained in disused shallow and medium depth workings to achieve safe conditions in the current deeper mines.

A further 20% of the total pumped volume is associated with the Barnsley, South Yorkshire and Western Areas. Each exhibits a historical record of coal mining activity and smaller exposed Coal Measures outcrops with concealed extensions dipping beneath water bearing formations of the Permo-Trias.

A trend is therefore discernable between the total volume of water pumped and three parameters applicable to that area:

1) Area of exposed coalfield outcrop
2) Length of historical mining activity
3) Presence of water bearing strata over a concealed extension.

Table 1.5 shows this trend expressed in terms of these loosely defined parameters. The applicability and magnitude of each parameter value with respect to the 12 areas is purely arbitrary and based on the authors judgement. Each is obviously open to discussion,

	Type of Field Area		Historical Working		Concealed Extension	Presence of Water Bearing Formations	Litres/d x 10^6 1980 Area Volume Mℓ/day
			pre 1800	post 1800			
North East	Exposed	Large	X	X	Yes	Yes	305
S. Wales	Exposed	Large	X	X	-	-	231
Scottish	Exposed	Large	X	X	-	-	146
Barnsley	Exposed	Moderate	X	X	-	-	98
S. Yorks.	Exposed	Small	-	X	Yes	Yes	57
Western	Exposed	Moderate	X	X	Yes	Yes	39
S. Notts.	-	-	-	X	Yes	Yes	34
N. Derbys.	Exposed	Moderate	X	X	-	-	20
S. Midlands	Exposed	Small	X	X	Yes	Yes	18
N. Yorks.	Exposed	Small	X	X	Yes	Yes	13
Doncaster	-	-	-	X	Yes	Yes	11
N. Notts.	-	-	-	X	Yes	Yes	3
							977

Table 1.5 Classification of Minewater Abstraction against Type of Coalfield

contention and redefinition, but is intended merely as a guide to illustrate a trend and not establish a 'hard and fast' rule.

Finally, Table 1.5 reveals that at one extreme a large exposed Coal Measures outcrop extensively worked at all depths and with a concealed extension overlain by water bearing strata requires considerable Area deployment of resources for mine pumping and dewatering. At the other extreme, a concealed extension at depth, overlain by water bearing strata and with recent mining activity need only consider dewatering problems at an individual mine level.

1.8 Minewater Disposal

A breakdown of the total volume of mine water disposed, reveals that 88% is discharged either to local water courses or in the case of coastal collieries directly to the sea. A further 5% is used for public supply and the remaining 7%, internally by the Board. However, increasing attention is now being focused on the use of this water for colliery operational purposes, subject to certain quality criteria, Levitt (18).

Table 1.6 shows the various sources of water drawn for colliery operations, while Table 1.7 shows the various functions for which it is used.

In addition to water abstracted from underground operations, two further important sources are also yielded from a colliery. Process water associated with washing operations in the coal preparation plant and surface runoff from the pit head complex and coal stocking sites.

Source	Percentage
Mines Drainage	29
Licensed Abstractions	51
Public Supply	20
	Total 100

Table 1.6 Sources of Water Drawn for Colliery Operations (after Levitt (18)).

Use	Percentage
Pithead baths, drinking, ablutions	14
Steam raising and cooling	10
Coal preparation	41
Firefighting and dust suppression	27
Other	8
	Total 100

Table 1.7 Water Usage in Colliery Operations (after Levitt (18)).

The direct discharge of all three types of water, if not rigorously controlled can have serious environmental consequences on the surrounding countryside and in particular local water courses, Headworth et al (20). Treatment of these various waters may be necessary to remove suspended matter or potentially toxic substances before discharge.

Finally, although disposal and impact of mine water discharges on the environment is beyond the scope of this work, the author suggests further reading of work by Glover (21), Woodward (22) and Edwards (23).

1.9 Conclusions

British Coal Measures sequences contain only low yielding aquifers when compared with major aquifer systems in the Triassic and Cretaceous periods. A close geological and hydraulic relationship exists between the Coal Measures and these other strata, which allows the passage of water to occur via a series of interrelated mechanisms. When Coal Measures are exposed at surface, a downward passage of water still occurs which can eventually find its way to a mining horizon.

The potential flow of water to a mining horizon can be identified with six main sources:

1) Surface recharge to shallow workings.
2) Groundwater recharge to workings.
3) Shafts.
4) Disused mine workings.
5) Geological - structural features.
6) Operational water.

All of these should be considered in proposals for a new working area or before initiation of a dewatering scheme.

It is currently estimated, Levitt (18), that 2.5 tonnes of water are pumped for each tonne of saleable output of coal. However, consideration of future trends suggests that as more mines are closed, particularly in the exposed coalfields with a long history of working, increased quantities must be pumped in order to maintain satisfactory levels of safety in adjoining collieries. The quantity of water pumped for each tonne of saleable output will therefore increase along with further pumping costs.

In addition, as more water is pumped to the surface more will be discharged to the environment. Increased environmental concern over the affects of industrial discharges will therefore require larger and more sophisticated treatment plants which again will result in increased costs. Maximum use of the pumped water should therefore be made before its eventual discharge, such as its use in coal preparation, pit head uses, public and/or agricultural supplies, Gisman and Szczypa (24).

CHAPTER 2

GROUNDWATER FLOW AND EVALUATION IN PERMEABLE AND SEMI-PERMEABLE AQUIFER SYSTEMS

CHAPTER 2

GROUNDWATER FLOW AND EVALUATION IN PERMEABLE SEMI-PERMEABLE AQUIFER SYSTEMS

2.1 Introduction

The importance of groundwater as both a local and national resource has long been recognised by many countries. But the complex relationship between water occurrences experienced in mine workings and these aquifer resources has seldom been appreciated. Similarly, extensive research programmes have been initiated to understand the mechanics of groundwater movement and well flow systems, although very little has been undertaken on the flow characteristics around mine openings and extraction horizons. Walton (25) gives an excellent though brief account of the historical development of groundwater resource evaluation and suggests that the first recorded mention dates from 1000 BC in Book 21 of Homers Iliad. It is only in the past 5 years (26, 27, 28) that a concerted international effort has been made to begin solving the problems associated with mining activity and mine water occurrences. However, when considering both ground and mine water problems, the equations developed for assessing flow have all been originally derived from work undertaken by Darcy (29) during the 1850's and it is upon this foundation that all subsequent work has been derived. It is therefore intended to examine the Darcy equation along with its limitations and develop it for the determination of formation parameters in both intergranular and fracture/fissure aquifer systems. Consideration is also given to the application of the various techniques in Coal Measures strata.

2.2 Darcy's Law

In its simplest form Darcy's Law can be stated, Walton (25),

'the flow rate through a porous media is proportional to the head loss and inversely proportional to the length of flow path'.

This can be expressed by equation 2.1,

$$Q = \frac{-KA(H_2 - H_1)}{L} \qquad \text{..........} \quad 2.1$$

where

Q - Flow rate through the system.

A - Cross-sectional area.

H_1 and H_2 - Head loss across the system.

L - Distance between points H_1 and H_2, at which head is measured.

K - Hydraulic conductivity of material.

The minus sign indicates that flow is in the opposite direction to increasing H.

Bouwer (6), restates the Darcy equation as

$$v = \frac{K\ (h_1 + z_1) - (h_2 + z_2)}{L} \qquad \text{..........} \quad 2.2$$

where

v - Darcy velocity of water.

h_1 and h_2 - Pressure head at points 1 and 2.

z_1 and z_2 - Elevation head at points 1 and 2.

L - Distance of flow between points 1 and 2, measured along streamline.

K - Hydraulic conductivity of material.

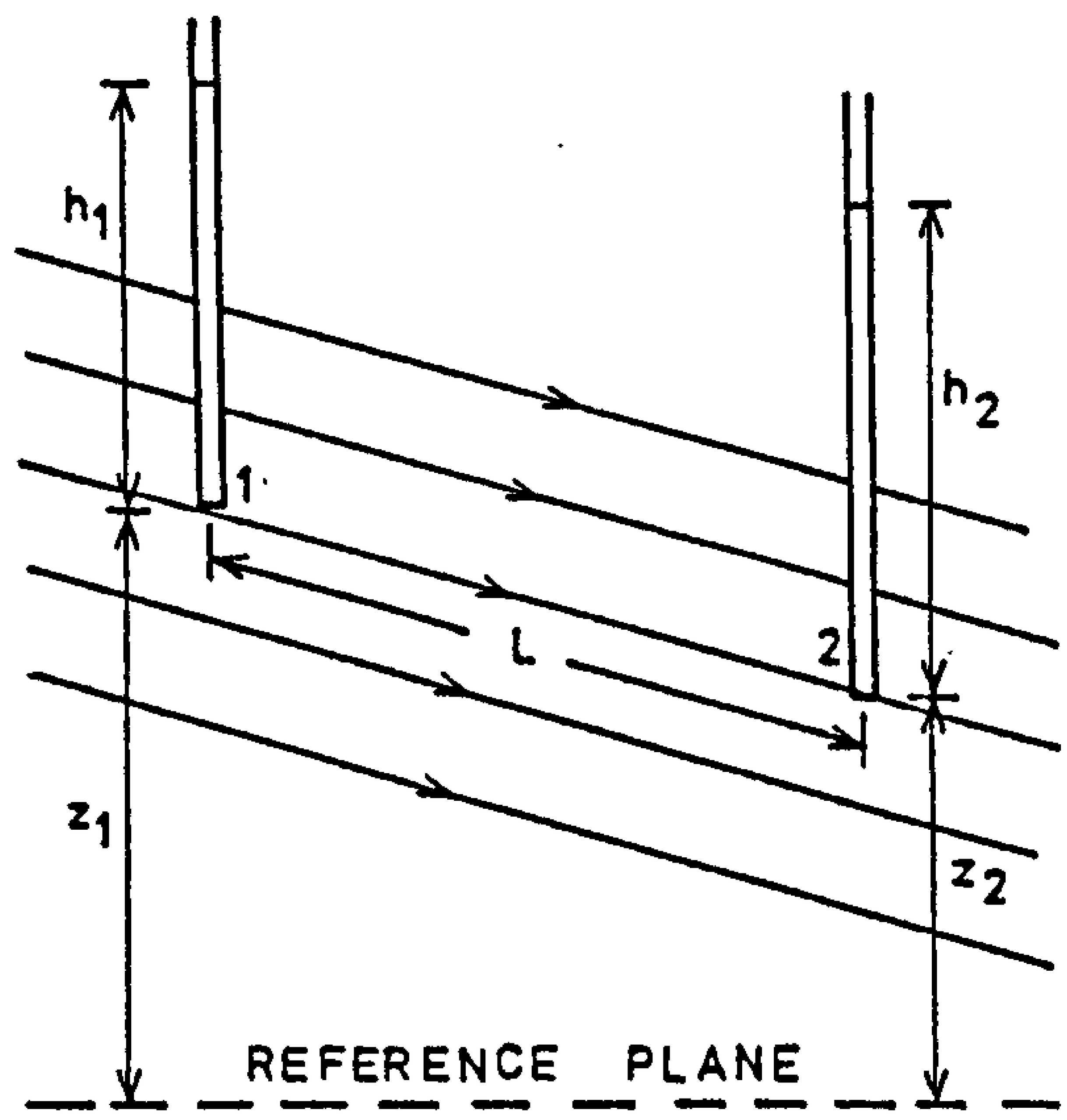


Figure 2.1 Vertical Cross-Section Showing Groundwater Flow with Linear, Parallel Streamlines, Developing Equation 2.2
(after Bouwer (6))

where Equation 2.2 is

$$v = \frac{K(h_1 + z_1) - (h_2 + z_2)}{L} \qquad \text{..........} \quad 2.2$$

(see Section 2.2)

Equations 2.1 and 2.2 are both identical and developed from the same line of reasoning.

Figure 2.1 shows in schematic form the conditions developed by equation 2.2. The flow of water is assumed to occur along smooth paths which cross both pore spaces and intervening solid particles of homogeneous material, instead of passing around them via the pores and micro-fissure network. The resultant smooth lines of travel are known as stream lines, Bouwer (6). The stream lines in Figure 2.1 cross a vertical section of aquifer parallel to the direction of flow. Therefore provided the flow neither changes with distance nor time, the groundwater velocity between points 1 and 2 can be calculated using equation 2.2.

Dimensional analysis reveals that the units of K (hydraulic conductivity or coefficient of permeability) are LT^{-1}. In mining hydrology K is normally expressed as either metres/day or centimetres/second.

Darcy's Law can also be expressed in terms of pressure (P) and density (ρ) of the liquid, Scheidegger (30),

$$Q = -KA\ (P_2 - P_1)/(\rho gh) + 1 \qquad \text{..........} \quad 2.3$$

If a new constant is introduced K', assuming ρ and g to be constants, Scheideggar (30),

$$Q = \frac{-K'A(P_2 - P_1 + \rho gh)}{h} \qquad \text{..........} \quad 2.4$$

where

P_1 and P_2 - Pressure at points 1 and 2.

ρ - Density of liquid.

g - Acceleration due to gravity.

h - Difference in elevation head at points 1 and 2.

In equation 2.2, the volume rate of flow through any given cross-sectional area perpendicular to the flow direction can be calculated by, Bouwer (6),

$$Q = vA \qquad \text{..........} \quad 2.5$$

where

Q - Volume rate of flow

v - Darcy velocity

A - Area normal to flow direction.

Therefore, if flow is assumed to occur through a bundle of straight capillary tubes in the streamline direction, the actual water velocity will be greater than the Darcy velocity, Bouwer (6),

$$V_m = \frac{A}{A_{cap}} v \qquad \text{..........} \quad 2.6$$

where

V_m - Actual velocity

A - Total cross-sectional area

A_{cap} - Sum of the capillary tube cross-sectional areas

v - Darcy velocity.

However, the ratio A_{cap}/A is also known as porosity and equation 2.6 can be re-written, Bouwer (6),

$$V_m = \frac{v}{n} \quad \text{.......... } 2.7$$

where

n - Porosity.

Although equation 2.7 is only correct for straight capillary tube models, it has been found under certain circumstances, Bouwer (6), to give acceptable estimates of actual water velocity in aquifer materials.

2.3 Validity of Darcy's Law

2.3.1 Laminar Flow

Darcy's Law is only valid for laminar flow conditions, where the flow velocity is both slow and small and the water travels in smooth paths more or less parallel to the solid pore boundaries. In capillary tubes with uniform diameter, water moves exactly parallel to the tube walls. Laminar flow is therefore governed by the viscous forces of the fluid and head losses will vary linearly with velocity as in the Darcy equations 2.1 and 2.2. Although most strata types are assumed to exhibit laminar flow for evaluation purposes, this will rarely occur either in practice or under field conditions. However, of those sequences which can exhibit near Darcian conditions, the thick Permo-Triassic sandstones are probably the best example.

2.3.2 Turbulent Flow

An increase in water velocity will result in a situation where inertial forces within the water become significant and turbulent flow conditions predominate. Under these circumstances, water begins to travel in irregular paths and the head losses vary exponentially with fluid velocity. Turbulent flow conditions often develop in the immediate vicinity of pumped wells (due either to the well pack system or formation development) or in very porous strata (such as certain limestones or well fissured formations).

2.3.3 Pipe Flow

In certain conditions, such as subterranean channels in cavaneous limestones, the flow can no longer be considered turbulent, but must be treated as open channel or pipe.

When considering channel or pipe conditions, the type of flow developed is characterised by the Reynolds number N_R. This is dimensionless and expresses a ratio between inertia and viscous forces within the fluid. It can be expressed, Bouwer (6),

$$N_R = \frac{\rho v D}{\mu} \qquad \text{..........} \quad 2.8$$

where

v - Fluid velocity

D - Dimension of conduit

ρ - Fluid density

μ - Fluid viscosity.

The ratio μ/ρ can be expressed as a single parameter known as the kinematic velocity γ (dimension LT^{-2}).

Pipe flow is laminar when $N_R < 2000$, but if $N_R > 2000$ a transition to full turbulent flow occurs. When considering groundwater movement, laminar flow occurs when $N_R < 50$, but increasingly turbulent conditions exist when $N_R > 50$.

2.3.4 Clay Materials

Darcy's law is not valid when considering the flow of water through dense clay materials. The pore spaces of such materials are often so small that the water molecules become influenced by 'double layer' effects due to the clay particles, Bouwer (6). The presence of polarised water molecules in the vicinity of electrically charged clay particles, results in the water becoming more 'ice-like' or crystalline in structure, which in turn increases its viscosity.

Similarly hydraulic gradients may be insufficient to produce water movement, which results in threshold gradients and a non-linearity between the flow rate and hydraulic gradient. Additional factors such as the rearrangement of clay particles due to frictional drag of the water or electro-osmotic counterflow can also result in non-Darcian flow, Bouwer (6).

2.3.5 Non-Darcian Conditions

Except in the vicinity of pumped wells, non-Darcian behaviour is seldom if ever considered in mine water evaluation, for two main reasons:

1) Accurate solutions of underground flow systems can never be achieved because of the heteromorphic nature of aquifers. In particular, a lack of information is usually apparent concerning their geometry and boundary conditions.

2) Analysis of non-Darcian flow is extremely complicated, involving the use of hydraulic conductivity values dependent upon Reynolds numbers. It can therefore be appreciated, the complexity

involved in determining and assigning Reynold's numbers to an anisotropic aquifer medium.

2.4 Factors Effecting Hydraulic Conductivity

2.4.1 Temperature

Temperature effects hydraulic conductivity, because it affects the viscosity of water. The relationship between viscosity and hydraulic conductivity is linear. A 50% reduction in viscosity doubles the value of hydraulic conductivity.

Hydraulic conductivity (K) is normally expressed at a temperature of 20°C, but can be calculated for other temperatures using equation 2.9, Bouwer (6),

$$K_t = \frac{\mu_{20}}{\mu_t} K_{20} \qquad \text{..........} \quad 2.9$$

where

K_t - Hydraulic conductivity at temperature t^oC

K_{20} - Hydraulic conductivity at 20^oC

μ_{20} - Absolute viscosity of water at 20^oC

μ_t - Absolute viscosity of water at t^oC.

2.4.2 Ionic Composition of Water

The ionic composition of groundwater can affect hydraulic conductivity, particularly if the aquifer material contains a clay component. An imbalance of ions in either medium can result in either flocculation or dispersion, which in turn increases or decreases the hydraulic conductivity of the material. This factor is particularly important when considering artificial recharge to existing production aquifers.

2.4.3 Entrapped Air or Gas

If an aquifer is subjected to an influx of foreign water at a lower temperature than the host water, an increase in temperature of the foreign water can result in previously dissolved gases being given off. These gases can cause physical blockages within the aquifer medium and result in decreased hydraulic conductivity values.

2.4.4 Bacterial Growth

Mixing of foreign and host aquifer waters, particularly during artificial recharge can result in a variety of bacterial and algal growths, which not only occur on the well screens but also within the surrounding aquifer medium. The presence of these growths results in physical blockage of the pore spaces, restriction of water flow and a decrease in the hydraulic conductivity of the aquifer. Loofbourow (31) mentions that this method has not yet undergone field trials to stem inflows of water to mining horizons.

2.5 Formation Parameters

2.5.1 Intrinsic Permeability

It can be shown, Bouwer (6), that the product of K_{μ} in equation 2.9 is a constant. This is the basis of intrinsic permeability, where the permeability of a porous medium is expressed as a parameter dependent on the medium, rather than the fluid density and viscosity.

Intrinsic permeability can therefore be defined by equation 2.10, Bouwer (6),

$$k = \frac{K\mu}{\rho g} \qquad \text{..........} \quad 2.10$$

where

k - Intrinsic Permeability

K - Hydraulic Conductivity

μ - Absolute viscosity of the fluid

ρ - Fluid density

g - Acceleration due to gravity.

In CGS units, k is expressed in cm^2 and therefore has dimensions of L^2. Units of k are the Darcy, which although seldom used in ground or mine water hydrology, is often used in petroleum and gas engineering where the reservoir fluids vary in both density and viscosity and contain both gaseous and liquid phases.

2.5.2 Transmissivity

When considering one dimensional flow in aquifer systems, equation 2.2 can be modified, Bouwer (6), to

$$Q = WDKi \qquad \text{..........} \quad 2.11$$

where

Q - Flow rate

K - Hydraulic Conductivity

i - Hydraulic gradient

W - Aquifer width

D - Aquifer thickness.

However, the product DK in equation 2.11 can be combined into a single parameter known as transmissivity, which is a function of the transmissive properties of the aquifer. Equation 2.11 can therefore be written:

$$Q = WTi \qquad \text{..........} \quad 2.12$$

Dimensionally T has units of M^2T^{-1}.

2.5.3 Storage Coefficient

The storage coefficient of an aquifer can be defined as, Kruseman and De Ridder (32),

'the volume of water released or stored per unit surface area of the aquifer per unit change in the component of head normal to that surface'.

It is also known as the specific yield of an aquifer and is dimensionless.

In unconfined aquifers, the yield occurs due to drainage of pore spaces, where water is replaced by air from the atmosphere as the water table drops. However, in confined aquifers, water is not yielded by pore space drainage because there is no falling water table and the aquifer material remains saturated. Three mechanisms exist by which water can be yielded from confined aquifers, Bouwer (6):

1) Consolidation or compression of the aquifer (particularly interbedded clay or silt bands) and confining strata due to lowering of the piezometric surface,

2) Leakage from other aquifers (an overlying unconfined aquifer separated by an aquitard),

3) Drainage of pore space, if the confined aquifer becomes unconfined with a free water table at outcrop.

2.5.4 Formation Anisotropy

The schematic diagram of a two dimensional isotropic aquifer, where hydraulic conductivity is the same in both directions (K_y = K_x) is shown in Figure 2.2.1. Isotropic conditions are seldom, if ever, encountered either in practice or under field conditions. However, some massive Permo-Triassic sandstones can exhibit localised isotropy of between 50 and 90%.

Two dimensional anisotropic aquifers must be considered on two conceptual levels:

1) Random orientation of particles of consistent dimension, Figure 2.2.2.,

2) Individual layers of isotropic material, which when stratified form an anisotropic medium, Figure 2.2.3.

Three dimensional isotropic and anisotropic aquifers also exist, which follow the same models as outlined above, Figures 2.2.4, 2.2.5 and 2.2.6.

All aquifer systems and geological strata types exhibit anisotropy on either the macro or micro scale. Similarly, in addition to lithological anisotropy, all strata types exhibit structural anisotropy in the form of bedding planes, faults, joint systems and fissure networks.

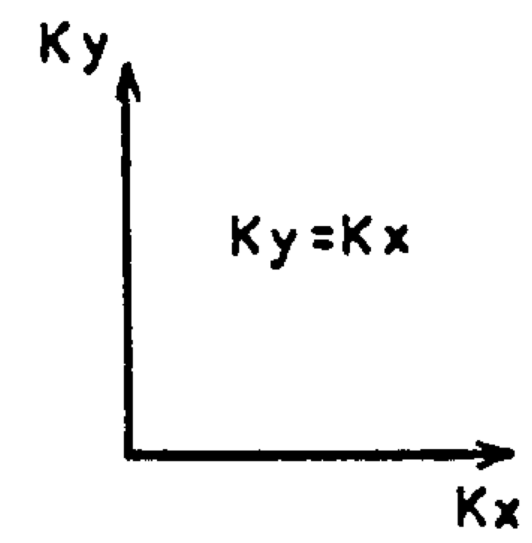


Figure 2.2.1

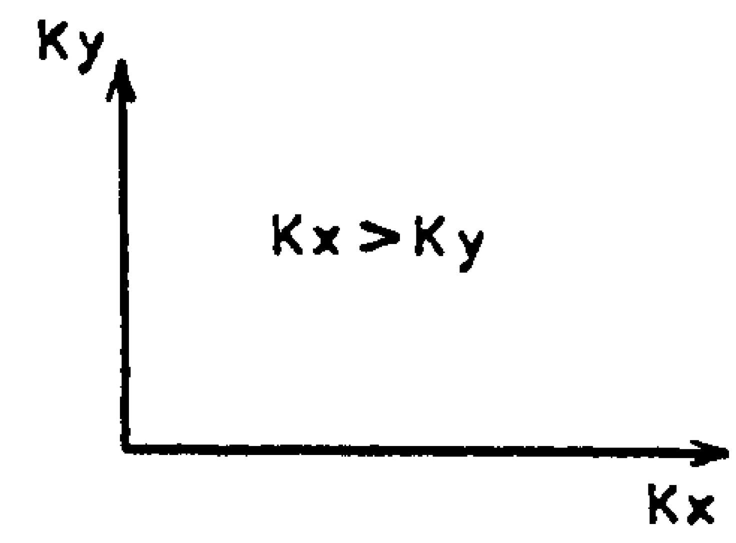


Figure 2.2.2

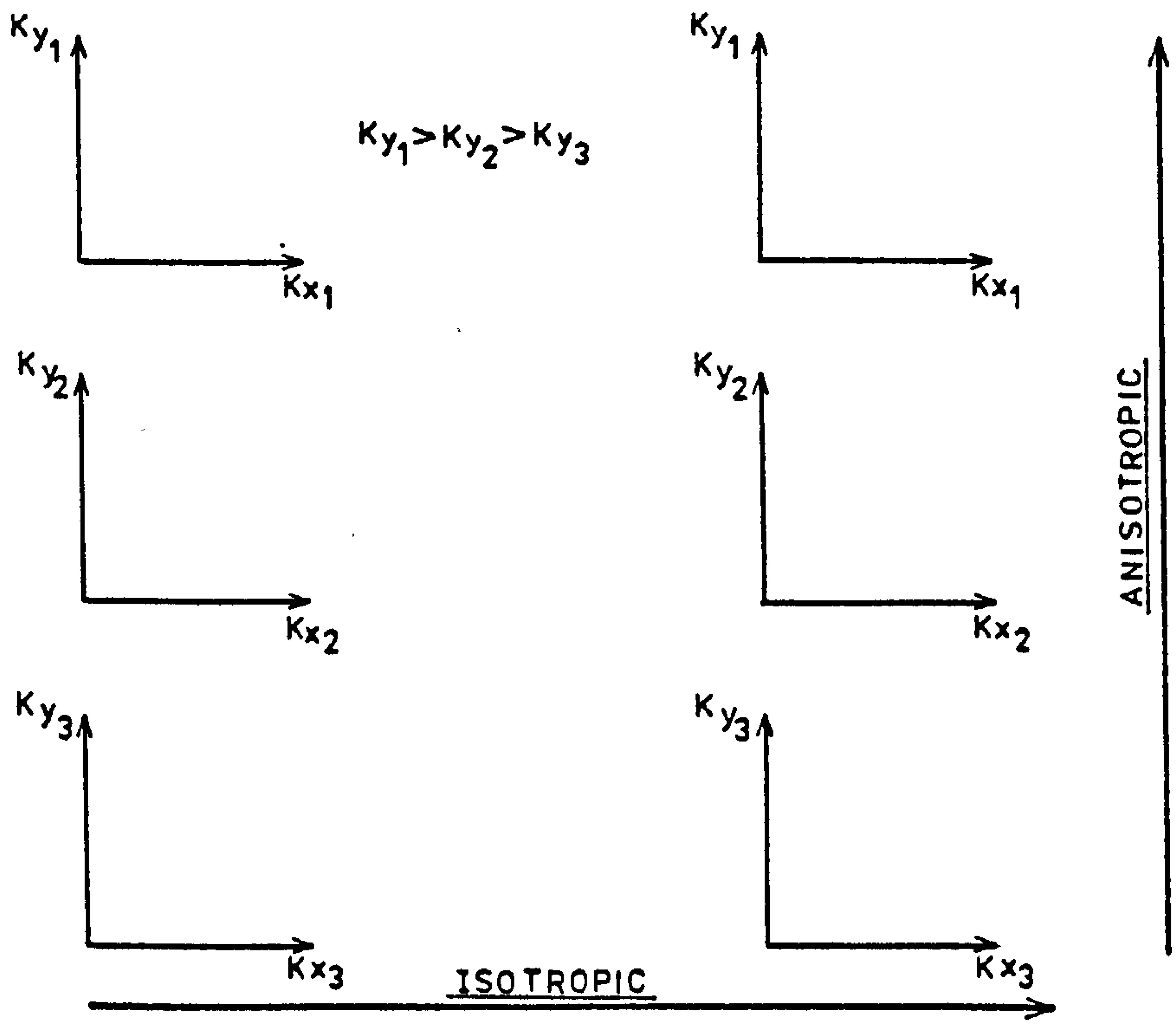


Figure 2.2.3

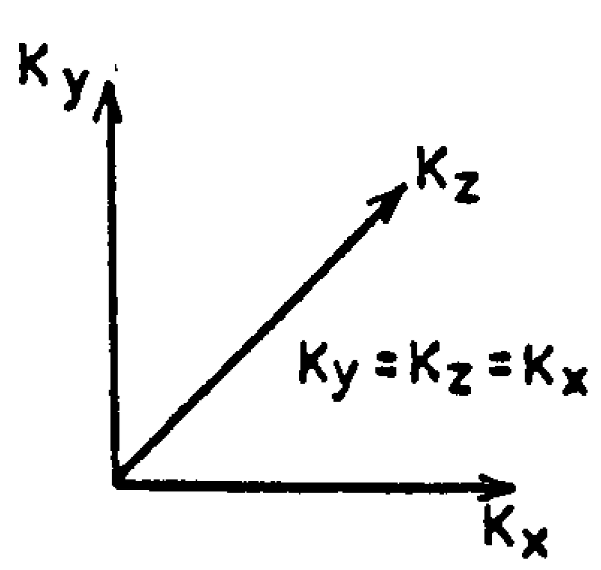


Figure 2.2.4

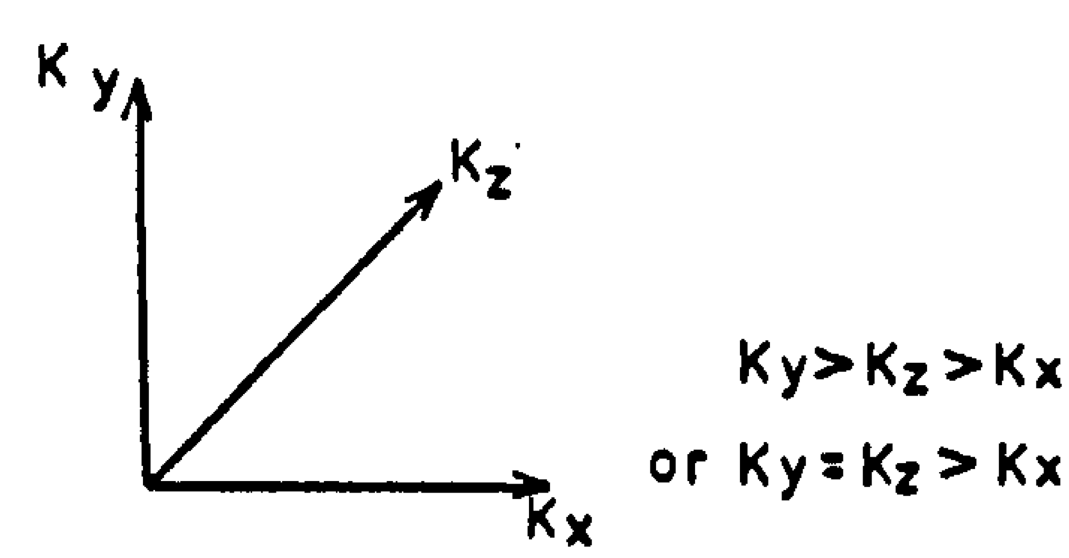


Figure 2.2.5

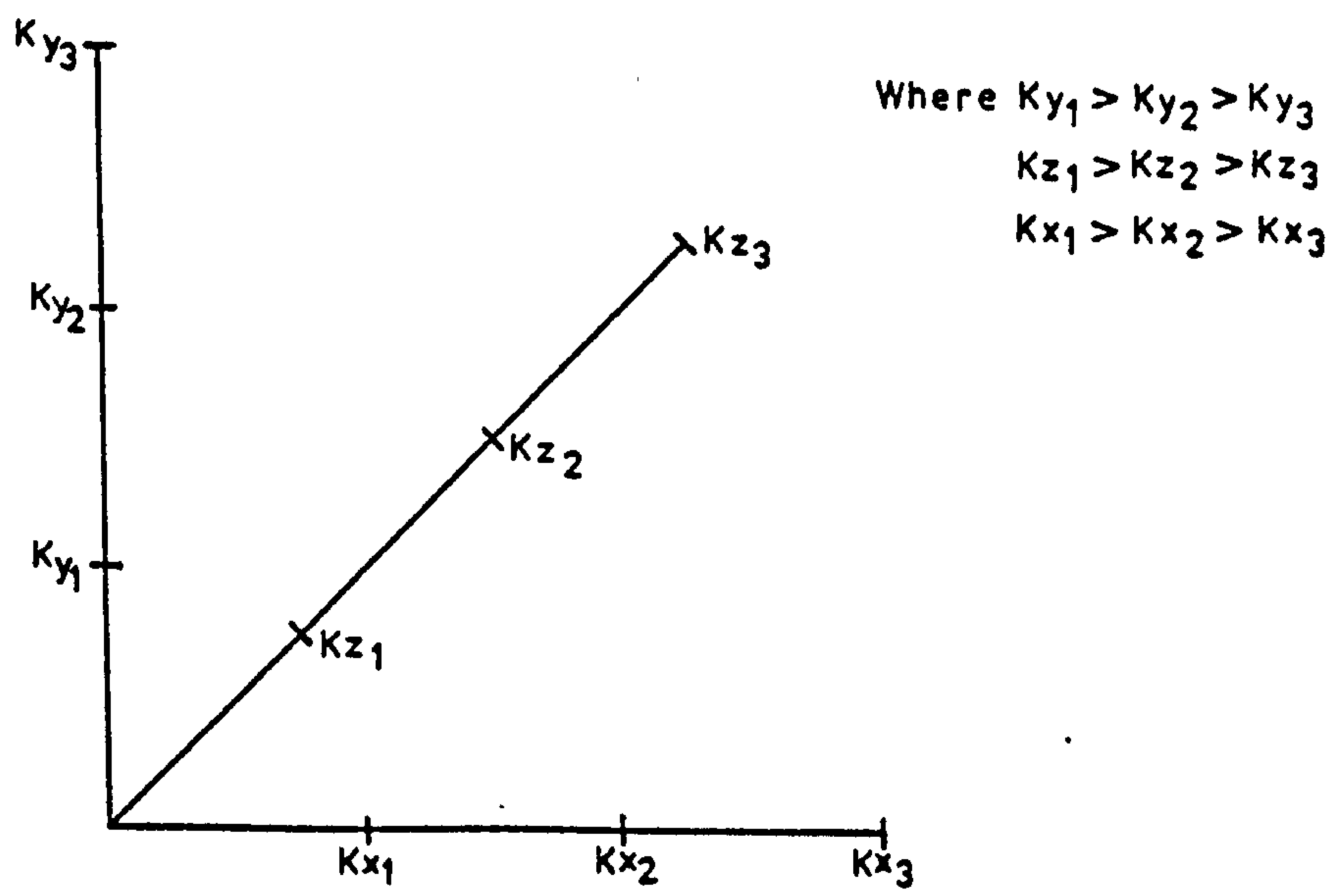


Figure 2.2.6

<u>Figure 2.2</u> <u>Concepts of Formation Anisotropy</u>

2.6 Physical Aspects of Permeability

2.6.1 Simple Models

In development of the Darcy equations 2.1 and 2.2, it was necessary to assume that flow occurred along streamlines, which passed irrespectively through both pore spaces and particles. Similarly, in equation 2.5 it was shown that provided the flow through a porous medium occurred along straight capillary tubes, in the streamline direction, the actual flow velocity would be greater than the Darcy velocity.

This work has only been developed in its most simplistic form. Extensive work by many authors has produced many volumes of mathematical treatises on the flow dynamics of a porous medium. Much of this work has been collected and described by Scheidegger (30) and it is proposed to briefly summarise the various conceptual models which have been developed.

2.6.2 Conceptual Models

To understand permeability it is first necessary to reduce it to fundamental concepts. In the case of a porous medium, this can be achieved by considering the parameters of capillary pressure and internal surface area, which in turn are related to the geometrical arrangement of pore spaces. Considerable work has therefore been undertaken, Scheidegger (30), to establish a relationship between permeability and the geometrical pore arrangement of a porous medium.

Model Type	Brief Description	Equation
Straight Capillary	A bundle of straight capillary tubes of uniform diameter	$k = \frac{P\delta^2}{32}$ or $k = \frac{P}{T^2S^2}$
Parallel Type	Re-orientation of the capillary tubes: 1) With one third in each of the 3 spatial dimensions 2) As above, but with variable pore diameter	$k = \frac{P\delta^2}{96}$ or $k = \frac{P}{96}\int_0^\infty \delta^2\alpha(\delta)d\delta$
Serial Type	Capillary tubes with different pore diameters, put together in series	$\frac{1}{k} = \frac{96T^2}{P}\left(\int \delta^2\alpha(\delta)d\delta\right)^2 \int \frac{\alpha(\delta)}{\delta^6}\,d\delta$
Branch Type	Where one capillary tube splits into two or more branches	——
Hydraulic Radius	Based on the assumption that porous medium are equivalent to a series of channels	$k = \frac{cm^2}{F(P)}$
Koseny	An assemblage of channels with various cross-sectional areas	$k = \frac{cP^3}{(TS^2)}$
Drag Type	The walls of the pores are treated as obstacles to an otherwise straight flow of the fluid	Fibre Theory $k = \frac{3}{16}\frac{P\delta^2}{1-P}\frac{2-\ln[\delta q\rho/(\mu P)]}{4-\ln[\delta q\rho/(\mu P)]}$ Sphere Theory $k = \frac{R^2}{18}\left(3 + \frac{4}{1-P} - 3\sqrt{\frac{8}{1-P} - 3}\right)$ where $\frac{4}{3}\pi n R^3 = 1-P$

List of Symbols

P - Porosity
δ - Pore diameter
k - Permeability
T - Tortuosity
S - Specific Surface Area of Pores or Tube

$\alpha(\delta)$ - Differential Pore Size Distribution
c - Dimensionless Constant
m - Hydraulic Radius
F(P) - Porosity Factor
ρ - Fibre density

μ - Fluid Viscosity
n - No. of spheres
R - Radius of spheres
q - Volumetric Flow

Table 2.1 Conceptual Models of Flow through a Porous Media (after Scheideggar (30))

Table (2.1), lists seven conceptual models which have been derived to explain flow through a porous medium. Inspection of the relevant equations immediately reveals, rather vague parameters such as Tortuosity (T), Differential pore size distribution ($\alpha(\delta)$) and Specific surface area of the pore or tube (S). Scheidegger (30), neatly sums up the existence of these parameters in the statement:

'Most empirical correlations contain factors which are usually vaguely defined and related to alleged geometrical quantities of the material. However, they are nothing but undetermined factors used in order to make the data fit the desired equations.'

In general, derived empirical methods do not fit either laboratory or field results with any marked degree of certainty. This suggests that in formulating an equation, too many arbitrary assumptions have to be made. In the authors opinion, while these exercises are admirable in both theoretical content and the attempt at mathematically defining flow through idealised materials, they are seldom applicable to real rock types. It is therefore considered necessary to derive empirical relationships from collected field and laboratory results, rather than to firstly derive an equation and then fit it to the results.

2.7 Determination of Aquifer Parameters

2.7.1 Choice of Technique

A variety of field and associated analytical techniques are available for determining aquifer and groundwater parameters. In each case the analysis assumes laminar flow and Darcian conditions to exist within the test formation. The choice of analysis along with a description of its use has been adequately described by numerous authors, notably Kruseman and De Ridder (32), Walton (25), Bouwer (6) and Bear (33).

Choice of field and analytical methods is dependent upon existing site conditions and the required parameters. However, ultimately the choice of test is always between one of two types:

1) 'Pumping Out' Tests.

2) 'Pumping In' Tests.

2.7.2 'Pumping Out' Tests

All aquifer systems with abundant supplies of water, such as the Permo-Trias and Cretaceous, allow determination of their parameters by 'pumping out' techniques. In essence, this involves test pumping a well and monitoring the resultant drawdown, not only in the pumped well but also in nearby observation wells. Pumping can be undertaken at either constant or variable rates with the number of monitoring sites dependent upon the number available.

The type of analysis to which the data is subjected, will depend on the test classification:

1) Steady State - where pumping is continued in the production hole until water levels in adjacent observation wells reach equilibrium. In practice, true equilibrium is seldom reached.

2) Nonsteady or Transient State - where water levels in both the production and observation wells are monitored with respect to time.

The wide selection of analytical techniques available are well documented by Kruseman and De Ridder (32). However, all have been subsequently derived from the original work of Darcy (29) and Theis (34). In practical terms, the equations work extremely well for aquifer systems which exhibit predominantly intergranular permeability and this has been proved world-wide by consistent and dependable results.

2.7.2.1 Application to Formations Exhibiting Fracture Permeability

It has previously been noted, that aquifers which show a marked degree of fracture/fissure permeability do not display the same relationship to Darcian conditions, as aquifers with intergranular permeability. The question therefore arises, with what degree of certainty can results be relied upon, when derived from a test formation exhibiting predominantly fracture/fissure permeability.

Eagon and Johe (35), examined results from 76 wells and boreholes sunk in limestone/carbonate formations of the Silurian/Devonian in Ohio, USA. At each well site, three types of test where undertaken:

1) A 2 hour trial test, with variable pumping rate.
2) A standard step test [Clarke (36)].
3) A 24 hour constant rate test.

Analysis of the results from each well was then undertaken using the most applicable method. In every case, it was assumed that aquifer permeability was via fracture/fissure networks and that these were connected on an areal basis. Similarly, although each test site was confined beneath a layer of glacial till, recharge capability was assumed by vertical leakage.

It was concluded that both the data and analytical methods were valid, although the results should be interpreted with care. Minor inconsistencies did exist, but realistic transmissivity values could be obtained, especially when the results were considered on a regional basis.

In another case, Radhaknshna and Venkateswarlu (37), examined constant rate tests conducted in an ultramafic orebody at Orissa, India. Detailed examination of the results concluded that weathered, semi-weathered and fractured aquifers in hard rock terrain could be successfully analysed by existing techniques and produce reliable hydrogeological parameters.

In conclusion, parameters for predominantly fracture/fissure permeability aquifers can be reliably determined using 'pumping out' techniques, although the results should be treated with caution.

2.7.2.2 Application to Coal Measures Strata

Although 'pumping out' techniques can be used in some of the Carboniferous formations (localised areas of the Limestone and Millstone Grit sequences), they are seldom successful in Coal Measures. The Coal Measures lithology is extremely variable, both in areal extent and sequence. Aquifers, therefore tend to be small with finite quantities of water contained within low yielding formations. Insufficient water is usually present to allow prolonged pumping.

'Pumping out' techniques usually require the installation of a submersible pump. Test boreholes must therefore be of sufficient diameter to allow insertion and stable enough to prevent subsequent damage. Since only vertically downholes can be used, the technique has limited use underground. Similarly, because stable borehole conditions are required, monitoring cannot be successfully undertaken in the vicinity of dynamic conditions, such as a working longwall panel.

2.7.3 'Pumping In' Tests

When strata formations are encountered which have a low permeability or are impermeable, insufficient yield is available for determination of their hydrogeological parameters by 'pumping out' techniques. An alternative method must therefore be used whereby water is 'pumped in'. Three main types of 'pumping in' test are available:

1) Falling Head - involves the addition of a known quantity of water to a hole and then measuring the time taken for the water level to return to its original level.

2) Constant Head - involves the addition of water to maintain a given head level within the hole.

3) 'Packer' Testing - involves either the addition or extraction of water from a section of borehole which has been isolated from the remainder of the hole.

2.7.3.1 Analytical Techniques

A comprehensive analysis of falling and constant head techniques for the determination of permeability has been described by Hvorslev (38). In summary, the two basic equations are, Hoek (39), Constant Head $$K = \frac{Q}{F.H_c} \qquad \text{..........} \quad 2.13$$

Falling Head $$K = \frac{A}{F(T_1-T_2)} \cdot \ln \frac{H_1}{H_2} \qquad \text{..........} \quad 2.14$$

where

A - Cross-sectional area of water column.

F - Shape factor of the hole.

H_1 and H_2 - Water levels in the hole, measured from the rest level, at times T_1 and T_2.

Q - Flow rate.

H_c - Water level, measured from the rest level.

The shape factor F, of the hole or test cavity is particularly important, since it is used as a means of accounting for a variety of conditions which may exist, such as partial penetration and/or impermeable boundaries. Table 2.2, summarises five of the most relevant end conditions usually encountered in field tests, along with their applicable shape factor equations.

Constant and falling head tests are normally conducted in vertically down holes, to which equations 2.13 and 2.14 are applicable. However, as previously mentioned, downholes have only limited use when applied to underground workings. If pressure testing of the hole(s) is undertaken and constant head - steady state conditions achieved or assumed, then this method can be used for holes of any inclination. However, to apply equation 2.13, the test pressure must be converted to an equivalent head,

$$P_1 - P_2 = P_E \qquad \text{..........} \quad 2.15$$

and $$P_E \times c = H_E \qquad \text{..........} \quad 2.16$$

where

P_1 - Test pressure at base of hole.

P_2 - Pressure loss between hole base and test cavity.

c - Conversion constant.

H_E - Equivalent head.

If a shape factor equation (Table 2.2, No. 4) is substituted into equation 2.13, Hoek (39),

$$K = \frac{Q \ln(2m\ L/D)}{2\ LH_R} \qquad \text{..........} \quad 2.17$$

and

$$m = (k_h/k_v)^{\frac{1}{2}} \qquad \text{..........} \quad 2.18$$

where

k_h - Horizontal strata permeability.

k_v - Vertical strata permeability.

L - Length of test cavity.

D - Borehole diameter.

H_R - Constant head above original level.

The significance of anisotropic permeability within a test hole (or cavity)is shown in Table 2.3. In equation 2.17, the term ln(2m L/D) does not significantly effect the value of K. Normally, the ratio of k_h/k_v is not known and estimated values have to be substituted. Hoek (39) suggests values of $k_h/k_v = 10^6$ and $m = 10^3$ as being reasonable. Additional work by Whittaker and Singh (40) and Neate (41) confirms the hypothesis.

This technique can be used to monitor dynamic conditions, since no pumping system has to be installed within the hole(s). Neate (41) describes three such tests conducted within the vicinity

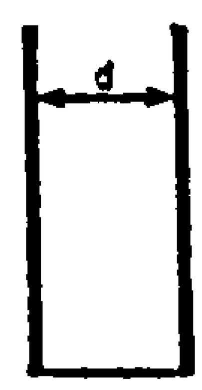
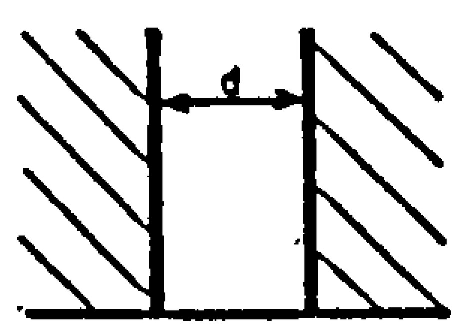
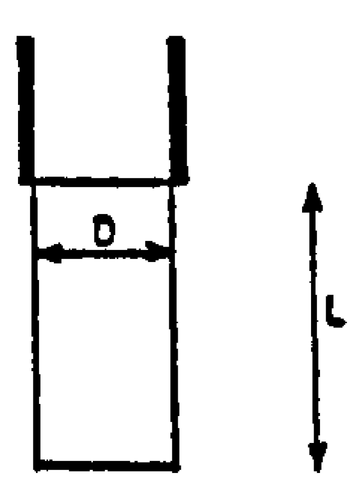
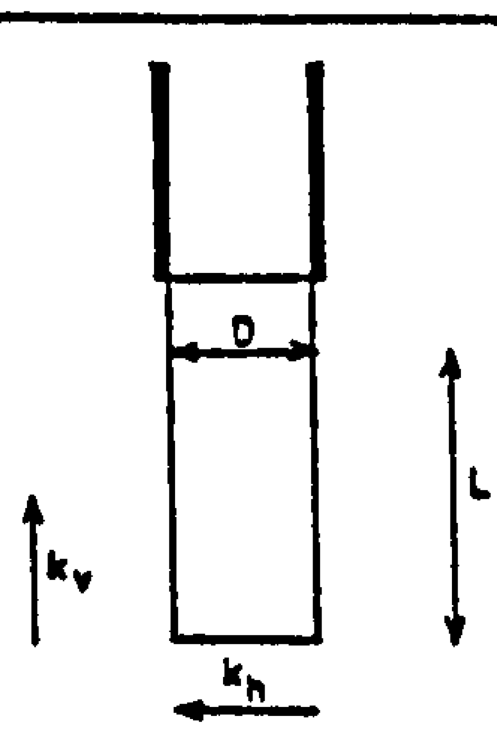
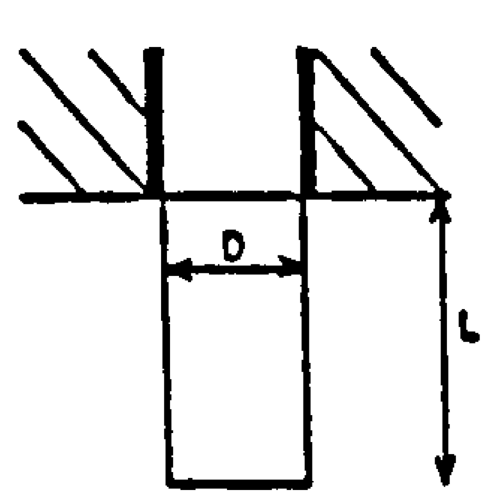
End Conditions		Shape Factor F	
	Casing flush with end of borehole in soil or rock of uniform permeability. Inside diameter of casing is d.	$F = 2.75d$	No. 1
	Casing flush with boundary between impermeable and permeable strata. Inside diameter of casing is d.	$F = 2.0d$	No. 2
	Borehole extended distance L beyond the end of casing. Borehole diameter D	$F = \dfrac{2\pi L}{\mathrm{Log}_e(2L/D)}$ for $L > 4D$	No. 3
	Borehole extended distance L beyond end of casing in a stratified soil or rock mass with different horizontal and vertical permeabilities	For determination of K_h $F = \dfrac{2\pi L}{\mathrm{Log}_e(2mL/D)}$ where $m = (k_h/k_v)^{\frac{1}{2}}$, $L > 4D$	No. 4
	Borehole extended a distance L beyond the end of casing which is flush with an impermeable boundary	$F = \dfrac{2\pi L}{\mathrm{Log}_e(4L/D)}$ for $L > 4D$	No. 5

Table 2.2 Table of End Conditions and Shape Factors for Constant and Falling Head Conditions
(after Hoek and Bray (39))

k/k_p	1.0	10^2	10^4	10^6	10^8	10^{10}	10^{12}
m	1.0	10^1	10^2	10^3	10^4	10^5	10^6
$\mathrm{Log}_e(2m\ L/D)$	2.1	4.4	6.7	9.0	11.3	13.6	15.9

Table 2.3 Significance of Anisotropic Permeability on a Test Cavity (After Hoek (39)

of working longwall faces. The application of this method is considered particularly important for measuring potential permeability changes around workings which lie beneath large surface or sub-surface water bodies. However, it should be noted that the equations discussed are only applicable to test formations which exhibit a predominant intergranular permeability. Laminar flow and Darcian conditions are assumed to occur around the test hole or cavity.

2.7.3.2 Application to Fractured Strata

In chalk and sandstone aquifers with predominantly fracture/ fissure permeability, Price et al (42) have found that the assumption $(k_h/k_v)^{\frac{1}{2}}$ (equation 2.18) cannot be justified when the test cavity is significantly intersected by fissure systems. However, an approximate solution has been derived by Barker (43) for the transmissivity of a 'packer' test cavity intersected by an infinite horizontal fissure, equation 2.19.

$$T = (Q/2\ H)\ \ln\ [T/Cr_w(k_h\ k_v)^{\frac{1}{2}}] \qquad \text{..........} \quad 2.19$$

where

Q - Flow rate

H - Injection head

r_w - Borehole radius

k_h and k_v - Horizontal and vertical matrix permeability

C - Exp γ, where γ is Eulers constant.

T can be solved by calculating a sequence of values $T^{(1)}, T^{(2)}, T^{(3)}\ldots$ using

$$T^{(n)} = (Q/2\pi H)\ \ln\left[T^{(n-1)}/Cr_w(k_h k_v)^{\frac{1}{2}}\right] \quad \ldots\ldots\ldots\ldots \quad 2.20$$

until convergence is obtained with reasonable accuracy. Barker (43) suggests that an initial value of $T^{(o)}$ should be chosen as greater than $Q(2\pi H)$, since this guarantees convergence within a few iterations.

Price et al (42) have simplified equation 2.19 to,

$$T_f = \frac{Q}{2\pi H}\left|\ \ln\left(\frac{T_f}{r(k_h k_v)^{\frac{1}{2}}}\right) - 0.5772\right| \quad \ldots\ldots\ldots\ldots \quad 2.21$$

where investigations within the chalk aquifers of Hampshire have produced the following range of transmissivity values for a cavity intersected by a single large fissure,

57 m/day	(Equation 2.13)
130 m/day	(Equation 2.17)
210 m/day	(Equation 2.21)

It is therefore concluded that where large fissures are known to exist, calculation of aquifer parameters by the normal Hvorslev equations, seriously underestimates the contribution made by these features.

The author suggests that this method should be extended from test cavities intersected by single large fissures to cavities in fractured rock. Since this would be particularly useful when monitoring strata conditions in the vicinity of longwall workings where induced fracture/fissure networks are known to occur. A full range of results could then be calculated using equations 2.13, 2.17 and 2.21 and the results interpreted with respect to first hand knowledge of the strata behaviour.

2.7.3.3 Application to Coal Measures Strata

Although different lithological and structural variations are encountered within the Coal Measures, except where highly fractured zones exist, the strata has either a low permeability or is impermeable. 'Pumping in' techniques are ideally suited to these conditions, since they were originally developed for low permeability environments and are not dependent upon high or consistent formation yields. In addition, the technique is not restricted to surface or static sites, but can be readily adapted for underground and dynamic conditions with the use of up and/or inclined boreholes.

The siting of test boreholes should be undertaken to minimise boundary effects between one or more rock types. This is particularly important within the Coal Measures. A detailed knowledge of the site geology is therefore essential before installation. Similarly, the assumptions necessary for the development of workable equations means that absolute 'in-situ' permeability values cannot be calculated. However, a series of observations at a static site, will provide base values against which values collected under dynamic conditions can be compared.

2.8 Conclusion

Initial development of analytical techniques assumed that a material was isotropic, homogeneous and exhibited Darcian conditions. Subsequent work has developed further equations suitable for a variety of simplified aquifer properties and boundary effects, but still assuming Darcian conditions. Examination of field site data reveals the flaw in this hypothesis. Aquifer formations seldom display Darcian conditions and except in extremely localised areas exhibit considerable lithological and structural anisotropy. The solving of non-Darcian flow regimes is extremely complex and involves considerable substitution of arbitrary values into undetermined factors. A compromise therefore has to be reached between semi-applicable Darcian equations and the extremely complex non-Darcian equations.

Studies on a world wide basis have revealed the full extent and limitations of Darcian equations. In a majority of cases, the results although not necessarily 'absolute' are at least reliable and can be interpreted provided the interpreter has sufficient experience and knowledge. A dilemma therefore confronts the mining hydrogeologist between the use of various analytical techniques and interpretation of different results from the same original field data.

The two main techniques for field determination of aquifer parameters have been outlined and discussed with respect to their use in intergranular and fracture/fissure conditions. The Coal Measures in particular have low permeabilities and exhibit both types of condition. 'Pumping in' techniques are considered preferrable to 'pumping out' under these circumstances, primarily because of the ease and versatility of use under a variety of conditions.

CHAPTER 3

MINING SUBSIDENCE AND ITS RELATIONSHIP TO SHALLOW WORKINGS AND GEOLOGICAL CONDITIONS

CHAPTER 3

MINING SUBSIDENCE AND ITS RELATIONSHIP TO SHALLOW WORKINGS AND GEOLOGICAL CONDITIONS

3.1 Introduction

Historical mining activity throughout the world has resulted in a variety of ground movements which ultimately form either sink hole structures or subsidence profiles. Formation of these profiles at ground surface (particularly since the introduction of longwall mining techniques) can result in serious damage to surface structures. A gradual understanding of the formation mechanism, particularly during the 1950's, led to completion during the 1960's of the NCB Subsidence Engineers Handbook (3), which allows accurate prediction (±10%) of longwall subsidence profiles by empirical methods. However, since the prediction methods are empirical, it is often difficult to quantify the effects of single parameters such as geology or the presence of very shallow workings (less than 100 m depth) in relation to the eventual profile magnitude.

Hence, after a brief description of the concepts and methods of calculating subsidence profiles, this chapter examines the prediction of shallow working profiles (less than 140 m), the effects of geology on profile formation and the relationship between ribside fissure zones and surface geology. A comprehensive understanding of profile formation for all mining conditions will eventually lead to minimization of damage and a reduction in mining costs.

Name	Date	Worker	Theory
Vertical Theory	1820-30's	-	Subsidence only occurs vertically above the mining area. Hence leaving a coal pillar below a surface structure of similar size and shape to the structure, will result in zero damage.
Normal Theory	1850's	Gonot	Subsidence occurs in an area defined by normals projected from the seam to ground surface.
Dome Theory	1880's	Fayol	Ground movements around a working are delinated by an eccentric ellipsoid shape at right angles to the working.
Beam Theory	1900's	Halbaum	Strata above the working acts as a cantilever beam, when caving into the goaf area.
Empirical Methods	1950's	Orchard Wardell	Development of empirical methods in which prediction methods are calculated by the examination of numerous actual case histories. Subsequent analysis, usually involving dimensionless ratios allows the establishment of guidelines which can accurately predict the magnitude of surface subsidence and ground strains.
Empirical Methods	1960's 1970's	Subsidence Eng. Handbook	

Table 3.1 Historical Development of Subsidence Theory

(after Shadbolt (44))

3.2 Development of Subsidence Theories

A relationship between surface damage and underground mining activity was first recognised early in the 19th Century by Belgium engineers, Shadbolt (44). The subsequent historical development of subsidence theories is outlined in Table 3.1. Early theories tended to apply analytical methods, which due to the various parameters involved and lack of accurate and available field data, frequently failed and led to reappraisal. The gradual application of empirical methods during the 1940's and 1950's and in particular work by Wardell (45) and Orchard (46) led to an increased understanding of both the mechanisms involved and potential prediction techniques. A detailed account of the development of mining subsidence theory is given by Oliver (47).

Direct analysis of mining and subsidence parameters led to extremely variable results with low correlations. However, further analysis utilising dimensionless ratios revealed that maximum subsidence was related to the width-depth (w/h) ratio. Eventually, the collection and analysis of over 200 case histories spanning a wide range of mining and geological conditions allowed accurate prediction (±10%) of subsidence by empirical methods. This information is collected and presented in the Subsidence Engineers Handbook (3), along with details for numerically calculating subsidence profiles for any one of a variety of different mining conditions which may be encountered.

3.3 Mining Subsidence Parameters

Factors which affect the magnitude of an induced subsidence profile can be divided into two main types. However, a third type also exists, which although not related to the profile magnitude, is related to the amount of damage a surface structure will suffer during profile formation.

3.3.1 Mining Factors

Three main mining parameters can be related to the magnitude of an induced subsidence profile:

1) Depth of working
2) Width of working
3) Extracted seam height

Additional parameters which can also affect the magnitude of an induced profile are: Seam inclination, Presence of old workings, Packing systems and Length of extracted panel.

3.3.2 Site Factors

Four main site parameters can influence the final magnitude of the induced subsidence profile:

1) Soil type
2) Strata type
3) Geological discontinuities
4) Strata hydrogeology.

However, unlike mining parameters which can be rigorously defined, site parameters can only be loosely defined. Their influence on the final profile should therefore be interpreted by empirical methods.

3.3.3 Structural Damage Factors

The amount of damage caused to a surface structure, will be as much due to its structural condition as the final magnitude of the subsidence profile. Potential structural damage is related to seven parameters:

1) Size of structure
2) Shape of structure
3) Foundation design
4) Infra-structure design
5) Construction materials
6) Age
7) Standard of maintenance.

Although these parameters are rather vague, they can if necessary be more rigorously defined, Breeds (48). A good example showing the interaction of structural damage factors is given by Shadbolt (44):

' An old dilapidated building suffering from the effects of lack of maintenance and repair, will be more sensitive and react more violently than a similar building which has been well maintained and properly repaired, when both are subjected to the same amount of movement'.

A detailed description relating ground movement to the amount of surface damage, is considered beyond the scope of this work. However, the interested reader is referred to either Shadbolt (44) or for a more comprehensive treatment of subsidence damage estimation, the Subsidence Engineers Handbook (3).

3.3.4 Pseudo Mining Damage

Damage which although characteristic of mining activity, can also be caused by completely different forces. When this happens, it is usually due to one or a combination of the following reasons:

1) Subsidence due to aquifer pumping.
2) Clay shrinkage resulting from differential settlement.
3) Sulphate attack of brickwork and mortar.
4) Corrosion of inset reinforcements and supports.
5) Removal of underlying soil by drainage water.

A comprehensive treatment of pseudo mining damage is given in the Subsidence Engineers Handbook (3).

3.4 The Subsidence Profile

If the extracted area behind a face is left to cave naturally, the maximum subsidence experienced at surface will normally be 85 - 90% of extracted seam height. However, if some method of 'stowing' is used, then the maximum subsidence experienced at surface is reduced. Table 3.2 shows maximum subsidence as a percentage of extracted seam height for a variety of conditions, Shadbolt (44).

Condition	% Extraction
Natural Caving	90%
Solid Caving	45%
E. Midlands 1st seam working	80%
E. Midlands 2nd and subsequent seam working	90%

Table 3.2 Maximum Subsidence as a Percentage of Extracted Seam Height for a Variety of Mining Conditions

Consideration of a subsidence profile reveals that any given point on the surface will be displaced both vertically downwards and horizontally towards the central working axis. Resultant differential displacement will therefore result in a zone of apparent extension on the convex section of profile and a complimentary zone of compression on the concave section. These simplified zones of compression and tension, together with five basic ground movements which can be measured at surface, listed below, are shown in Figure 3.1, Shadbolt (44):

1) Vertical subsidence
2) Curvature
3) Tilt
4) Displacement
5) Strain - extension and compression.

The limit at surface to which the effect of subsidence can be measured is known as the 'angle of draw'. In British coalfields, the angle of draw is 35^{o} to seam normal.

During development of a subsidence profile, in addition to the formation of a transverse profile normal to the face direction, a dynamic or longitudinal profile is also formed parallel to the direction of working. The net result is development of a subsidence trough with associated zones of tensile and compressive strain, which is shown diagrammatically in Figure 3.2. The five basic parameters of ground movement develop simultaneously on both transverse and longitudinal profiles.

It was found during compilation of the Subsidence Engineers Handbook (3), that unless a face travelled more than a certain distance

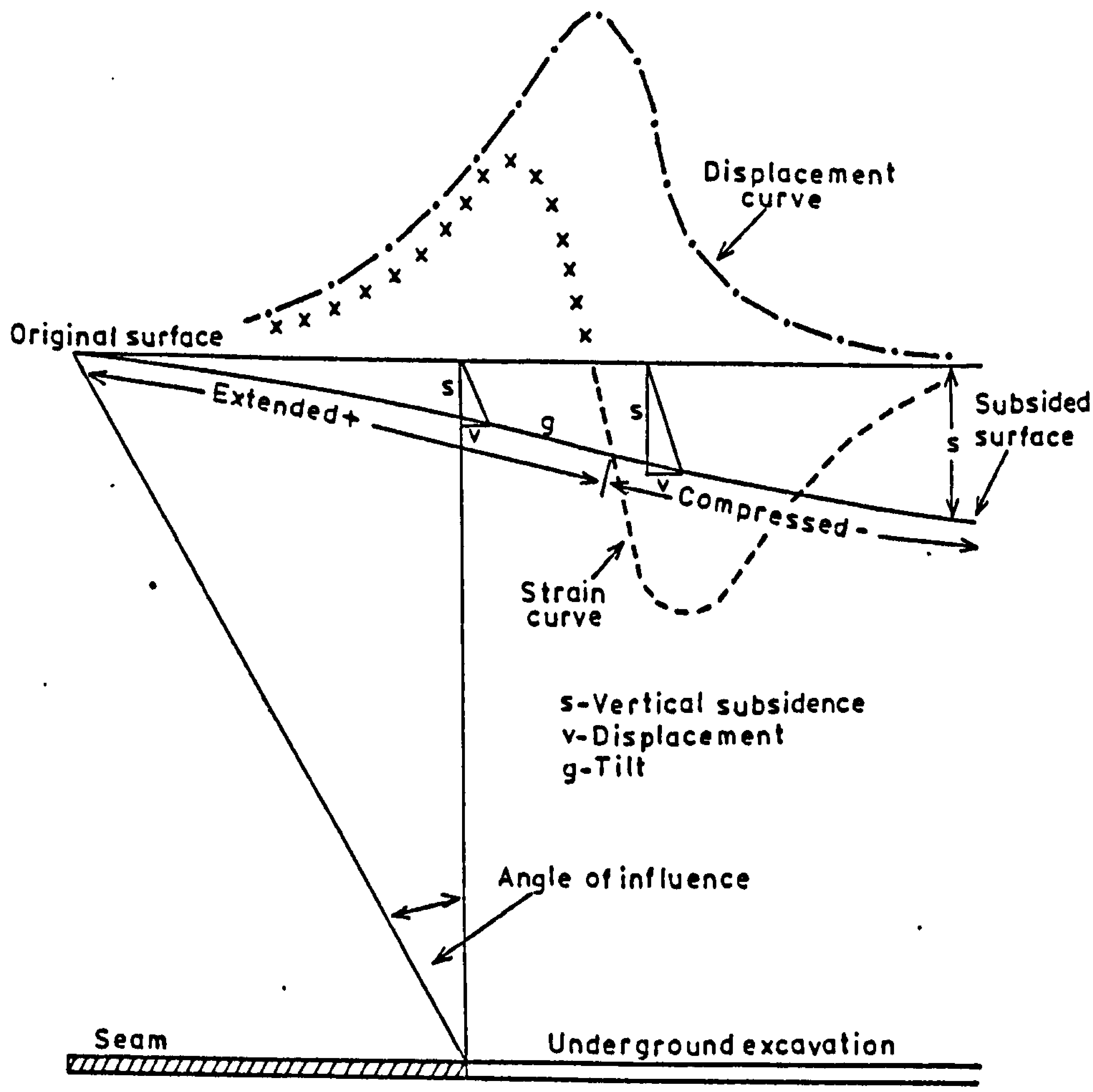
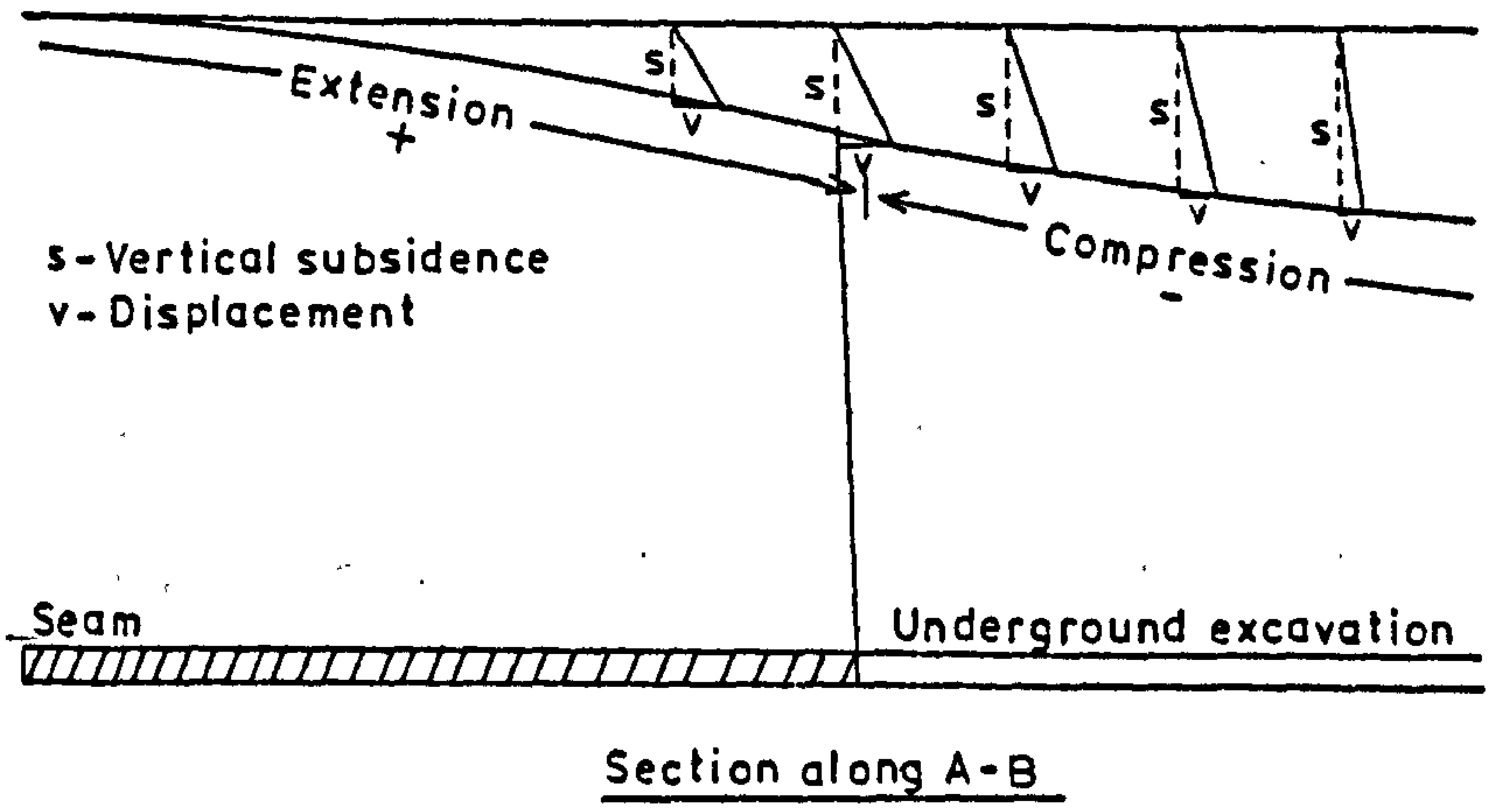


Figure 3.1 Elements of Ground Movement Associated with Subsidence Profile Formation
(after Shadbolt (44))



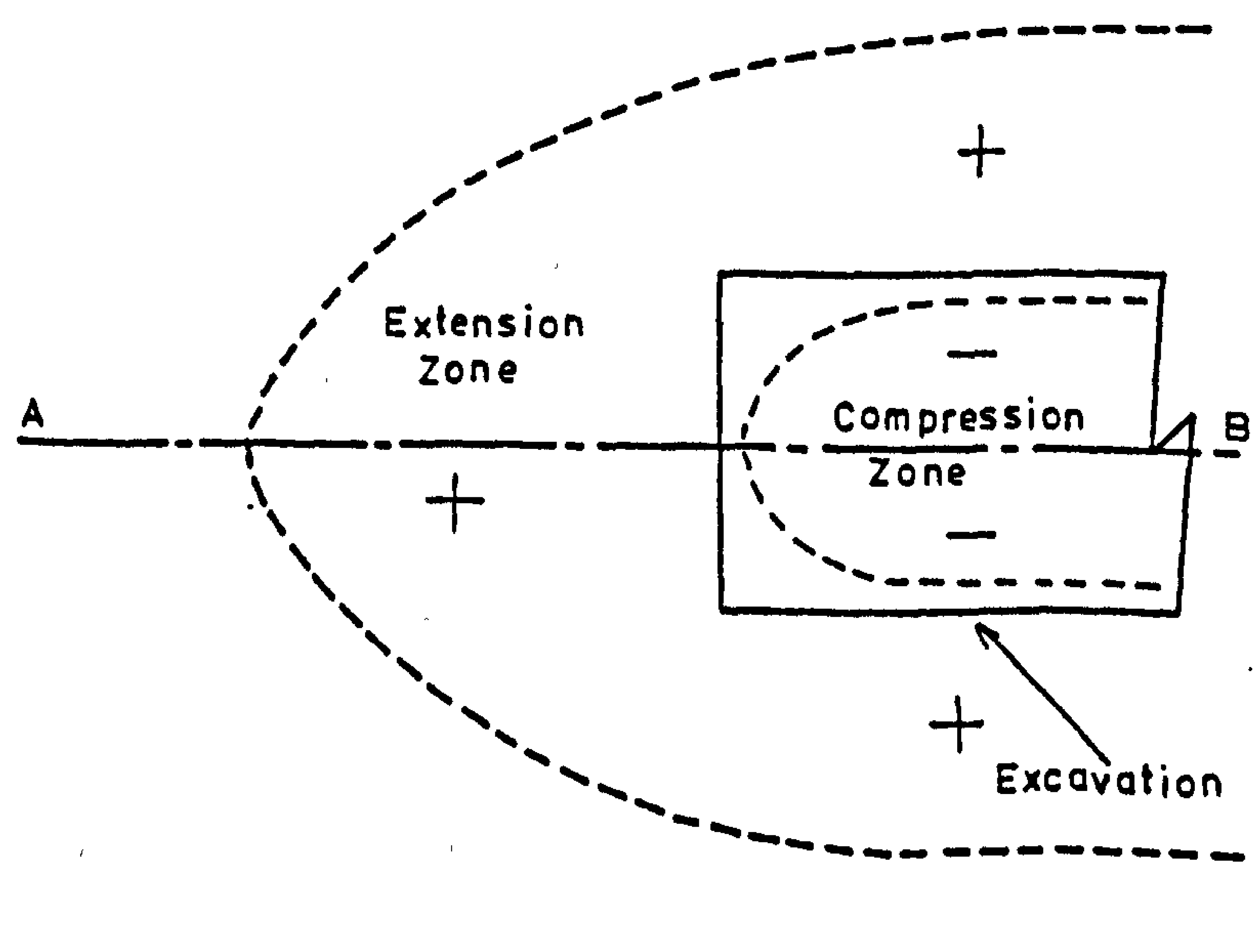


Figure 3.2 Zones of Tensile and Compressive Strain Developed Around a Longwall Extraction (after Shadbolt (44))

from the start line, only partial subsidence developed at surface. Subsequent analysis revealed this figure to be 1.4 times the seam depth. Three types of underground working can therefore be defined on this basis:

1) Sub-Critical Area - an area of working smaller than the critical area or when a point at surface does not undergo complete subsidence (Face advance < 1.4 seam depth).

2) Critical Area - an area of working which causes a point at surface to undergo complete subsidence (Face advance = 1.4 seam depth).

3) Super-Critical Area - an area of working greater than the critical area, but where a surface point still undergoes complete subsidence (Face advance > 1.4 seam depth).

3.4.1 Manual Subsidence Calculation

Details for calculating both transverse and longitudinal subsidence profiles have already been adequately described in the Subsidence Engineers Handbook (3) and will not be repeated here. However, it was found by the author, that the process could be simplified by development of a calculation format sheet. A copy of the transverse profile format sheet along with sample calculation is given in Figure 3.3. The method of calculation is identical to that carried out in the Subsidence Engineers Handbook. But, in the case of sub-critical panels, the amount of partial subsidence has to be pre-calculated.

3.4.2 Computer Subsidence Calculations

The detailed analysis of subsidence troughs requires computation of numerous profiles. Manual methods, while quick and easy for

NCB AREA North-East COLLIERY Westoe SEAM Brass Thill (K) DISTRICT/PANEL 2

PANEL WIDTH (w) 165 (metres) SEAM THICKNESS (m) 1.83 (metres) PANEL LENGTH (ℓ) 367 (metres)

PANEL DEPTH (metres)(h) 296 from SEABED w/h ratio = 0.56 S/m = 0.56

∴ S = 1.83 x 0.56

Subsidence as s/S	0	0.05	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	0.95	1.00
Subsidence in metres	0.00	0.05	0.10	0.20	0.31	0.41	0.51	0.61	0.71	0.82	0.92	0.97	1.02
Distance in metres from panel centre	290	175	139	104	86	74	62	53	44	35	24	18	0
Distance in terms of h from panel centre	0.98	0.59	0.47	0.35	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0

STRAIN EXTENSION +E COMPRESSION -E

e/E	0	0.20	0.40	0.60	0.80	1.00	0.80	0	0.20	0.40	0.60	0.80	1.00	0.80	0.60	0.40	0.20	0
	0	0.5	1.1	1.6	2.2	2.7	2.2	0	0.8	1.6	2.3	3.1	3.9	3.1	-	-	-	-
h =	290	180	151	130	115	98	83	65	56	50	38	29	9	0	-	-	-	-
296 m	0.98	0.61	0.51	0.44	0.39	0.33	0.28	0.22	0.19	0.17	0.13	0.10	0.03	0	0	0	0	0

+E = .8 S/h where S/h = 1.02/296 metres -E = 1.15 S/h

= 0.8 x 3.4 mm/m = 1.15 x 3.4 mm/m

= 2.7 mm/m = 3.9 mm/m

Figure 3.3 Subsidence and Strain Calculation Sheet - Transverse Profile

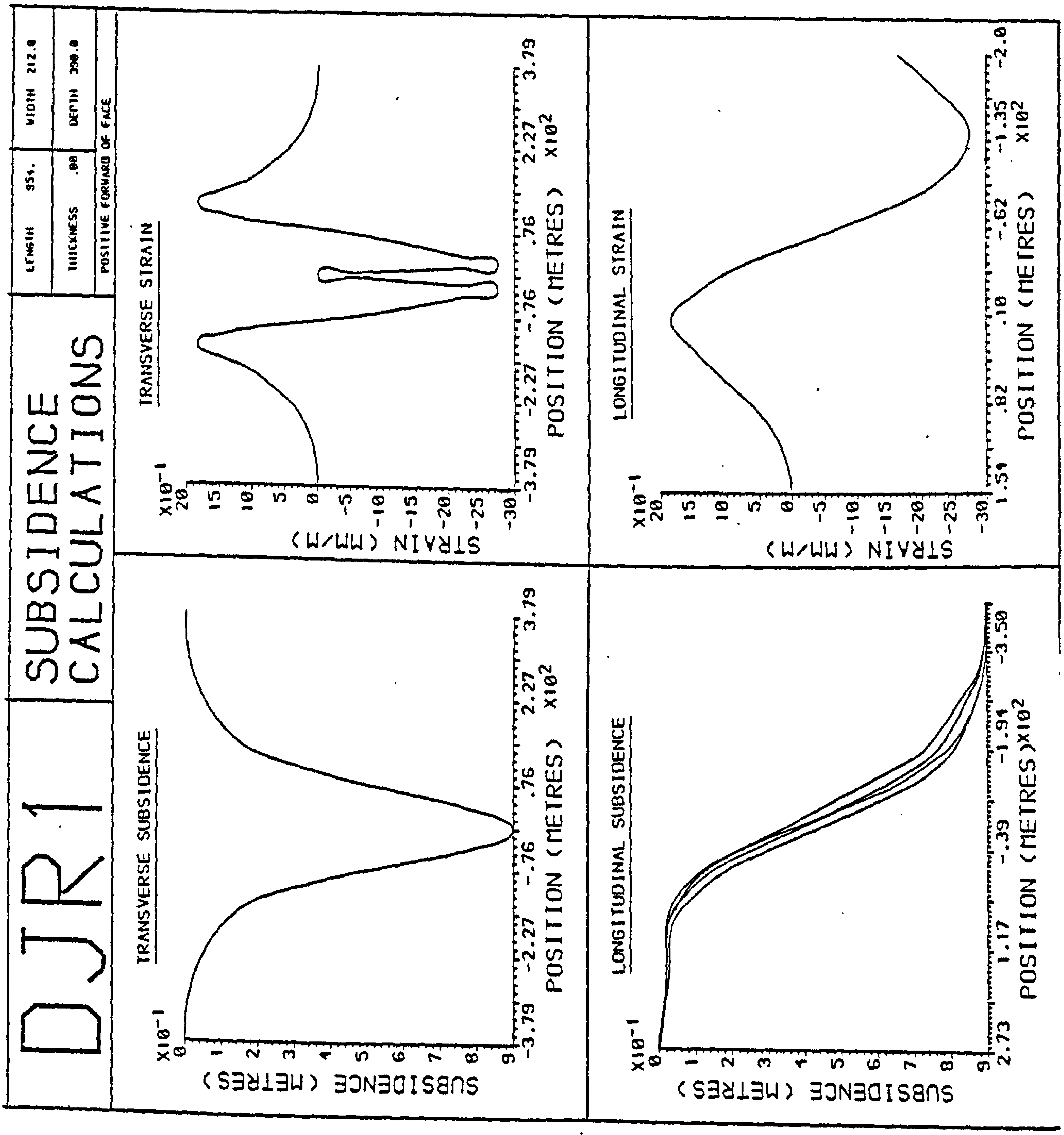


Figure 3.4 Two Dimensional-Subsidence and Strain Profiles
(after Reddish (50))

Figure 3.5 Two Dimensional-Subsidence Development Contours

(after Reddish (50))

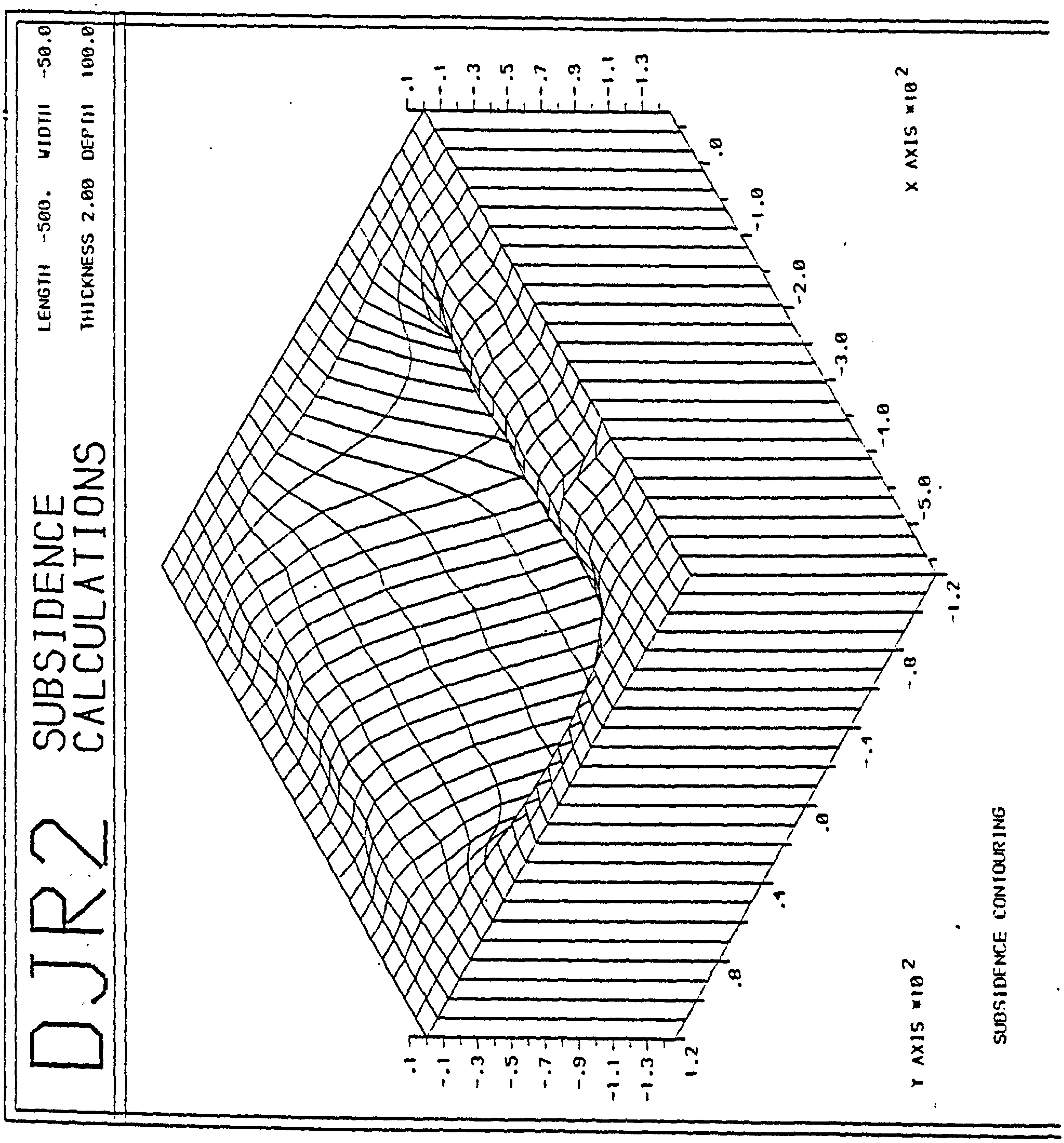


Figure 3.6 Three Dimensional-Subsidence Profile (after Reddish (50))

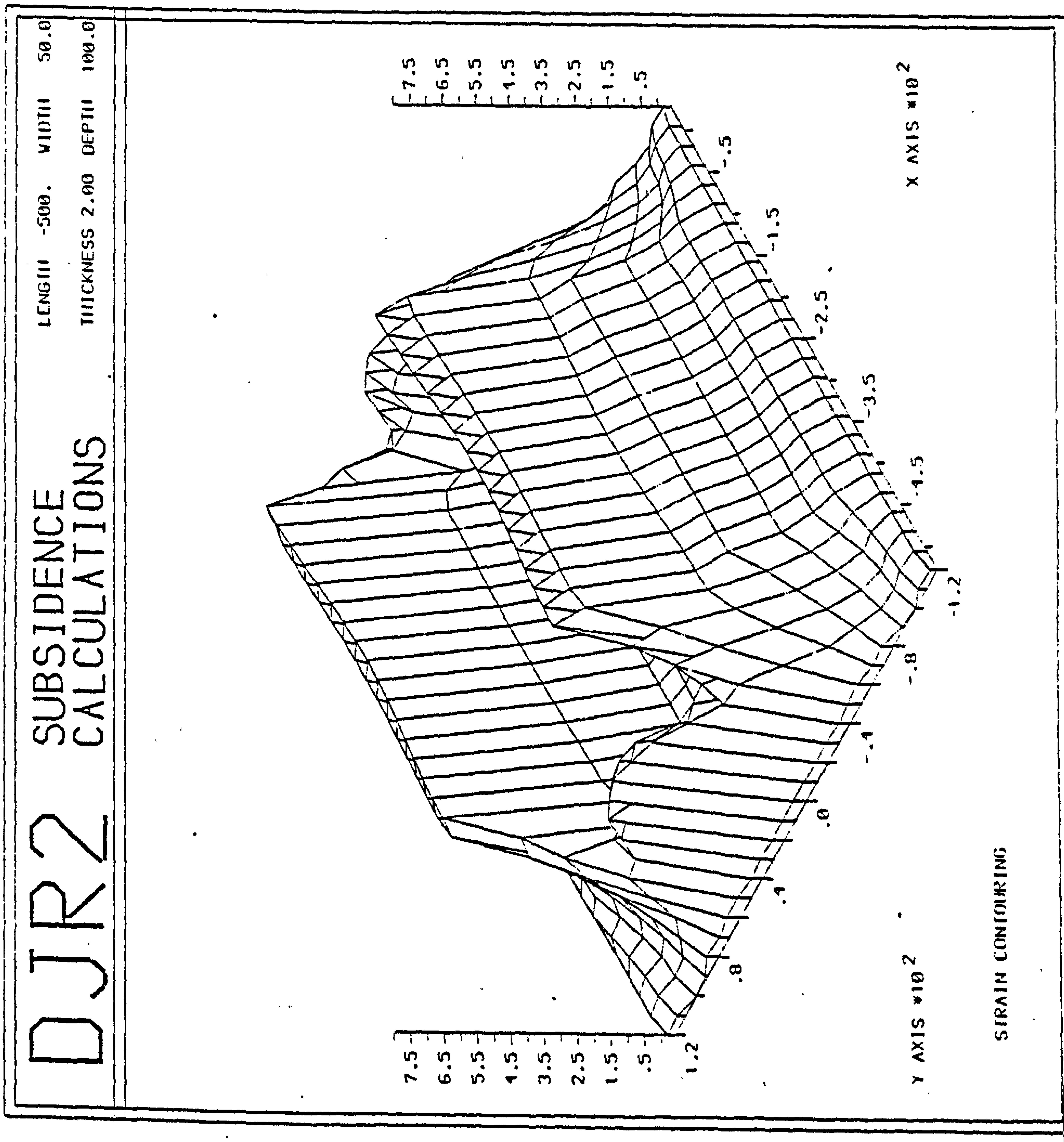


Figure 3.7 Three Dimensional-Strain Profile
(after Reddish (50))

limited calculations, rapidly becomes both tedious and time consuming for much greater numbers. A computerised method of calculating transverse and longitudinal profiles was therefore developed along lines of the manual methods outlined in the Subsidence Engineers Handbook (3). However, it was realised at the onset that provision should be made for calculation of shallow working profiles with width-depth ratios greater than 3.0. After consultation with the East Midlands Subsidence Unit (49), the Subsidence Engineers Handbook profile prediction graphs were extrapolated for transverse profiles, to a width-depth ratio of 14.0. In the case of longitudinal profiles, a set of prediction graphs was obtained from the East Midlands Subsidence Unit and again extrapolated.

Full details of the extrapolation methods are given in Appendix A, while the subsidence program and a step procedure diagram are given in Appendix B.

Finally, the 'subsidence' program has since been developed further by Reddish (50) to include graphical packages for visual presentation of the subsidence data, Figures 3.4 - 3.7.

3.5 Subsidence in Shallow Workings

Discussions by the author with the East Midlands Subsidence Unit, revealed that the Subsidence Engineers Handbook could be used to provide acceptable subsidence and strain profiles for both strata-strata and surface-strata interfaces. Since research by the author into the effects of subsidence on undersea workings (Chapter 8) has necessitated the derivation of profiles for significant geological and structural horizons. An attempt has therefore been made to test the validity of the theoretically derived profiles for the (w/h) range 3.00 - 14.0 against actual case histories. Recalibrated prediction values could then be calculated and substituted if necessary.

3.5.1 Data Collection and Method of Analysis

A search was made of case histories within the East Midlands Coalfield. The definition of a shallow working was initially taken as one at a depth of less than 100 m. However, the general lack of data required a revision to include all workings up to a depth of 140 m.

Data for 15 sites was eventually collected and a total of 23 sets of test data made available for analysis, Aston and Reddish (51). Actual face parameters were then fed through the 'subsidence' program described in Appendix B. The calculated (or predicted) profile parameters were then extracted and tabulated for comparison with actual profile parameters, Table 3.3. Once in tabulated form the various parameters were then processed by means of a graphical plotting routine to allow visual inspection of the results, Figures 3.8 - 3.15.

3.5.2 Calculated vs. Actual Profiles

In the case of Figures 3.8 - 3.12, all the points should lie on a straight line going through the origin. This would then indicate that prediction methods for shallow workings are a simple extrapolation of those for 'normal conditions' as given in the Subsidence Engineers Handbook (3).

CASE NO	CALC. S/H RATIO $\times 10^{-3}$	DEPTH (m)	WIDTH (m)	SUBSIDENCE Actual	SUBSIDENCE Calc.	TRANSVERSE STRAINS (mm/m) TENSILE Actual	TRANSVERSE TENSILE Calc.	TRANSVERSE TENSILE Ratio	TRANSVERSE COMPRESSIVE Actual	TRANSVERSE COMPRESSIVE Calc.	TRANSVERSE COMPRESSIVE Ratio	LONGITUDINAL STRAINS (mm/m) TENSILE Actual	LONGITUDINAL TENSILE Calc.	LONGITUDINAL TENSILE Ratio	LONGITUDINAL COMPRESSIVE Actual	LONGITUDINAL COMPRESSIVE Calc.	LONGITUDINAL COMPRESSIVE Ratio	W/D RATIO	ACTUAL S/H RATIO $\times 10^{-3}$
1	5.82	133.2	115.8	750	780	3.10	3.78	0.820	7.5	3.49	2.148	1.20	3.78	0.317	2.40	3.49	0.687	0.869	5.63
2	7.75	112.7	115.8	750	870	3.10	5.00	0.620	7.5	4.20	1.785	1.20	5.00	0.240	2.40	4.20	0.571	1.027	6.66
3	6.86	122.9	115.8	750	840	3.10	4.46	0.695	7.5	4.11	1.825	1.20	4.46	0.269	2.40	4.11	0.583	0.942	6.10
4	20.23	73.4	182.9	1379	1490	7.73	13.15	0.588	5.84	10.12	0.577	3.0	13.15	0.228	11.4	10.12	1.126	2.492	18.78
5	26.19	56.7	207.3	1379	1490	7.73	17.02	0.454	5.84	13.10	0.446	3.0	17.02	0.176	11.4	13.10	0.877	3.656	24.32
6	22.53	65.1	195.0	1379	1470	7.73	14.65	0.528	5.84	11.27	0.518	3.0	14.65	0.205	11.4	11.27	1.011	2.995	21.18
7	11.82	126.5	114.3	713	1500	2.09	7.70	0.271	2.81	7.10	0.396	2.23	7.10	0.289	1.10	7.10	0.155	0.904	5.63
8	19.72	88.4	114.3	713	1740	2.09	12.82	0.163	2.81	9.86	0.285	2.23	12.82	0.174	1.10	9.86	0.111	1.293	8.06
9	15.07	107.5	114.3	713	1620	2.09	9.80	0.213	2.81	8.05	0.349	2.23	9.80	0.228	1.10	8.05	0.137	1.063	6.63
10	53.73	33.5	132.0	2600	1800	58.00	34.90	1.660	52.00	26.90	1.933	-	-	-	-	-	-	3.940	77.61
11	13.88	86.9	375.0	735	1210	-	-	-	-	-	-	0.20	9.02	0.022	1.60	6.94	0.230	4.315	8.46
12	34.52	30.5	96.0	846	1050	10.90	22.40	0.487	19.40	17.26	1.123	-	-	-	-	-	-	3.148	27.74
13	25.71	56.0	254.0	1220	1440	13.00	16.70	0.778	18.70	12.90	1.450	-	-	-	-	-	-	4.536	21.78
14	36.30	30.0	90.0	940	1090	19.00	23.60	0.805	-	-	-	-	-	-	-	-	-	3.000	31.33
15	30.60	30.0	130.0	900	920	30.00	19.90	1.507	13.00	15.30	0.850	-	-	-	-	-	-	4.333	30.00
16	12.95	126.5	118.9	883	1640	3.67	8.42	0.435	6.04	7.77	0.778	2.55	8.42	0.303	2.44	7.77	0.314	0.940	6.98
17	11.18	140.2	118.9	883	1570	3.67	7.27	0.505	6.04	7.04	0.860	2.55	7.27	0.351	2.44	7.04	0.347	0.848	6.30
18	11.84	133.4	118.9	883	1580	3.67	7.70	0.477	6.04	7.10	0.851	2.55	7.70	0.331	2.44	7.10	0.344	0.891	6.62
19	15.80	100.0	138.0	1440	1580	16.50	10.27	1.610	14.40	7.90	1.820	-	-	-	-	-	-	1.380	14.40
20	18.88	88.0	126.0	1750	1660	16.00	12.27	1.300	13.60	9.44	1.440	-	-	-	-	-	-	1.432	19.88
21	10.80	122.0	187.0	1070	1320	7.00	7.02	1.000	4.00	5.40	0.741	-	-	-	-	-	-	1.533	8.77
22	15.84	110.0	200.0	1200	1740	-	-	-	-	-	-	-	-	-	-	-	-	1.818	10.91
23	10.69	112.0	314.0	-	1320	-	-	-	-	-	-	2.00	6.95	0.287	8.60	5.34	1.610	2.804	-

Table 3.3 Calculated and Actual Subsidence Profile Parameters for the 23 sets of Test Data

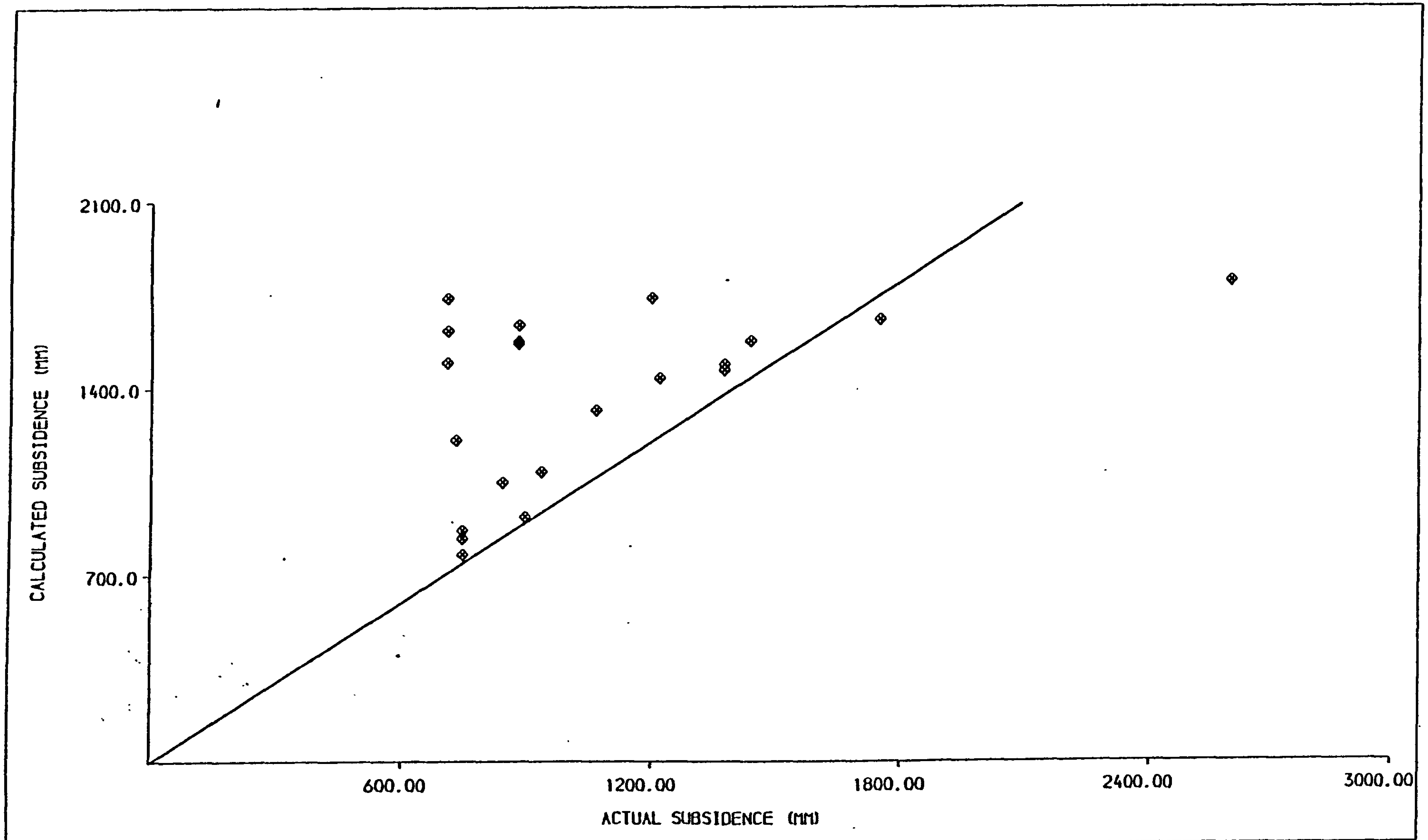


Figure 3.8 Calculated vs. Actual Maximum Subsidence

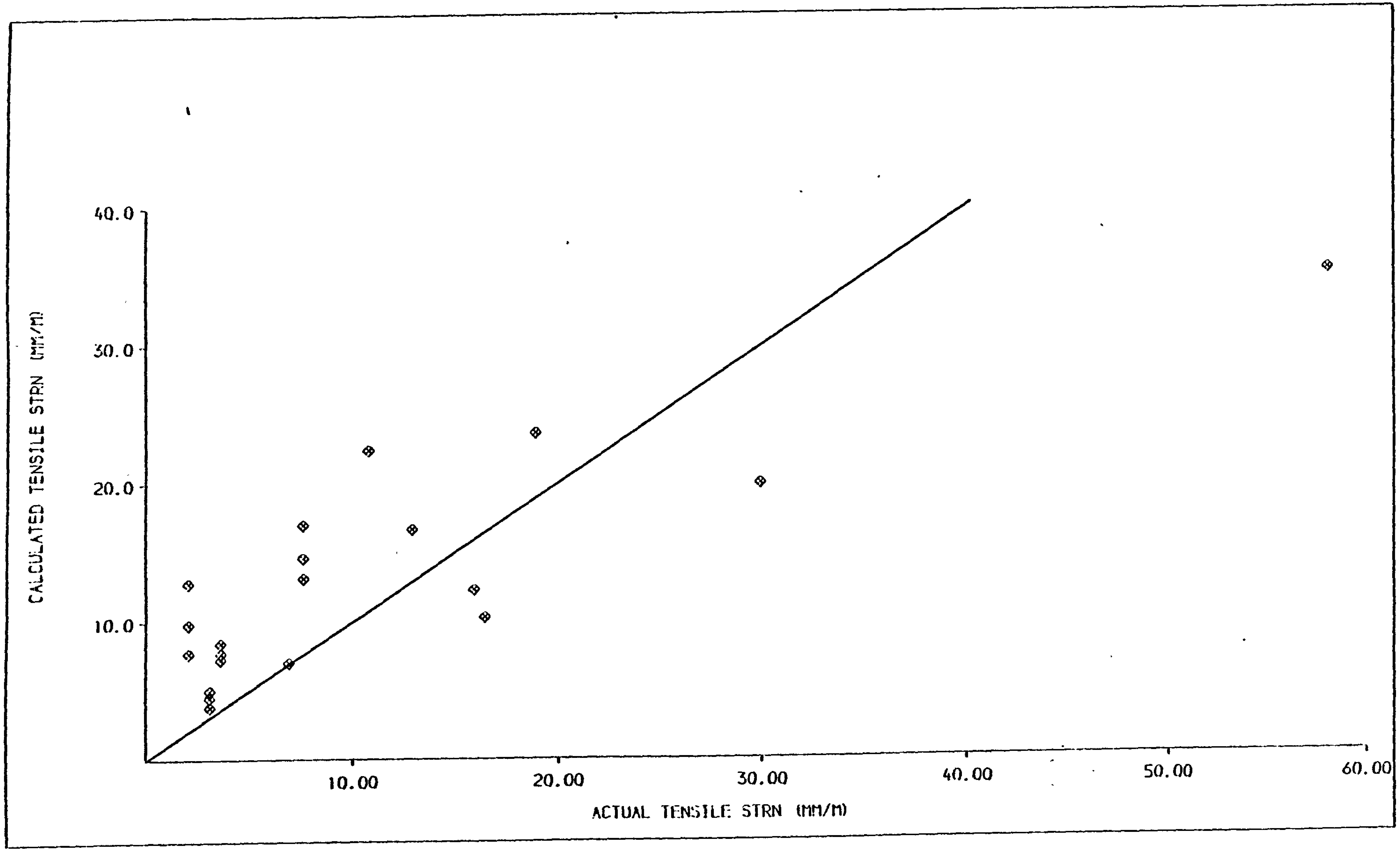


Figure 3.9 Calculated vs. Actual Maximum Tensile Strains - Transverse Profile

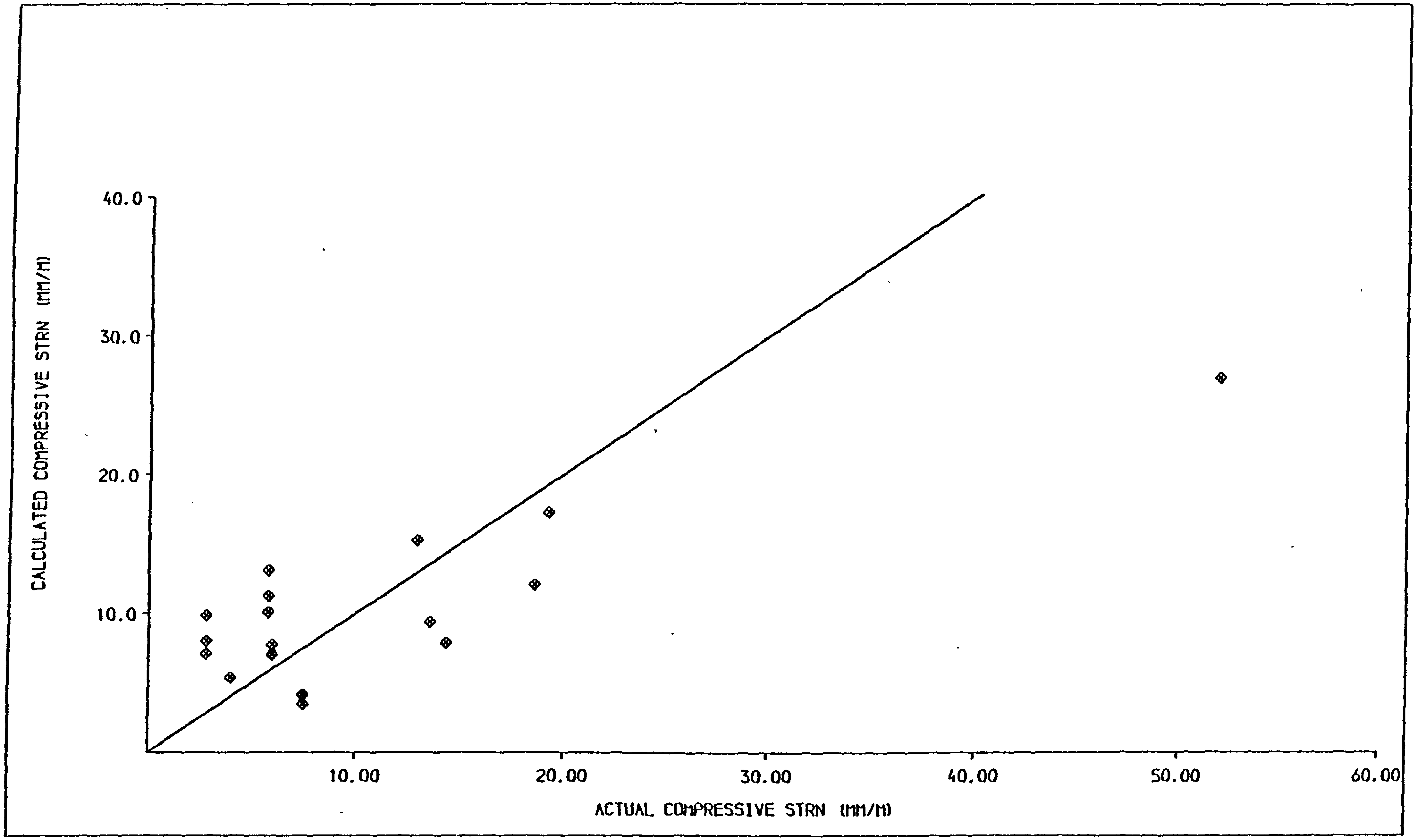


Figure 3.10 Calculated vs. Actual Maximum Compressive Strains - Transverse Profile

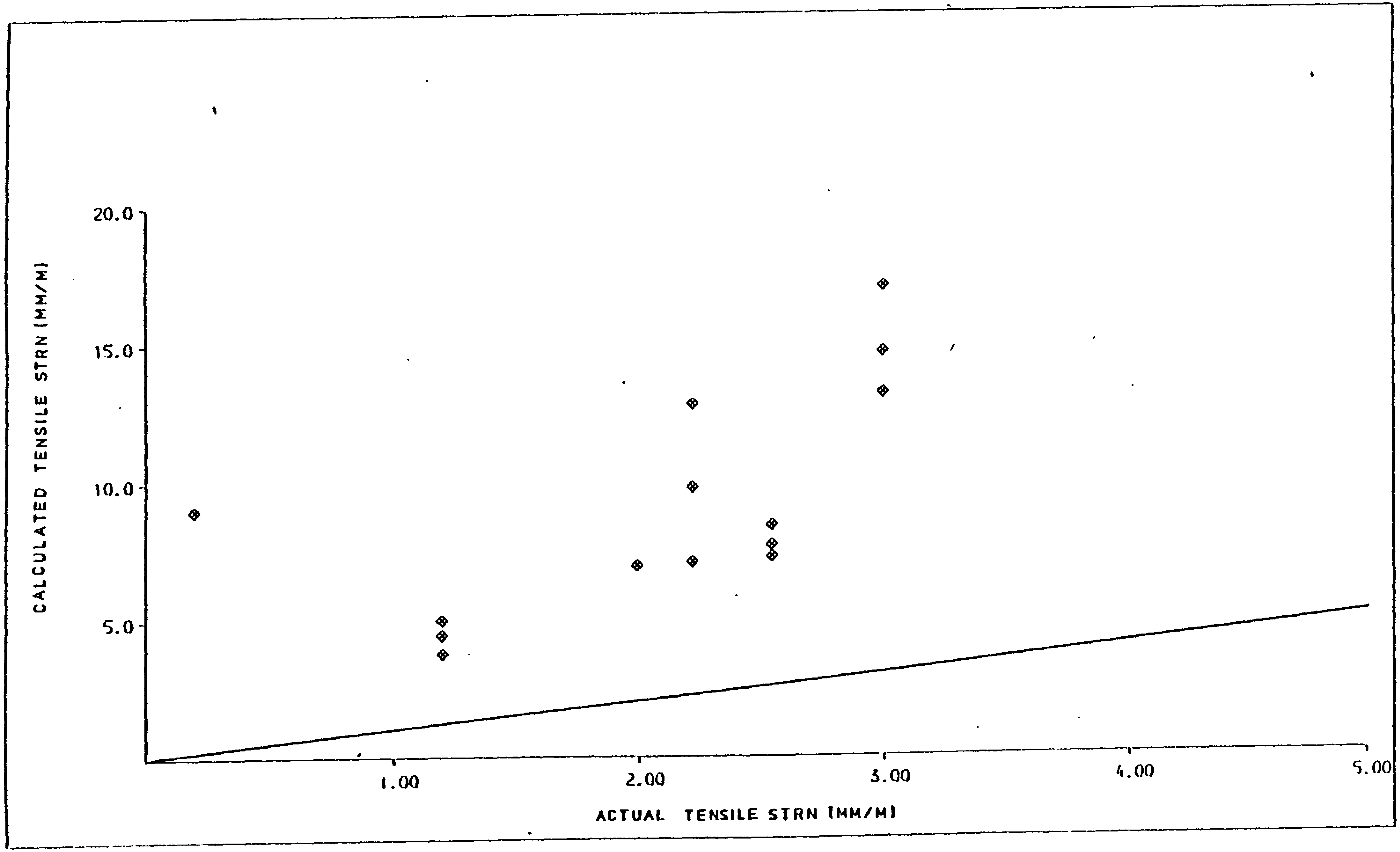


Figure 3.11 Calculated vs. Actual Maximum Tensile Strains - Longitudinal Profile

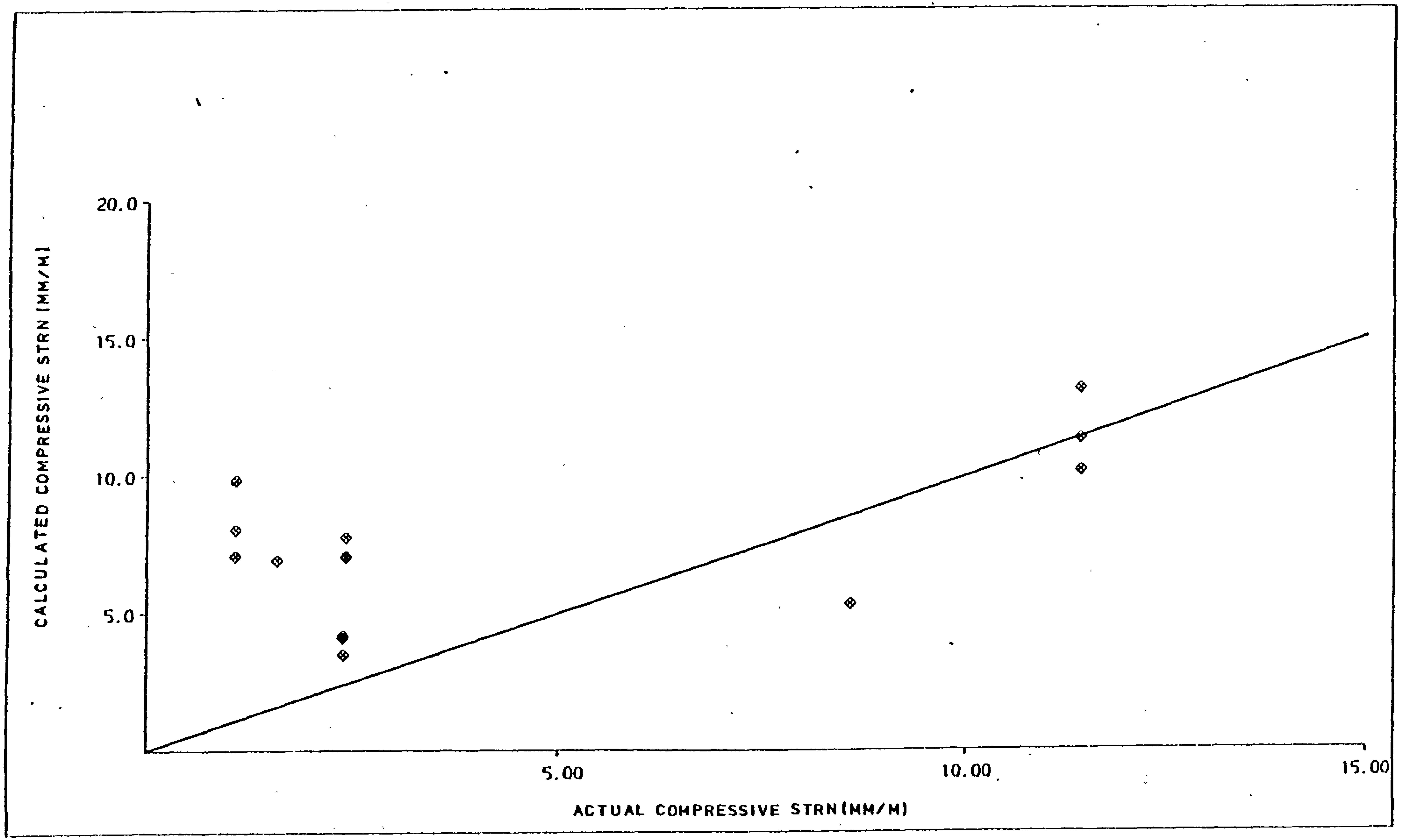


Figure 3.12 Calculated vs. Actual Maximum Compressive Strains - Longitudinal Profile

Figure 3.8 shows Calculated vs. Actual Subsidence with a scatter of points above the idealised line. This suggests that the current method of calculating subsidence for shallow workings produces higher values than those actually experienced in the field. Figures 3.9 and 3.10 show Calculated vs. Actual Tensile and Compressive Strains for transverse profiles. In each case the scatter of points lies around the idealised line, although a suggestion of skewness indicates that calculated values may be too low when very high strains are encountered. However, since only one point exists within this region it should be treated with extreme suspicion. Figures 3.11 and 3.12 show Calculated vs. Actual Tensile and Compressive Strains for longitudinal profiles. In the case of tensile strain, Figure 3.11, the scatter lies above the idealised line, while for compressive strain, Figure 3.12, it trends around the idealised line. This suggests that while prediction of longitudinal compressive profiles could be made from existing methods, the calculated values of tensile strain may be too high.

Table 3.4 shows the actual and calculated location, in metres, from face centre line (transverse profile) of the maximum tensile and compressive strains for the 23 cases. Table 3.5 shows similar values in metres, from face line for the longitudinal profile. In both cases, although some anomalies do exist, the agreement between actual and calculated positions is very good and this is further supported by the Actual/Calculated ratio.

A variation therefore appears to exist between calculated and actual profiles with respect to the magnitude of the induced strain, but not its position from the central axis.

CASE NO.	ACTUAL TENSILE STRAIN	CALC. TENSILE STRAIN	ACT/CALC TENSILE RATIO	ACTUAL COMP. STRAIN	CALC COMP. STRAIN	ACT/CALC COMP. RATIO
1	58.0	60.6	0.96	1.00	19.3	0.05
2	58.0	59.0	0.98	1.00	23.9	0.04
3	58.0	59.1	0.98	1.00	21.0	0.05
4	103.0	91.5	1.12	86.5	69.4	1.25
5	103.0	103.6	0.99	86.5	86.5	1.00
6	103.0	97.5	1.06	86.5	78.0	1.11
7	50.0	58.5	0.85	9.0	19.2	0.47
8	50.0	57.6	0.87	9.0	29.8	0.30
9	50.0	58.0	0.86	9.0	24.6	0.36
10	72.0	66.0	1.09	60.0	56.0	1.07
11	-	-	-	-	-	-
12	41.3	48.0	0.86	31.0	38.9	0.80
13	132.0	127.0	1.04	114.0	110.0	1.04
14	39.0	45.0	0.87	-	-	-
15	72.0	65.0	1.11	58.0	56.0	1.04
16	66.0	60.7	1.09	25.0	21.5	1.16
17	66.0	62.3	1.06	25.0	18.8	1.33
18	66.0	61.4	1.07	25.0	20.0	1.25
19	74.0	69.1	1.07	36.0	38.0	0.95
20	75.0	63.0	1.19	33.0	35.7	0.92
21	100.0	93.5	1.07	66.0	55.7	1.18
22	-	-	-	-	-	-
23	-	-	-	-	-	-

All distances in metres

Table 3.4 Location of the Maximum Tensile and Compressive Strains from Face Centre Line-Transverse Profile

CASE NO.	ACTUAL TENSILE STRAIN	CALC. TENSILE STRAIN	ACT/CALC TENSILE RATIO	ACTUAL COMP. STRAIN	CALC COMP. STRAIN	ACT/CALC COMP. RATIO
1	- 7.6	- 6.4	1.19	-30.4	-55.9	0.54
2	- 7.6	- 7.6	1.00	-30.4	-49.6	0.61
3	- 7.6	- 7.1	1.07	-30.4	-52.0	0.58
4	- 3.0	- 7.9	0.38	-35.7	-36.3	0.98
5	- 3.0	- 7.3	0.41	-35.7	-28.9	1.23
6	- 3.0	- 7.8	0.38	-35.7	-31.9	1.12
7	- 7.6	- 6.5	1.17	-46.9	-54.8	0.86
8	- 7.6	- 8.7	0.87	-46.9	-42.1	1.11
9	- 7.6	- 7.8	0.97	-46.9	-48.9	0.96
10	-	-	-	-	-	-
11	-15.0	- 8.4	1.78	-30.0	-41.2	0.73
12	-	-	-	-	-	-
13	-	-	-	-	-	-
14	-	-	-	-	-	-
15	-	-	-	-	-	-
16	- 4.0	- 6.5	0.61	-49.1	-54.8	0.89
17	- 4.0	- 4.9	0.82	-49.1	-58.9	0.83
18	- 4.0	- 5.6	0.71	-49.1	-56.8	0.86
19	-	-	-	-	-	-
20	-	-	-	-	-	-
21	-	-	-	-	-	-
22	-	-	-	-	-	-
23	-	-	-	-	-	-

All distances in metres

Table 3.5 Location of the Maximum Tensile and Compressive Strains from Face Centre Line-Longitudinal Profile

3.5.3 Strain vs. W/H Ratio

Figure 3.13 shows the National Coal Board graph of S/H multiplier values, Subsidence Engineers Handbook (3), used for producing maximum strains in the w/h ratio range 0.2 - 1.4. Table 3.6 gives these values in tabulated form for the w/h range 0.2 - 5.0, which are taken from Figure 3.13. These are the S/H multiplier values used for predicting calculated profiles.

Figures 3.14 and 3.15 show actual S/H multiplier values for the shallow workings studied (as against predicted S/H values for calculated profiles, Table 3.6 and Figure 3.13). Table 3.7 shows the actual S/H multiplier values in tabulated form. In Figure 3.14, the transverse profile, calculated profile values are seen as straight lines from a w/h ratio of 1.0 onwards. The actual values, however, appear to decrease in magnitude from w/h 0.8 to a minimum near w/h 3.0, after which they again increase to a maximum in the region w/h 4.0 - 4.5. In Figure 3.15, the longitudinal profile, a similar trend is seen as in Figure 3.14, although the multiplier values are in general of a much lower magnitude and do not appear to increase in the w/h range 3.0 - 4.5.

In both Figures 3.14 and 3.15, the scatter of points follows the very general pattern seen in Figure 3.13, until a w/h ratio of 3.0 is reached. A slight upturn then appears to occur, which differs from the predicted values. However, insufficient points exist on which to base a firm conclusion and this should be interpreted as a possible trend requiring further investigation.

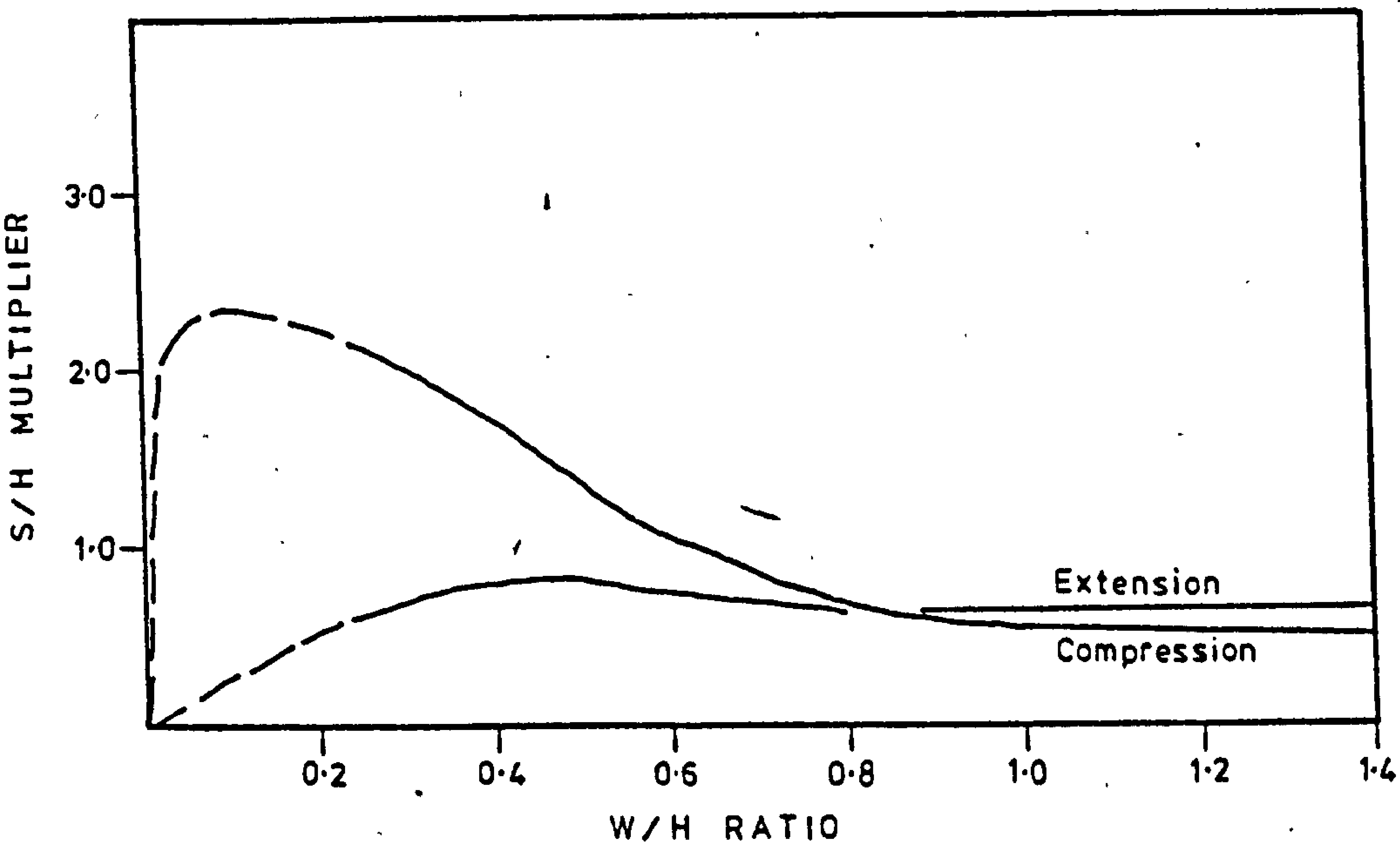


Figure 3.13 NCB S/H Multiplier Values for the Prediction of Maximum Tensile and Compressive Strains (after Subsidence Engineers Handbook (3))

W/H RATIO	TENSILE STRAIN	COMPRESSIVE STRAIN
0.2	0.55	2.25
0.4	0.80	1.70
0.6	0.80	1.05
0.8	0.65	0.70
1.0	0.65	0.55
1.2	0.65	0.50
1.4	0.65	0.50
1.6	0.65	0.50
1.8	0.65	0.50
2.0	0.65	0.50
3.0	0.65	0.50
4.0	0.65	0.50
5.0	0.65	0.50

Table 3.6 NCB S/H Multiplier Values for the Prediction of Maximum Tensile and Compressive Strains

(after Subsidence Engineers Handbook (3))

CASE NO	TRANSVERSE PROFILE TENSILE STRAIN	COMP. STRAIN	LONGITUDINAL PROFILE TENSILE STRAIN	COMP. STRAIN	W/H RATIO
1	0.55	1.33	0.21	0.43	0.87
2	0.46	1.13	0.18	0.36	1.03
3	0.51	1.23	0.20	0.39	0.94
4	0.41	0.31	0.16	0.61	2.50
5	0.32	0.24	0.12	0.47	3.65
6	0.36	0.28	0.14	0.54	3.00
7	0.38	0.50	0.40	0.20	0.91
8	0.26	0.35	0.28	0.14	1.29
9	0.32	0.42	0.34	0.17	1.06
10	0.75	0.67	-	-	3.94
11	-	-	0.02	0.19	4.32
12	0.39	0.70	-	-	3.15
13	0.60	0.86	-	-	4.54
14	0.61	-	-	-	3.00
15	1.00	0.43	-	-	4.33
16	0.53	0.87	0.36	0.35	0.94
17	0.58	0.96	0.40	0.39	0.85
18	0.55	0.91	0.39	0.37	0.89
19	1.15	1.00	-	-	1.38
20	0.80	0.68	-	-	1.43
21	0.80	0.46	-	-	1.53
22	-	-	-	-	1.82
23	-	-	-	-	2.80

Table 3.7 Actual S/H Multiplier Values for the 23 sets of Test Data

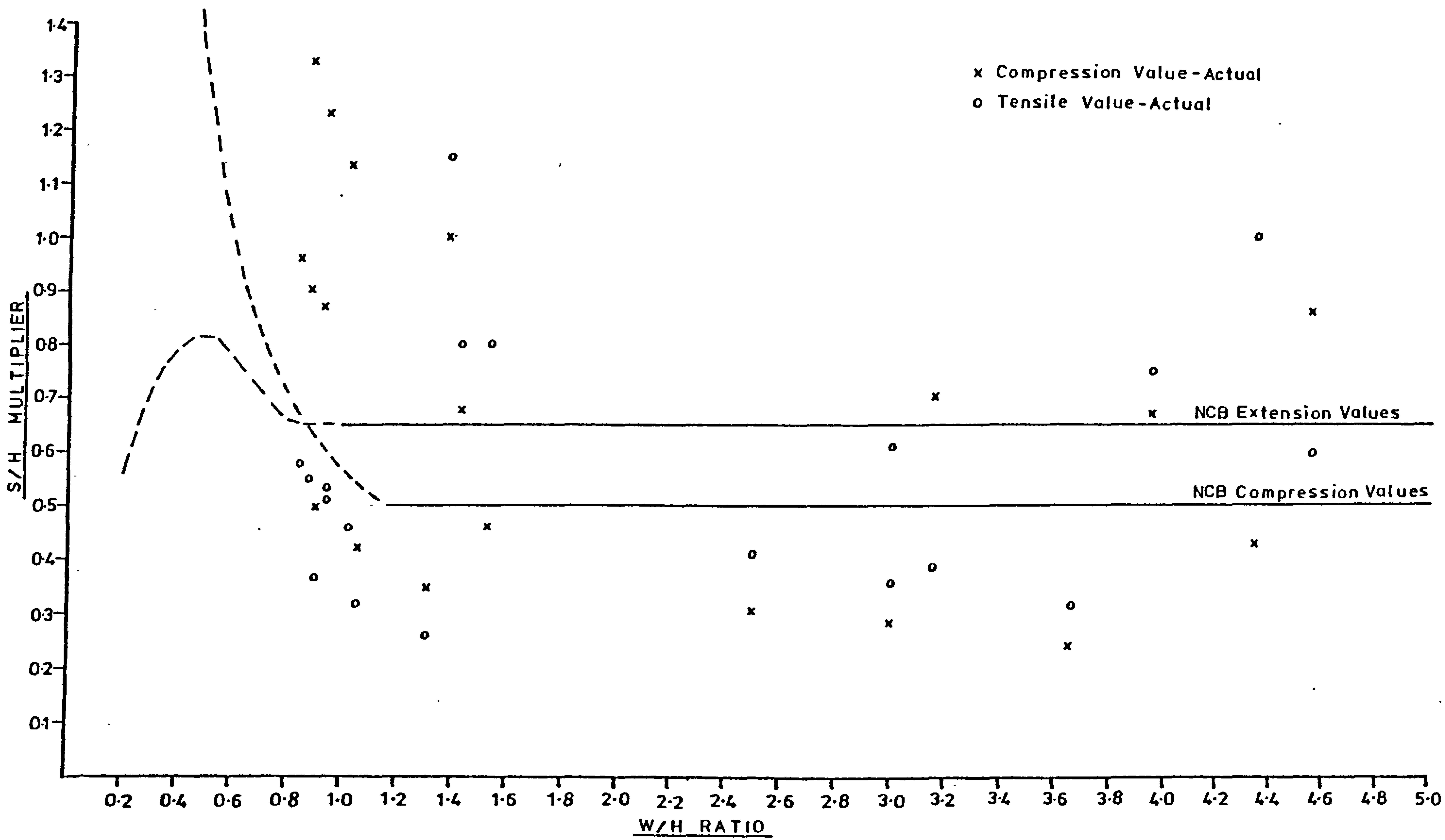


Figure 3.14 Calculated and Actual S/H Multiplier Values for Maximum Tensile and Compressive Strains – Transverse Profile

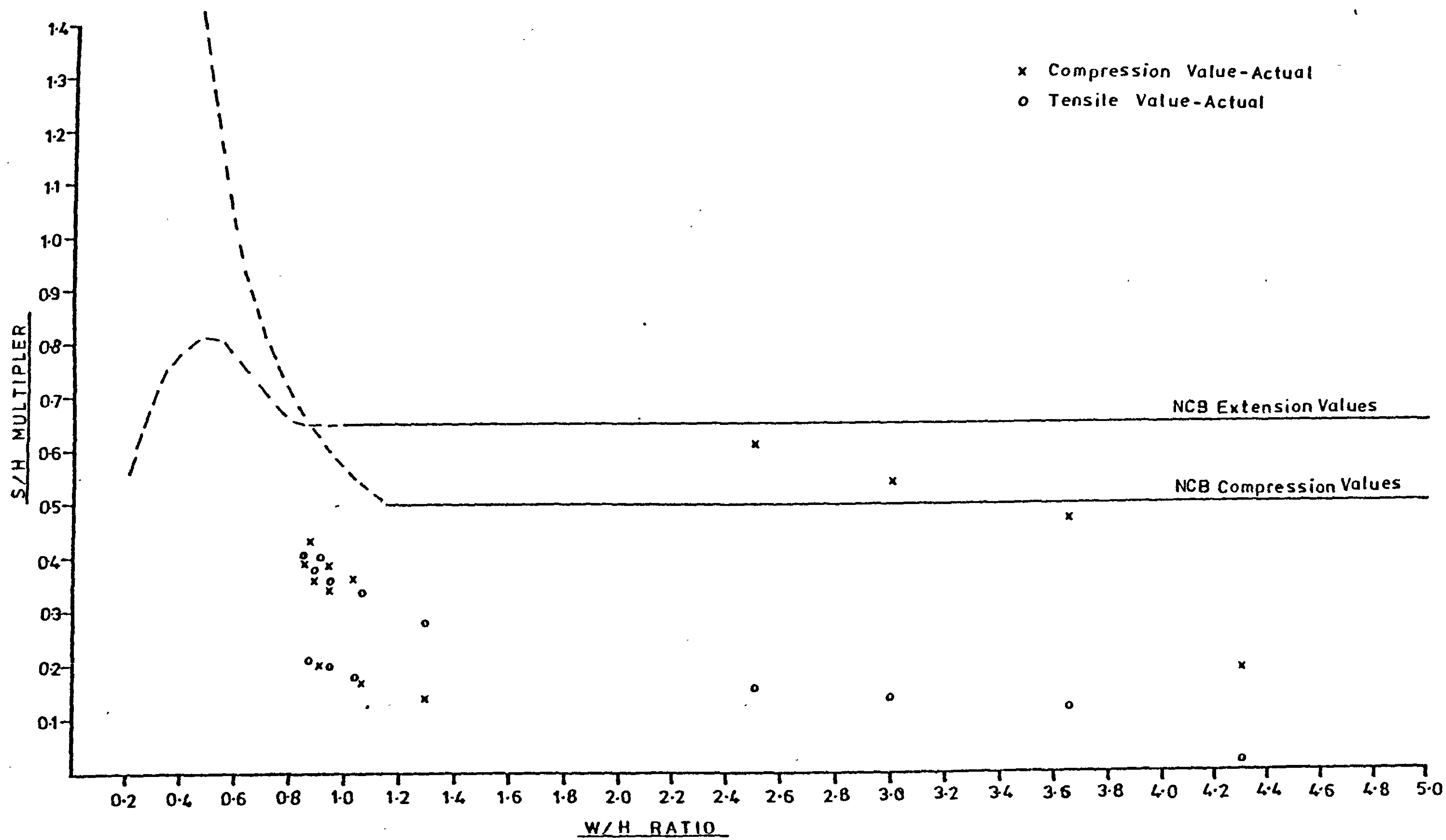


Figure 3.15 **Calculated and Actual S/H Multiplier Values for Maximum Tensile and Compressive Strains – Longitudinal Profile**

3.6 Geological - Subsidence Relationships

Shadbolt (44) and Breeds (48,52) have both stated that the transmission of mining induced displacement to surface is complicated by the intervening geological strata.

Breeds (52) has examined the effects of subsidence development on three principle strata types encountered at surface within the East Midlands Coalfield: Coal Measures, Permian Limestones and Triassic Sandstones. Superficial deposits of alluvial sands and gravels or boulder clay have been excluded.

Figure 3.16 shows maximum subsidence expressed as S/m% against the w/h ratio. Similarly, Figure 3.17 shows induced rib-side subsidence S_{rib}/S% against the w/h ratio. A national curve is given for comparison. In both cases, the magnitude of induced maximum and rib-side subsidence appears independent of surface strata type. However, interpretation of Figure 3.17 could suggest that rib-side subsidence is most fully developed when the surface strata is Triassic sandstone.

In addition to subsidence, the effects of surface geology on maximum tensile and compressive strain were also examined, Breeds (52). Actual S/H multiplier values for case histories within the three strata types were calculated and then compared with the national values. Figure 3.18 shows tensile strain S/H multiplier values against w/h, while Figure 3.19 shows similar compressive strain values. Polynomial curve fitting, Breeds (48, 52) was then carried out for each strata type. In Figure 3.18, a selection of curves are given along with the national curve. The Coal Measures

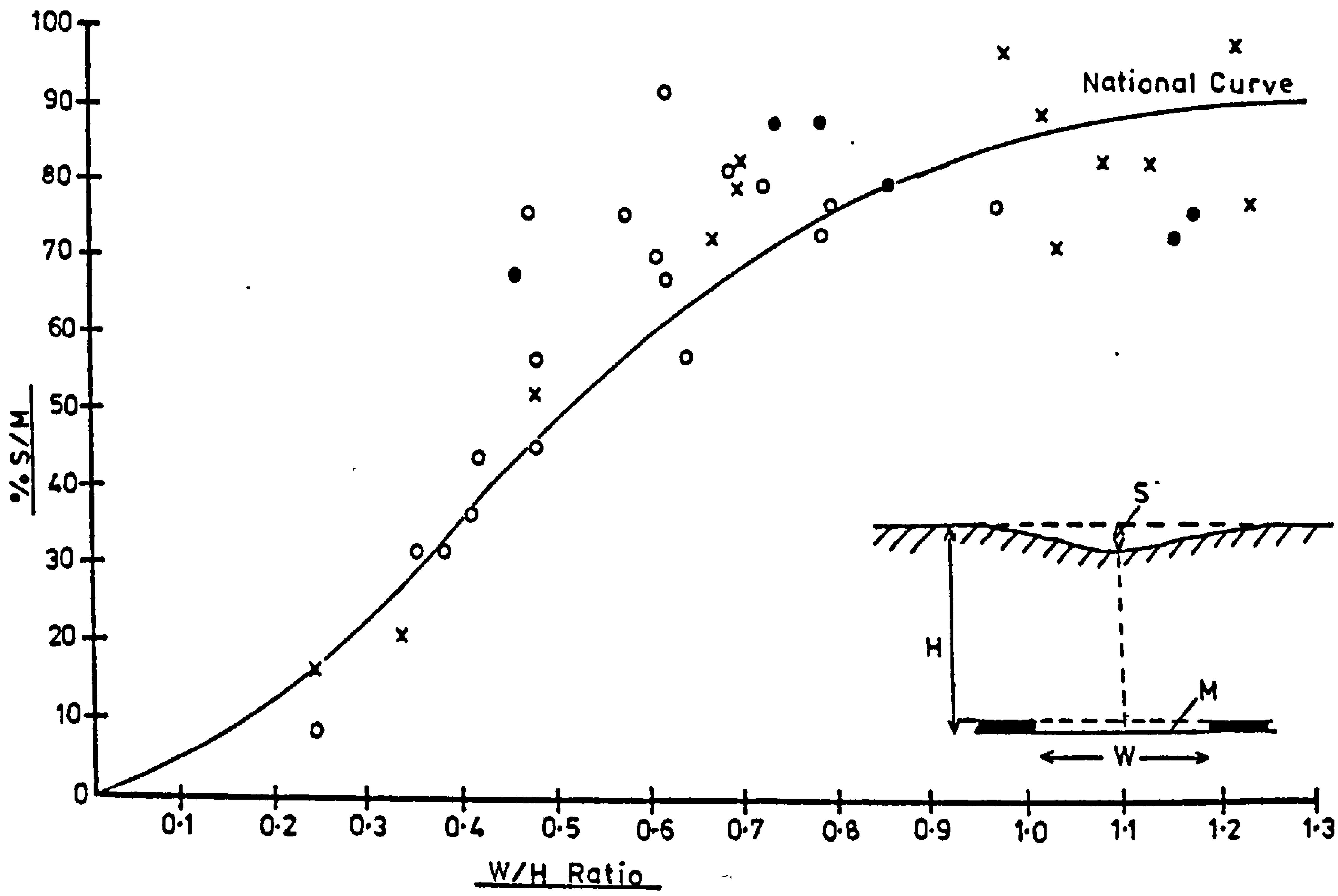


Figure 3.16 Partial Subsidence (%S/M) vs. Width-Depth Ratio (after Breeds (52))

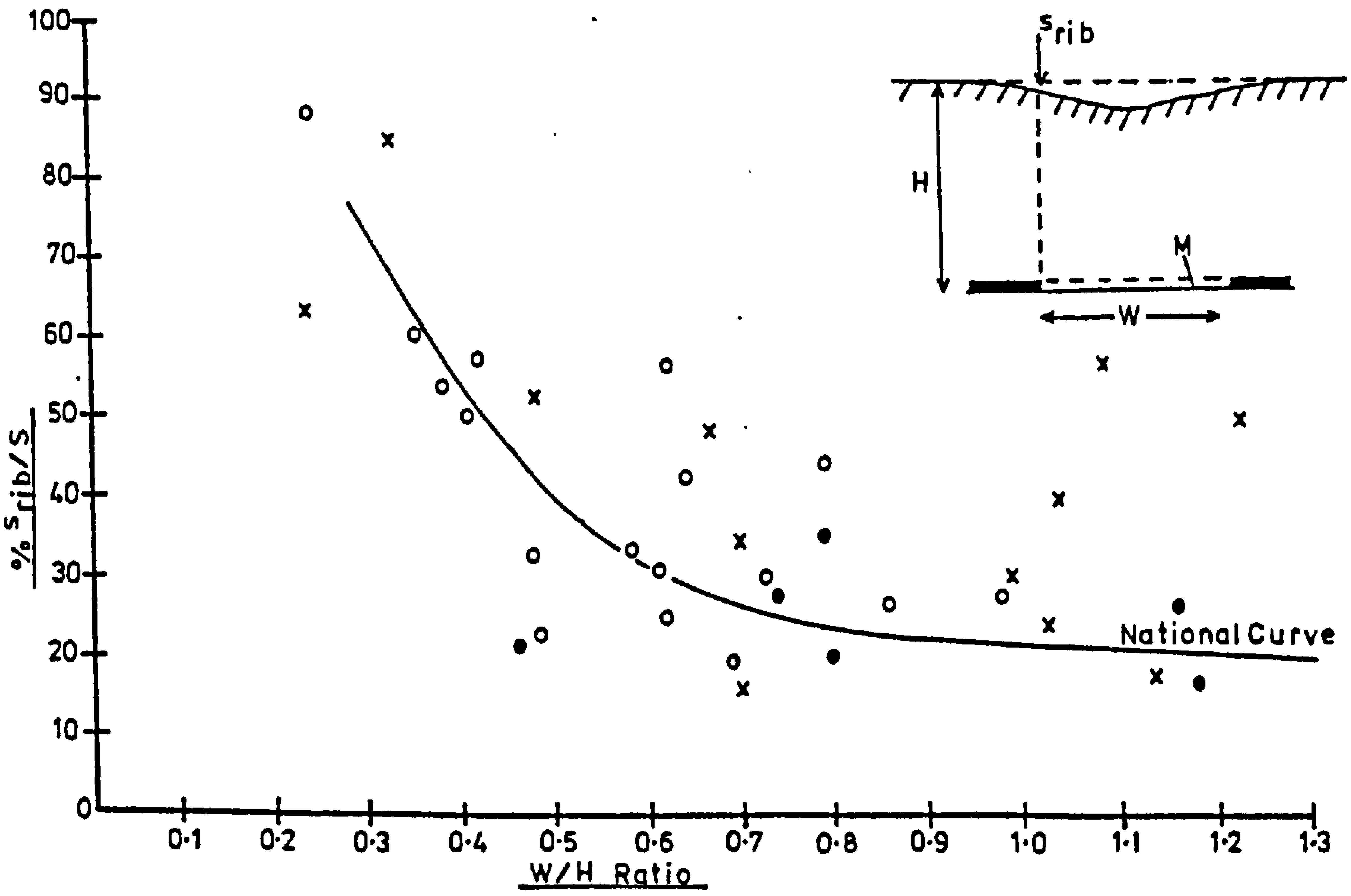


Figure 3.17 Ribside Subsidence ($\%S_{rib}$) vs. Width-Depth Ratio (after Breeds (52))

Key
o - Coal Measures
o - Permian Limestone
x - Bunter Sandstone Trias

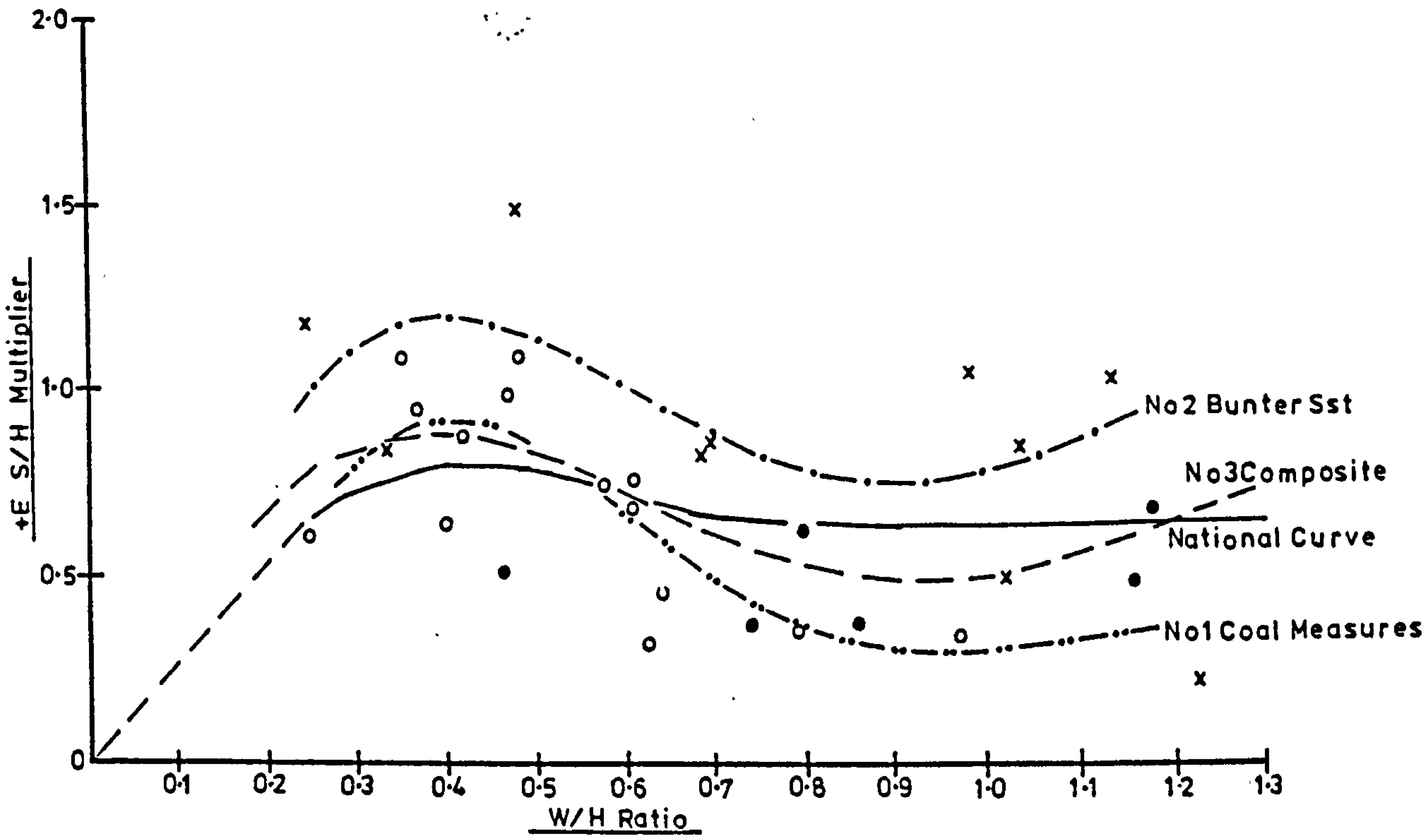


Figure 3.18 S/H Multiplier Values for Maximum Tensile Strain (+E) vs. Width-Depth Ratio
(after Breeds (52))

Key
o - Coal Measures
• - Permian Limestone
x - Bunter Sandstone Trias

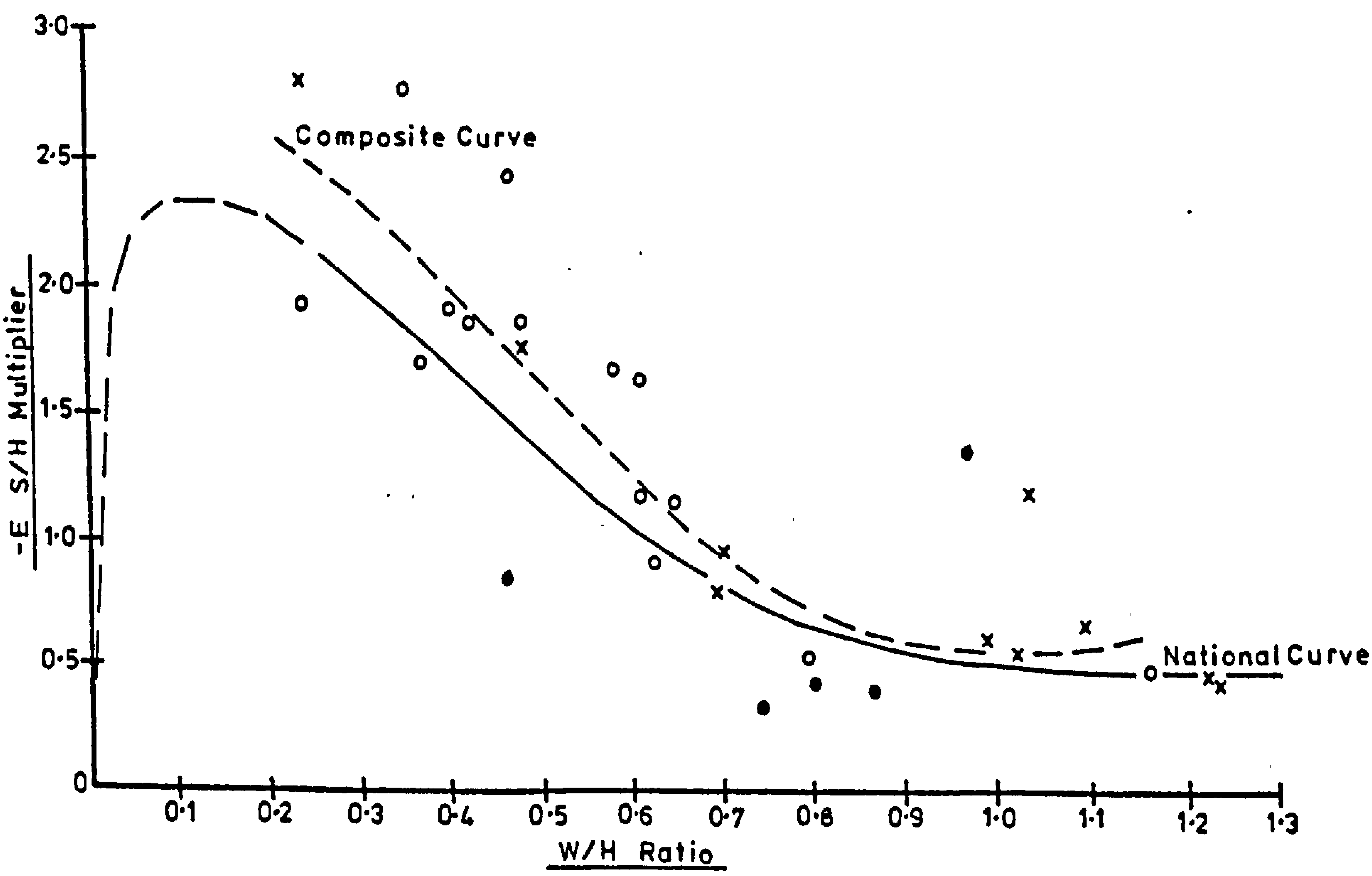


Figure 3.19 S/H Multiplier Values for Maximum Compressive Strain (-E) vs. Width-Depth Ratio
(after Breeds (52))

and Composite curves (Nos. 1 and 3) tend to closely resemble the national curve, while the Bunter Sandstone (No. 2) tends to give consistently higher results. In Figure 3.19, compressive S/H values were found to be unaffected by strata type and only the composite curve is given for comparison.

The development of subsidence and compressive strain is unaffected by surface geology although the magnitude of tensile strain does appear to be affected. In particular, induced tensile strains in the Bunter sandstone appear consistently higher than those in either the Coal Measures or Permian deposits. This is almost certainly due to inherently weak tensile properties of the material and the presence of natural discontinuities.

3.7 Subsidence - Fissure Development

Although work by Breeds (52) has discussed the effects of geological strata on subsidence development, no reference is made to the fact that fissure systems often develop around the rib-side area, in the zone of tensile strain. It was therefore decided to investigate further the relationship between geological strata type and fissure formation.

Work by Forrester (53) into the stability of spoil heaps underworked by mining activity concluded that:

'a zone of active cracking extends upto 0.4 x depth in advance of the face and upto 0.3 x depth outside the rib-side for a constrained area of influence and upto 0.6 x depth outside the rib-sides, if a steep flank slope is being undermined. Cracking can also occur upto 1.2 x depth behind the face.'

This work only considers un- and semi-consolidated materials, such as colliery wastes. Breeds (48) briefly mentions the existance of fissure zones over and around the rib-side area, while the National Coal Board and in particular the East Midlands Subsidence Unit monitors all reported cases of this phenomenon. Therefore, in conjunction with the East Midlands Subsidence Unit, a total of 90 cases were obtained where fissuring had occurred either on or near the rib-side (in the zone of induced tensile strain). However, in only 41 cases had the fissure width been recorded.

3.7.1 Analysis of Case Histories

Due to the difficulty in quantifying surface geology, it was decided to identify each strata type as a definite unit with an assigned arbitrary number. Each strata type could then be directly compared, using graphical techniques, with other parameters: Fissure width, Seam depth and Tensile strain at point of fissuring. The effect of previous workings has not been included due to the lack of quantifiable data.

Figures 3.20 to 3.23 show the results in graphical form and Table 3.8 the discrete units of surface geology.

3.7.1.1 Fissure Width vs. Surface Geology

Figure 3.20 shows Fissure width against Surface geology. In the Triassic, with one exception, only narrow fissures upto 0.3 m wide have been encountered. Similarly, in both recent and man made deposits, only narrow fissures upto 0.3 m wide have been found. However, in the Coal Measures and Permian, a full range of fissure widths upto 1.0 m or above exist.

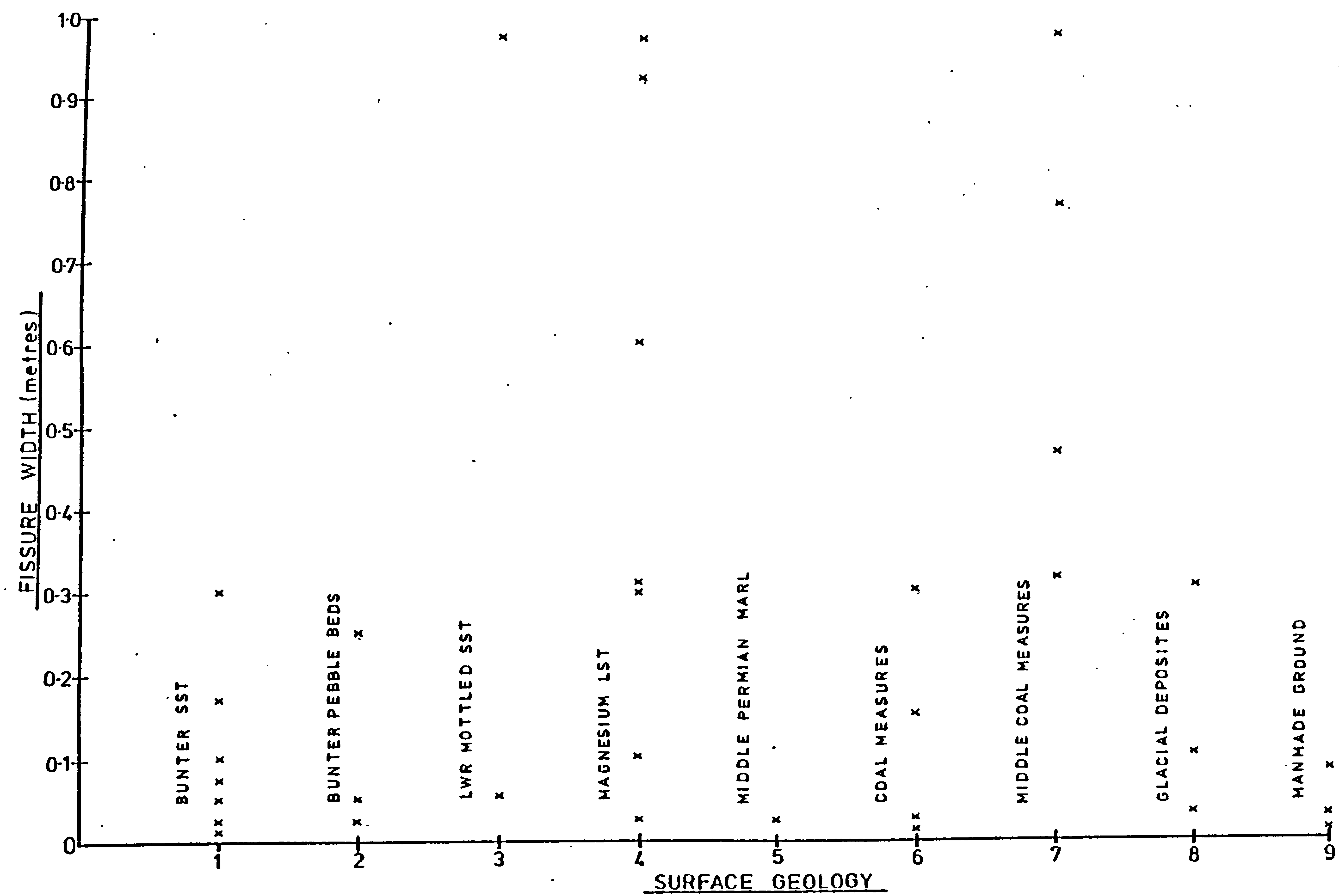


Figure 3.20 Fissure Width vs. Surface Geology, East Midlands Fissure Data

Arbitrary Identification Number	Geological Unit at Surface
1	Bunter Sandstone
2	Bunter Pebble Beds
3	Lower Mottled Sandstone
4	Magnesium Limestone
5	Mid Permian Marl
6	Coal Measures
7	Middle Coal Measures
8	Glacial Deposits
9	Man Made Ground

Table 3.8 Arbitrary Units Assigned to the Surface Geology

Surface Geology	Range of Tensile Strain (mm/m)
Man Made Ground	3 - 12
Glacial Deposits	4 - 15
Triassic	0 - 10
Permian	0 - 7
Coal Measures	0 - 19

Table 3.9 Geological Units with the Range of Tensile Strains Experienced

3.7.1.2 Fissure Width vs. Tensile Strains

Figure 3.21 shows Fissure width against Tensile strain at the point of fissuring. It can be divided into four basic areas. An expected scatter of 'low' tensile strains (upto 12 mm/m) with narrow fissures (upto 0.5 m) is seen in Area 1. Similarly, one would expect that as tensile strain increased, so to would the fissure width, Area 3. However, no points exist in Area 3. The remainder of the points fall into two anomalous zones, Areas 2 and 4. Area 2 represents large fissure widths associated with small tensile strains. This can be associated with deep workings, the presence of old workings or geological factors such as faults or joint networks. Area 4 represents small fissure widths with large tensile strains. Few points are seen in this area, which suggests that it is primarily applicable to geological factors. The intervening strata exhibits sufficient elastic properties to accommodate the large tensile strains generated.

3.7.1.3 Fissure Width vs. Depth of Working

Figure 3.22 shows Fissure width against Depth of working. It can be divided into two specific areas. 'Shallow' workings above a depth of 500 m which produce both wide and narrow fissures, which are due to relatively high tensile strains, the presence of old workings and geological factors. Deeper workings below 500 m, tend to produce less frequent occurrences of fissuring which are generally narrower in width (upto 0.3 m).

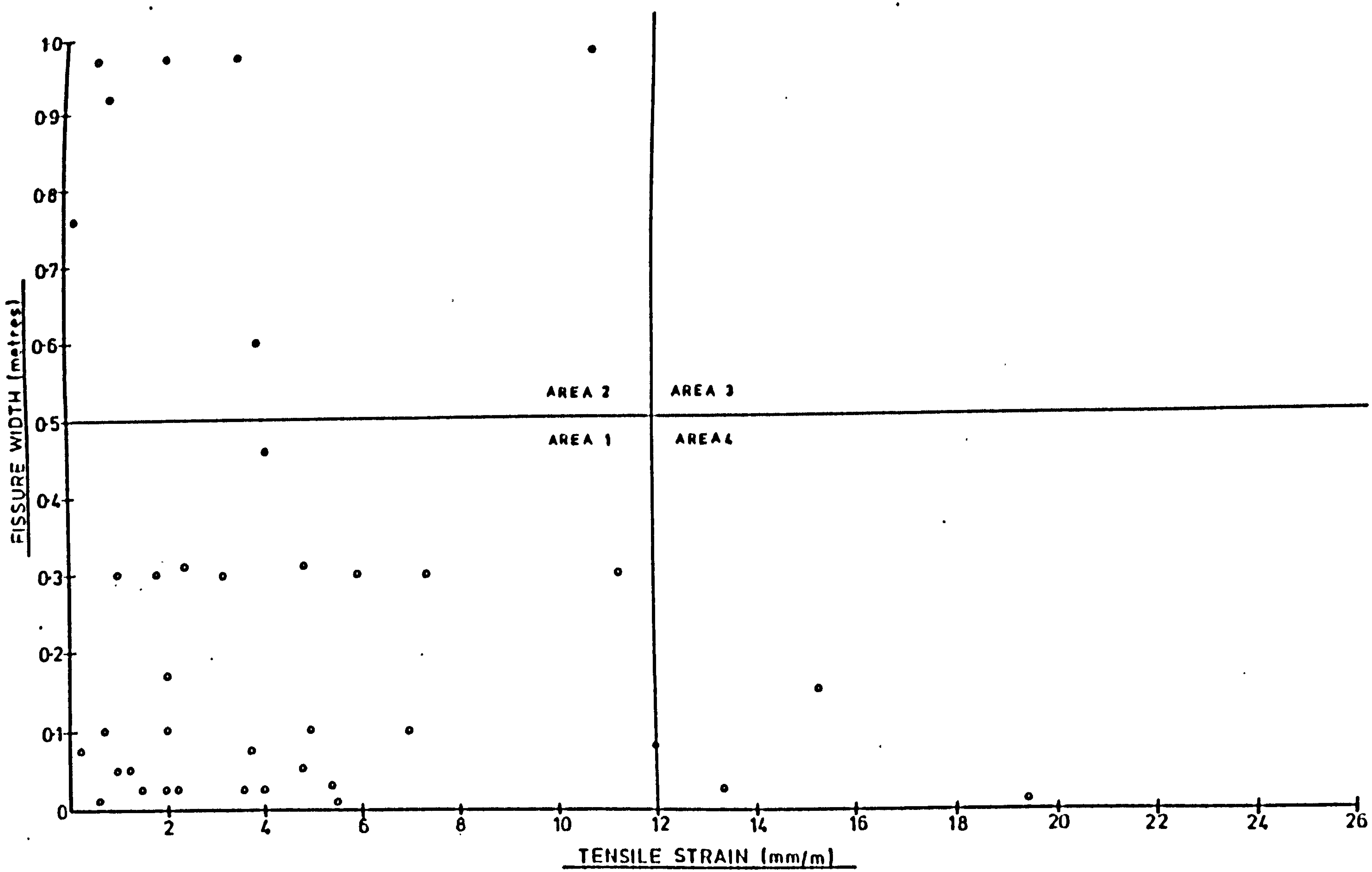


Figure 3.21 Fissure Width vs. Tensile Strain, East Midlands Fissure Data

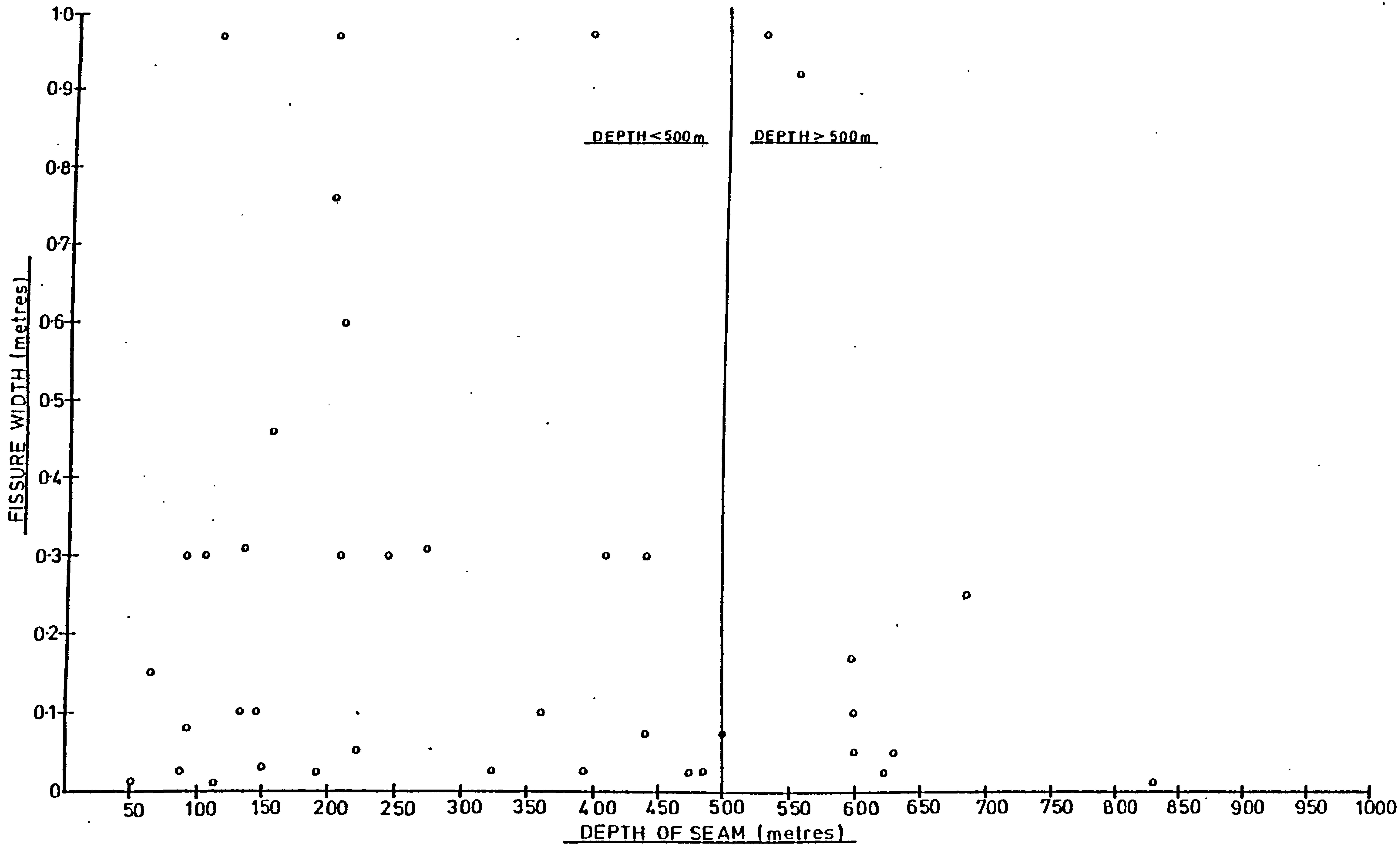


Figure 3.22 Fissure Width vs. Depth of Seam, East Midland Fissure Data

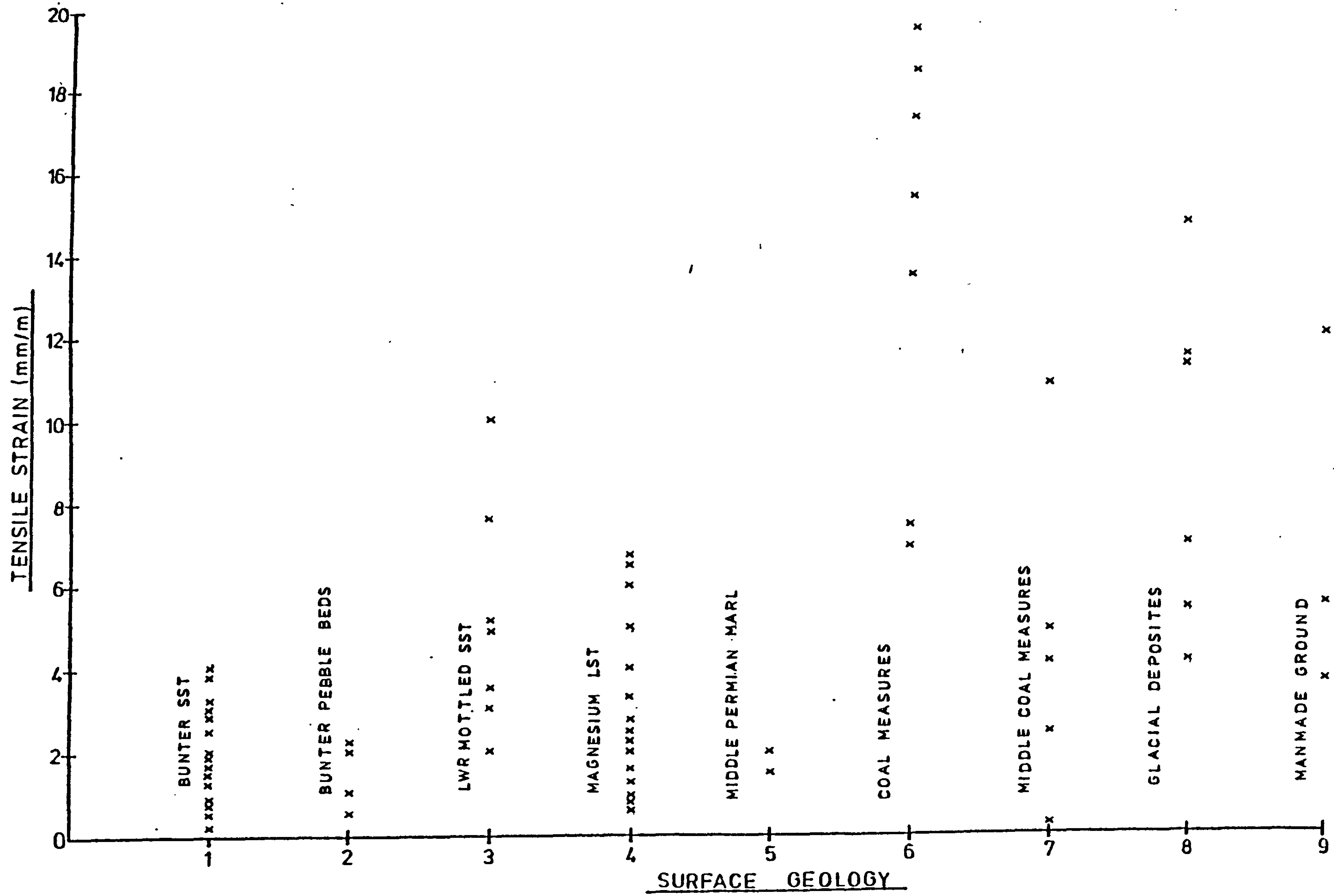


Figure 3.23 Tensile Strain vs. Surface Geology, East Midlands Fissure Data

3.7.1.4 Tensile Strain vs. Surface Geology

Figure 3.23 shows Tensile strain at point of fissuring against Surface geology, which is summarised in Table 3.9. In each case, it appears that strata can accommodate a degree of tensile strain before fissuring starts to occur. However, this will depend primarily on the physical properties of the strata, which includes rock strength and the presence of discontinuities. Fissuring usually occurs along a line of least resistance. Concentration of tensile strain along a fault, jointed zone or horizon of weak rock will therefore allow separation to occur.

The occurrence of fissuring at surface is not dependent purely on the magnitude of the induced tensile strain. Physical properties, such as rock strength and the presence of natural discontinuties play a significant role, which is often seriously underestimated by the mining engineer.

3.8 Conclusion

Consideration of mining activity under shallow and/or different geological conditions, allows certain broad trends to be drawn from the available results. In shallow workings, although the location of points along a profile are not altered, when calculated by the Subsidence Engineers Handbook (3), a difference does exist between the magnitude or actual and calculated subsidence and strain values. Similarly, although surface geology has no affect on the magnitude of either subsidence of compressive strain, it does appear to affect tensile strain. A comparison of fissure formation and tensile strain, reveals no direct link between either surface geology or induced fissure width.

A comparison of shallow workings and the presence of rib-side fissures, suggests that in theory a greater frequency and width of fissuring should occur. Figure 3.22 shows this is not necessarily true. Similarly, since all the cases studied come from the East Midlands, shallow workings will be overlain from seam to surface by Coal Measures strata. A cover of superficial deposits may or may not exist. Figure 3.23 reveals that for Coal Measures strata, a range of tensile strains can produce a range of possible fissure widths. However, the significant feature is not that tensile strain reacts with a strata type, but that it reacts with physical and structural properties of that strata, irrespective of name or type, to produce fissuring.

In each case, all the studied case histories come from the East Midlands Coalfield and reflect local rather than regional trends. It is therefore suggested, that in order to derive empirical methods for the prediction of shallow workings, fissure formation and the behaviour of subsidence profiles under different geological conditions, cases should ideally be collected and analysed from areas throughout the country. Generalised statements and/or conclusions derived from the interpretation of limited data should always be treated with caution.

The author considers that strata behaviour under a combination of shallow and different geological conditions is particularly important when considering undersea mining situations. In addition to the development of tensile strain zones, the possible formation of rib-side fissures should also be considered when assessing potential inflows of water to mine workings. However, only surface-strata effects have been considered in the cases studied. Conditions at strata-strata horizons may not necessarily react in the same way, although surface-strata effects could possibly occur at major horizons of discontinuity such as the Coal Measures-Permian boundary. The final formation of fissure networks, if any, will ultimately depend upon physical and structural characteristics of the individual strata types, rather than tensile strains induced at any given horizon.

CHAPTER 4

REAPPRAISAL OF EXISTING TEST SITE RESULTS

CHAPTER 4

REAPPRAISAL OF EXISTING TEST SITE RESULTS

4.1 Introduction

Extraction of coal by longwall methods results in a redistribution of stress in the surrounding strata. This in turn, causes 'strata failure' on both the micro and macro level with a resultant change in insitu permeability. A change in insitu permeability will affect the migration of fluids, such as methane or water, through the rock mass and into the mine workings. Gradual or rapid changes in these migrations, alters localised mining conditions, which under exceptional circumstances can lead to disasterous consequences.

A research group at Nottingham University has over the past five years conducted a series of investigations, using both field and theoretical techniques, to monitor the insitu permeability changes of British Coal Measures strata. Limited work in this field had previously been undertaken in Great Britain and only isolated studies elsewhere in the world, notably Williamson (54).

The object of this chapter is to examine existing Departmental studies in order to consolidate, rationalise, understand and appreciate that which has already been done, before progressing to further investigations.

4.2 Conceptual Approach

Two main theories have been proposed for the mechanism of permeability change around a longwall extraction. MacPherson (55) proposes that the induced forward abutment stress due to mining causes microfracturing of strata ahead of the face and in particular along bands of inherently weak rock, such as coal seams. This occurs in front of the face, both above and below the working horizon. Induced strain causes partial sealing to occur along the plane of microfracture, thereby reducing permeability to a level below that of the virgin strata.

Other workers have suggested that once microfracturing starts, it intensifies in magnitude with increasing face proximity. In both models, a face position is reached when macrofissure development becomes dominant. However, once extraction has occurred and the stress is relaxed, a large increase in strata permeability occurs, where the degree of relaxation experienced is dependent upon the physical properties of the surrounding strata. Similarly, when caving and initial compaction has occurred, further time dependent compaction of the goaf material should result in a decreased level of permeability, below the maximum attained value, but still above the original virgin value.

4.3 Research Approach

In order to monitor insitu permeability changes around longwall extractions, a series of investigations were made in collaboration with the National Coal Board, Neate (41), Whittaker and Singh (40).

A suitable site was first located and negotiated and a drilling programme initiated to provide boreholes, which could then be suitably instrumented. Subsequent monitoring of the site was then undertaken on a regular basis to assess strata behaviour in relation to panel advance.

The type of instrumentation installed at each site was dependent not only on the parameters to be measured, but also the existing site conditions. A variety of schemes were used, each of which was specially designed for the test site conditions encountered. In each case, the permeability determination test used was one of two types. Firstly, a static head method and secondly a constant rate pressure test. In order to maximise usage of the available boreholes, these were often divided into several test compartments, each of which could provide results for a variety of predetermined horizons. Large amounts of data collected were analysed using computer handling facilities.

A detailed description of the sites, instrumentation, data handling facilities, analysis and conclusions are considered beyond the scope of this work and has already been adequately described by Neate (41) and Whittaker and Singh (40).

4.4 Reappraisal Objectives

Examination of the work by Neate (41, 56) and Whittaker et al (40, 57) reveals perplexing concepts investigated in an original manner along with excellent presentation of both results and conclusions. However, in the authors opinion, optimum use has not been made of the results and in particular only limited interpretation was made of the permeability changes occurring around a longwall extraction. It is therefore intended to re-examine this work in two respects:

1) The change in permeability associated with test cavity geology. No attempt was made by previous authors to correlate and present this data and the author considers this aspect particularly important, since the data contains numerous insitu permeability tests conducted within Coal Measures strata.

2) Draw together work for individual sites, in order to present a comprehensive picture of insitu permeability changes which occur around a longwall extraction.

4.5 Existing Research Sites

Before discussing either of the two objectives outlined in section 4.4, it is firstly considered necessary to describe briefly each of the test sites examined to date. Full details of these sites are given by Neate (41, 56) and Whittaker (40, 57).

The spatial position of each test site in relation to the extraction panel, Table 4.1, can be conveniently divided into three main regions:

1) Strata in Advance of the Face.
2) Strata in the Face End Region
3) Adjacent Rib Pillars.

Site Name	N.C.B. Area	Relationship to Panel	Number of Boreholes	Total number of Test Cavities
Lynemouth	North-East	Ahead of Face Line	2	12
Wentworth	South Yorkshire	Ahead of Face Line	1	4
Annesley	South Nottinghamshire	Face End	5	5
Bagworth	South Midlands	Face End	3	15
Whitwick	South Midlands	Rib Pillar	8	8

Table 4.1 Permeability Investigations undertaken by the Department of Mining Engineering, Nottingham University

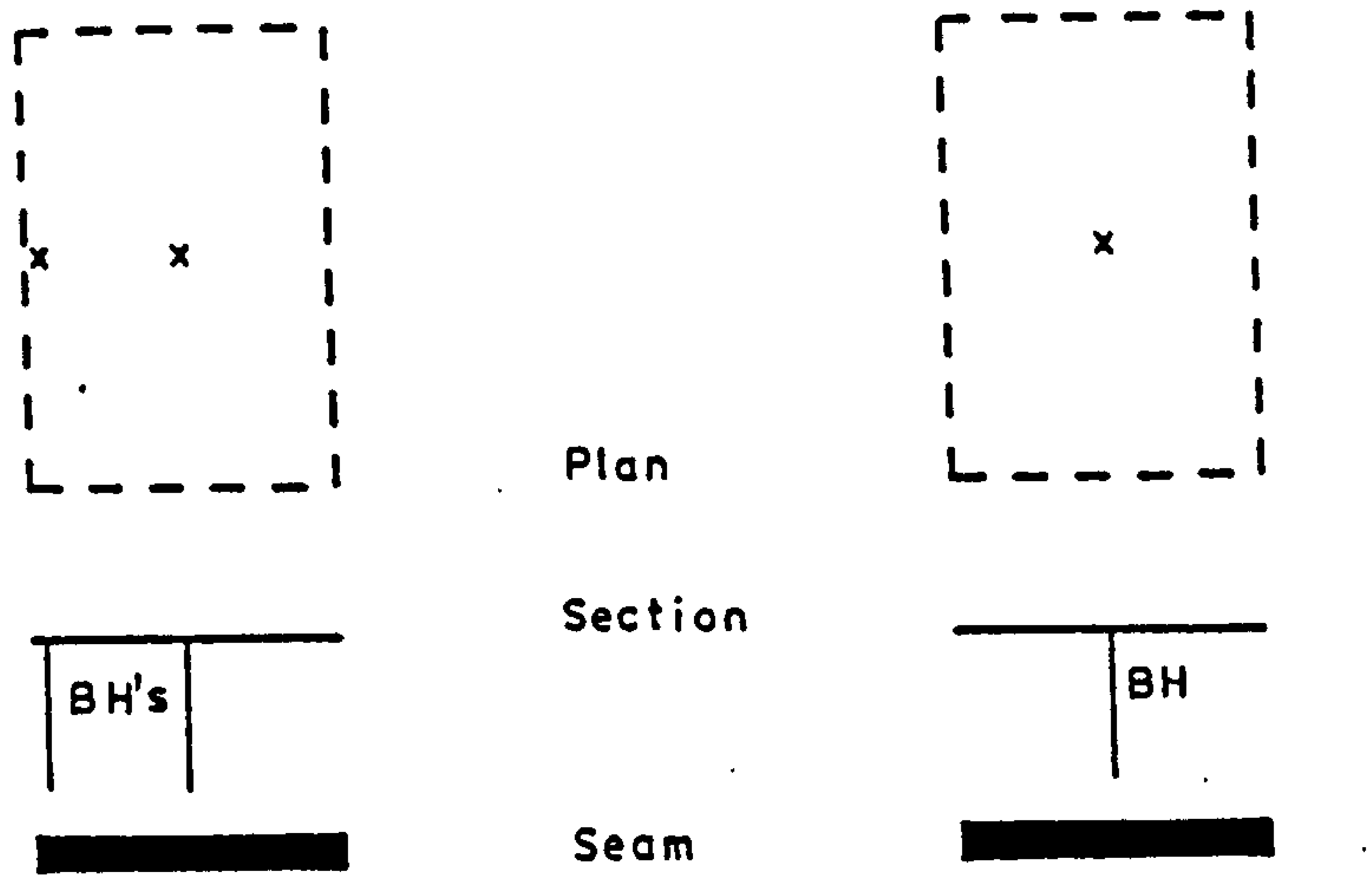


Figure 4.1.1
Lynemouth

Figure 4.1.2
Wentworth

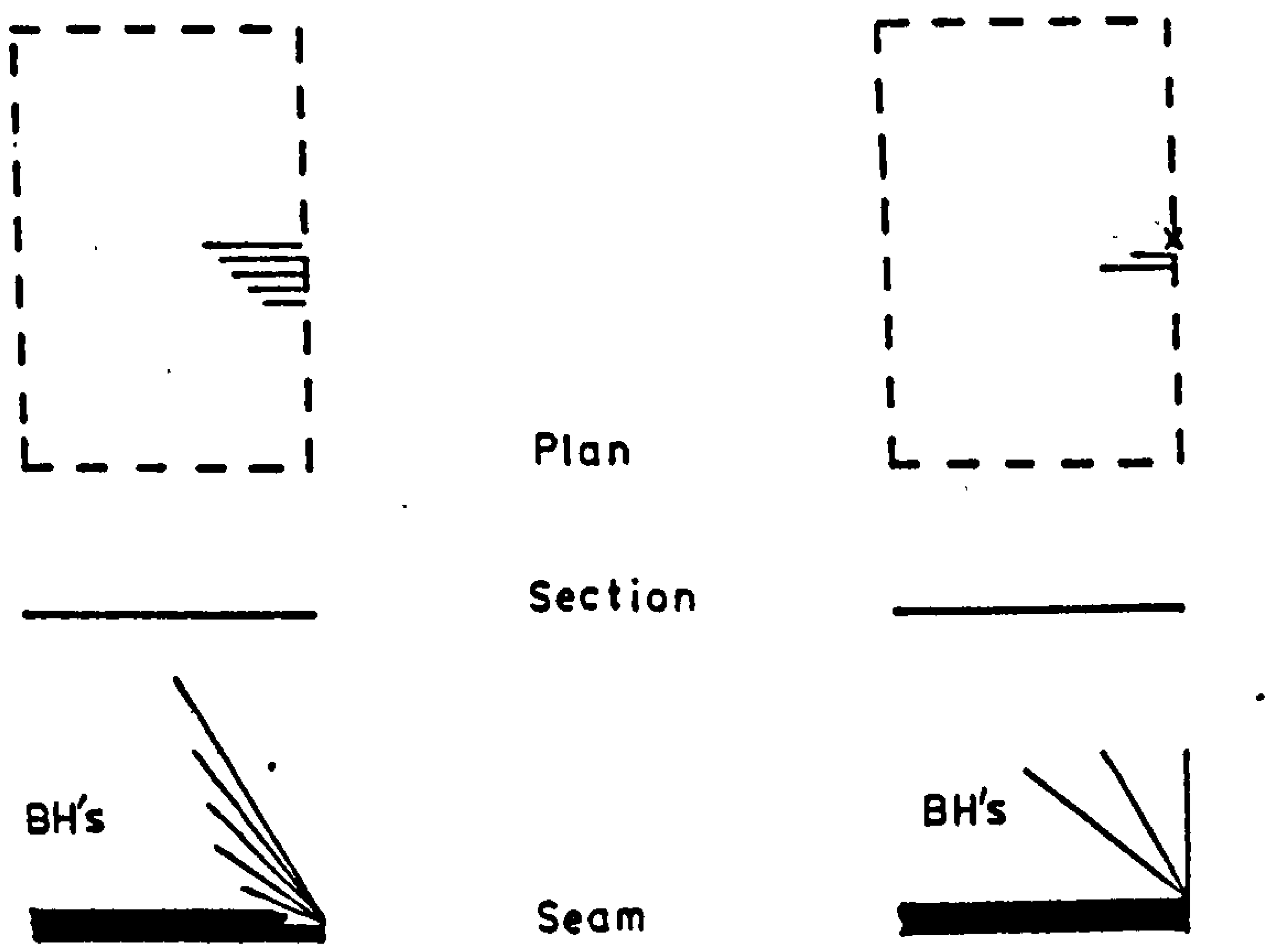


Figure 4.1.3
Annesley

Figure 4.1.4
Bagworth

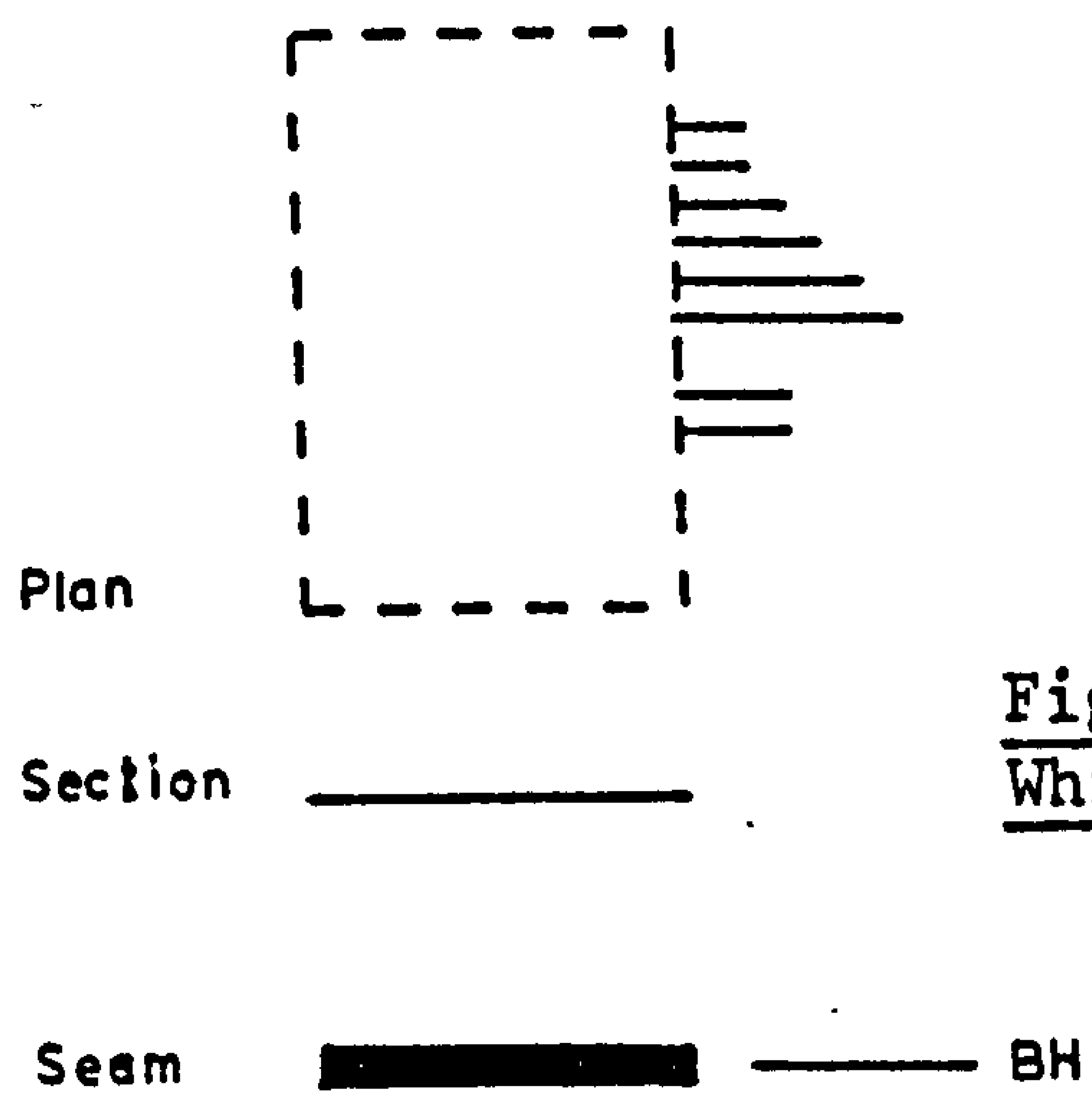


Figure 4.1.5
Whitwick

Figure 4.1 Spatial Position of the Test Sites in relation to the Extraction Panel

4.5.1 Strata in Advance of the Face

At Lynemouth, Figure 4.1.1, a 195 m longwall face with an extraction of 1.3 m in the Brass Thill seam was monitored by two boreholes drilled vertically downwards from extensive room and pillar workings in the overlying Main seam. One hole was sited close to the proposed face centre line and the second near the ribside. Both holes were cored to a depth of 63 m, stopping 11 m short of the Brass Thill seam. An accurate geological record of the intervening strata was obtained at both sites.

At Wentworth, Figure 4.1.2, a 190 m longwall face at 54 m depth and 2.1 m extraction in the Swallow Wood seam was monitored by means of a single borehole from surface. The hole was sited over the proposed face centre line and cored to a depth of 42.7 m. An accurate log of the intervening geological strata was obtained.

4.5.2 Strata in the Face End Region

At Annesley, Figure 4.1.3, a fan of 5 inclined boreholes, sited in a pre-driven gateroad, were placed across a panel of 220 m width and 0.81 m extraction, at a depth of 628 m in the Deep Soft seam. 'Open hole' drilling techniques were used and a single test cavity formed at the end of each borehole. The geological sequence was interpreted from a nearby cored site.

At Bagworth, Figure 4.1.4, 3 holes were sited on a 220 m retreat face with a 2.8 m extraction and at a depth of 150 m in the Five Foot Splent seam. The Minge seam 12.2 m above had already been extracted and the boreholes were therefore limited to a maximum vertical height of 10 m. 'Open hole' drilling was used for the holes, which were respectively inclined at 90^o, 64^o and 45^o to the horizontal and divided into 5 test compartments each.

4.5.3 Adjacent Rib Pillars

At Whitwick, Figure 4.1.5, 8 holes were sited in a rib pillar adjacent to a panel in the Minge seam, depth 70 m and extraction 1.8 m. Each hole was drilled horizontally in-seam and 'open holed'. Variable length holes were used with a maximum penetration of 40 m. A single cavity was created at the end of each hole to allow pressure/flow testing.

4.6 Test Cavity Geology vs Permeability

Table 4.2 lists the test cavity geology found at each site. The section has been derived either by direct examination of core samples or extrapolation of geological data from nearby 'logged' boreholes.

An examination of Table 4.2 reveals that the test site cavities can be divided into three main types:

1) Whitwick - test cavities situated in a coal seam.
2) Lynemouth - of the 12 cavities, 4 are situated in sandstone and the remaining 8 in a composite sandstone, mudstone, and coal seam sequence.
3) Wentworth, Annesley and Bagworth - apart from one Wentworth cavity situated in a sandstone sequence, all the remaining test cavities are situated in a composite sequence of interbedded sandstones, siltstones, mudstones and coal seam/seatearths.

Test Site	Test Cavity No. 1	Test Cavity No. 2	Test Cavity No. 3	Test Cavity No. 4	Test Cavity No. 5	Test Cavity No. 6
Lynemouth No. 1 B'H	Sandstone 6.10*	Sandstone 4.57 Coal 0.38 Mudstone 1.52	Sandstone 4.57 Mudstone 1.52 Coal 0.23	Sandstone 4.57 Mudstone 1.52 Coal 0.68	Sandstone 6.10	Sandstone 7.62 Mudstone 3.05 Coal 0.53
Lynemouth No. 2 B'H	Sandstone 6.10	Sandstone 4.57 Mudstone 1.52 Coal 0.23	Sandstone 4.57 Mudstone 1.52 Coal 0.46	Sandstone 4.57 Mudstone 1.52 Coal 0.38	Sandstone 6.10	Sandstone 7.62 Mudstone 3.05 Coal 0.38
Wentworth	No Permeability readings Composite Geology	Mudstone Sandstone Siltstone	Mudstone Siltstone Coal 0.79	Sandstone	-	-
Annesley	Mudstone Shale Siltstone Mudstone	Siltstone Mudstone Shale	Shale Mudstone Shale Sandstone Mudstone Coal Mudstone Coal Mudstone	Shale Sandstone Shale	Mudstone Shale Sandstone Shale Mudstone	-
Bagworth No. 1 B'H	Siltstone	Sandstone Ironstone Siltstone	Siltstone	Siltstone Mudstone	Mudstone	-
Bagworth No. 2 B'H	Siltstone	Sandstone Ironstone Siltstone	Siltstone	Siltstone Mudstone	Mudstone	-
Bagworth No. 3 B'H	Ironstone Siltstone	Siltstone	Siltstone Mudstone	Mudstone	Mudstone Ironstone	-
Whitwick	All cavities in Coal Ribside Pillar					

* All lengths in metres

Table 4.2 Geology of the Test Cavities

		Lynemouth No. 1 B'H						Lynemouth No. 2 B'H					
		Test Cavity						Test Cavity					
		1	2	3	4	5	6	1	2	3	4	5	6
Face behind site	+ 100	N/A	4.5	12.1	8.0	3.1	4.0	13.6	4.0	9.3	5.2	0.6	5.9
	70 - 100	5.7	12.0	40.1	42.0	69.1	19.1	5.6	1.4	5.3	6.8	2.3	9.8
	30 - 70	7.6	29.9	35.2	51.2	67.4	18.5	11.8	6.7	5.4	22.3	5.7	29.5
	10 - 30	4.8	35.4	25.7	16.3	40.7	18.7	5.5	11.7	15.6	14.2	9.5	16.0
	0 - 10	2.7	48.0	36.0	7.5	24.4	16.6	11.8	8.2	7.4	14.4	9.4	18.8
Face in front of site	20 - 0	-	-	-	-	-	-	-	-	-	-	-	-
	50 - 20	-	-	-	-	-	-	-	-	-	-	-	-
	Consolidation First 4 Days	4.4	25.2	30.2	3.1	6.4	7.2	9.2	10.4	11.1	37.9	9.1	12.4
	Consolidation After 4 Days	8.3	28.8	46.1	23.5	72.9	10.2	23.7	17.3	19.7	38.7	13.1	14.3

		Wentworth				Bagworth												Annesley	
						B'H1		B'H2					B'H3					Boreholes	
		1	2	3	4	1	2	1	2	3	4	5	1	2	3	4		2	5
Face behind site	+ 100	-	-	-	-	0.9	0.8	1.8	2.1	1.4	2.4	3.7	0.9	0.3	0.2	0.2	0.5	-	-
	70 - 100	-	-	-	-	1.6	1.0	24.0	35.0	6.8	5.6	10.9	1.3	0.3	0.4	0.2	0.5	-	-
	30 - 70	-	0.1	0.2	1.0	2.0	16.0	10.2	11.1	2.7	3.0	3.1	0.7	0.2	0.3	0.2	0.3	-	-
	10 - 30	-	0.1	0.1	11.9	2.1	4.3	5.4	1.4	1.5	2.0	1.9	5.6	1.6	1.2	0.8	0.8	1.2	13.7
	0 - 10	-	0.1	0.9	5.7	1.2	17.9	1.2	1.5	0.5	0.7	1.5	-	-	-	-	-	1.0	13.5
Face in front of site	20 - 0	-	-	-	-	26.2	29.2	14.8	10.4	7.6	3.5	31.8	-	-	-	-	-	-	-
	50 - 20	-	4.3	4.2	20.4	26.8	3.4	7.2	33.0	9.3	3.7	1.4	37.0	9.2	13.0	10.0	20.0	1.3	7.0
	Consolidation First 4 Days	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Consolidation After 4 Days	-	6.4	2.3	28.3	-	-	-	-	-	-	-	-	-	-	-	-	1.3	23.5

All permeability values have units of x 10^{-6} cms^{-1}

Table 4.3 Average Permeability Values for Test Sites with respect to a Range of Face Positions

In order to determine whether a relationship between geology and permeability exists, it is first necessary to divide the test cavities into four geological types:

1) Sandstone
2) Sandstone, Mudstone and Coal
3) Coal
4) Composite Sequence.

Mean permeability values can now be derived for a series of arbitrary face positions, Table 4.3, and a comparison undertaken between the various test cavity types. Similarly, values can be inserted onto Table 4.2, enabling comparison between different sites for the same face position. Table 4.4 lists mean permeability values at each site over the range 30 - 70 m (ahead of the face), as well as the test cavity geology. In this manner, values obtained for the range +100 m (ahead of the face) can be used as reasonable indicators of the virgin insitu strata permeability.

4.6.1 Sandstone Test Cavities

Table 4.5 gives tabulated mean permeability values for Lynemouth Borehole No. 1 cavities 1 and 5 (1.1 and 1.5), Borehole No. 2 cavities 1 and 5 (2.1 and 2.5) and the Wentworth No. 4 cavity.

At Lynemouth, Table 4.5 reveals that sandstones surrounding cavity 1 show a higher mean permeability in Borehole 2 than Borehole 1, while for cavity 5 it is Borehole 1 rather than Borehole 2 which exhibits the higher values. Permeability fluctuations in cavity 1, both holes, range between 2.7×10^{-6} cms^{-1} and 1.4×10^{-5} cms^{-1} but

All permeability values x 10^{-6} cms^{-1}

Test Site	Test Cavity No. 1	Test Cavity No. 2	Test Cavity No. 3	Test Cavity No. 4	Test Cavity No. 5	Test Cavity No. 6
Lynemouth No. 1 B'H	Sandstone 7·6	Sandstone Coal Mudstone 29·9	Sandstone Mudstone Coal 35·2	Sandstone Mudstone Coal 51·2	Sandstone 67·4	Sandstone Mudstone Coal 18·5
Lynemouth No. 2 B'H	Sandstone 11·8	Sandstone Mudstone Coal 6·7	Sandstone Mudstone Coal 5·4	Sandstone Mudstone Coal 22·3	Sandstone 5·7	Sandstone Mudstone Coal 29·5
Wentworth	No Permeability readings Composite Geology	Mudstone Sandstone Siltstone 0·1	Mudstone Siltstone Coal 0·2	Sandstone 1·0	-	-
Annesley	Mudstone Shale Siltstone Mudstone -	Siltstone Mudstone Shale -	Shale Mudstone Shale Sandstone Mudstone Coal Mudstone Coal Mudstone -	Shale Sandstone Shale -	Mudstone Shale Sandstone Shale Mudstone -	-
Bagworth No. 1 B'H	Siltstone 2·0	Sandstone Ironstone Siltstone 16·0	Siltstone -	Siltstone Mudstone -	Mudstone -	-
Bagworth No. 2 B'H	Siltstone 10·2	Sandstone Ironstone Siltstone 11·1	Siltstone 2·7	Siltstone Mudstone 3·0	Mudstone 3·1	-
Bagworth No. 3 B'H	Ironstone Siltstone 0·7	Siltstone 0·2	Siltstone Mudstone 0·3	Mudstone 0·2	Mudstone Ironstone 0·3	-
Whitwick	All cavities in Coal Ribside Pillar					

Table 4.4 Test Site Geology with Mean Permeability Values for Face Range 30-70 m.

Face Position (m)	LYNEMOUTH				WENTWORTH
	B'H1		BH2		
	1	5	1	5	4
+ 100	N/A	3.1	13.6	0.6	-
70 - 100	5.7	69.1	5.6	2.3	-
30 - 70	7.6	67.4	11.8	5.7	1.0
10 - 30	4.8	40.7	5.5	9.5	11.9
0 - 10	2.7	24.4	11.8	9.4	5.7
20 - 0	-	-	-	-	-
50 - 20	-	-	-	-	20.4
First 4 Days	4.4	6.4	9.2	9.1	-
After 4 Days	8.3	72.9	23.7	13.1	28.3

All permeability values are x 10^{-6} cms^{-1}

Table 4.5 Permeability Values for the Lynemouth and Wentworth Sandstone Cavities

Face Position (m)	LYNEMOUTH								WENTWORTH
	B'H1				B'H2				
	2	3	4	6	2	3	4	6	3
+ 100	4.5	12.1	8.0	4.0	4.0	9.3	5.2	5.9	-
70 - 100	12.0	40.1	42.0	19.1	1.4	5.3	6.8	9.8	-
30 - 70	29.9	35.2	51.2	18.5	6.7	5.4	22.3	29.5	0.2
10 - 30	35.4	25.7	16.3	18.7	11.7	15.6	14.2	16.0	0.1
0 - 10	48.0	36.0	7.5	16.6	8.2	7.4	14.4	18.8	0.9
20 - 0	-	-	-	-	-	-	-	-	-
50 - 20	-	-	-	-	-	-	-	-	4.2
First 4 Days	25.2	30.2	3.1	7.2	10.4	11.1	37.9	12.4	-
After 4 Days	28.8	46.1	23.5	10.2	17.3	19.7	38.7	14.3	2.3
Coal seam thickness (m)	0.38	0.23	0.38	0.53	0.23	0.23	0.38	0.15	0.79

All permeability values are x 10^{-6} cms^{-1}

Table 4.6 Permeability Values for the Lynemouth and Wentworth Sandstone, Mudstone and Coal Seam Cavities with no discernable trend seen ahead of the face. However, in cavity 5, Borehole 1 shows an increase over the base value once face proximity becomes less than 100 m, while in Borehole 2 although a similar increase is seen it is of lesser magnitude.

In both boreholes, the initial 4 day consolidation value is generally higher than base value, but not necessarily the maximum range value. However, after 4 days consolidation, all values are greater than both the base and maximum range values. It is worth noting that although the site was not undermined, advance ceased 7 m in front of the site, readings were continued in order to monitor consolidation effects.

At Wentworth, Table 4.5, a fluctuation in values is seen with face position, although once the site has been undermined and consolidation effects begin to occur, the permeability increases in a similar manner to that seen at Lynemouth.

A comparison of Wentworth and Lynemouth sandstone cavities therefore reveals similar changes in permeability magnitude, 10^{-6} to 10^{-5} cms^{-1}, once the face has either undermined the site or come into close proximity with it.

4.6.2 Sandstone-Mudstone-Coal Cavities

Detailed logging of the Lynemouth and Wentworth boreholes has provided a unique opportunity to examine permeability changes associated with test cavities intersected by coal seams. In each case, with the exception of Lynemouth No. 6 and Wentworth No. 3, the mean cavity length was 6 m. The Lynemouth Nos. 2, 3 and 4 cavities have a typical geological sequence with upto 4.5 m sandstone and 1.3 m mudstone, in which are contained coal seams of variable thickness up to a maximum of 0.6 m. Table 4.6 lists mean permeability values for a range of arbitrary face positions along with the thickness of coal found in each cavity.

A comparison of mean permeability values for Lynemouth Boreholes 1 and 2, Tables 4.3 and 4.6, reveals fluctuations not only with face proximity but also between cavities. However, no apparent overall trend can be seen. Similarly, a comparison of Lynemouth values against a similar Wentworth cavity, reveals that Wentworth exhibits a much lower range of permeability 10^{-7} to 4.2×10^{-6} cms^{-1} compared with 4.5×10^{-6} to 4.8×10^{-5} cms^{-1} at Lynemouth.

The thickness of coal within monitored test cavities varied between 0.15 and 0.53 m, Table 4.6. A comparison of permeability values between sandstone-mudstone-coal and pure sandstone cavities reveals no difference in the magnitude of recorded values, 10^{-6} to 10^{-5} cms^{-1}. However, cavities with coal seams do seem to exhibit slightly larger changes than those without.

It was initially thought that the presence of coal within a test cavity, might effect the magnitude of recorded permeability values. Similarly, the greater the seam thickness, the greater the change. However, careful examination of the results reveals no indication of either trend and no conclusive evidence to support the argument that when weak strata is subjected to induced stress, either an increase or decrease in permeability occurs. Wentworth No. 3 cavity shows a good example of this argument. The permeability values are lower than those seen at Lynemouth, even though it is intersected by a coal seam thicker than any occurring at Lynemouth: 0.79 m compared with a maximum of 0.53 m.

It was also proposed that the exact location of the seal within the borehole, would govern the amount of coal exposed to testing. In Lynemouth cavity 1.4, 0.38 m of coal is recorded in the cavity, but a further 0.61 m may be exposed in the cavity top, depending on seal location. Similarly, in Lynemouth 1.6, 1.5 m above the test cavity is a further 2.2 m of Yard seam coal. Further examples also exist within the Lynemouth No. 2 borehole. However, no evidence can be found to suggest that any of the seams in close proximity to the test cavities has affected the recorded permeability.

4.6.3 Composite Geology Cavities

In each of the composite geology test cavities: Bagworth, Wentworth No. 2, Annesley No. 2 and 5, the permeability values are all lower than those seen in either the sandstone or sandstone-mudstone-coal cavities.

An examination of the values in Table 4.2, reveals a fluctuation not only with arbitrary face position, but also between cavities. No overall trend is observable, except that values obtained in close proximity to the face are greater than recorded base values. A general range of composite cavity permeability values exists between 10^{-7} and 10^{-6} cms^{-1}.

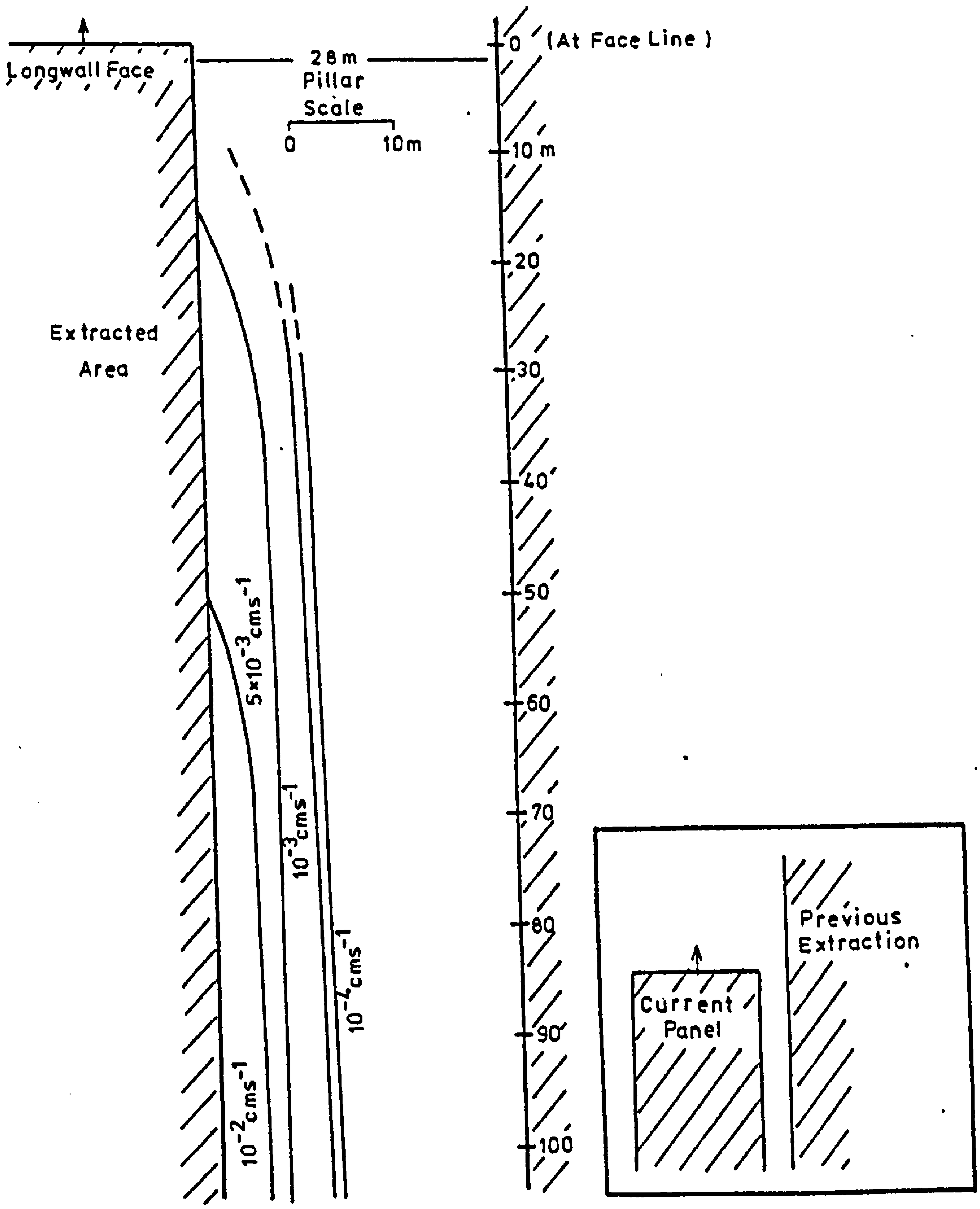


Figure 4.2 Iso-line Permeability Values for a Ribside Coal Pillar, Whitwick Colliery (after Whittaker and Singh (40))

4.6.4 Coal Cavities

It is worth mentioning that before being used to determine permeability, water infusion of coal seams had long been used for dust suppression purposes, Chandler and Hotchkiss (58).

Figure 4.2, shows iso-line permeability values for a coal rib-side pillar, after Whittaker and Singh (40). It can be seen that changes in permeability at any point within the pillar are governed by face proximity to the test site and therefore occur under dynamic conditions. General insitu permeability values of greater than 10^{-2} cms^{-1} for the first 5 m of rib pillar decrease to between 10^{-2} and 10^{-4} cms^{-1} for the next 10 m, after which point they become progressively less.

4.6.5 Discussion on Coal Measures Permeability

Apart from readings in the ribside coal pillar, the highest permeability values and fluctuations are seen in the sandstone-mudstone-coal cavities and these become progressively less through the sandstone and composite cavities. The question therefore arises, how can these changes be related to the test cavity geology?

All Coal Measures strata exhibit a variable lithology both sequentially and spatially, which is usually intersected by either micro or macro level discontinuities. A test cavity therefore exhibits three dimensional anisotropy based on variable lithology, each sequence of which exerts a combination of intergranular and fracture permeability, Figure 4.3.

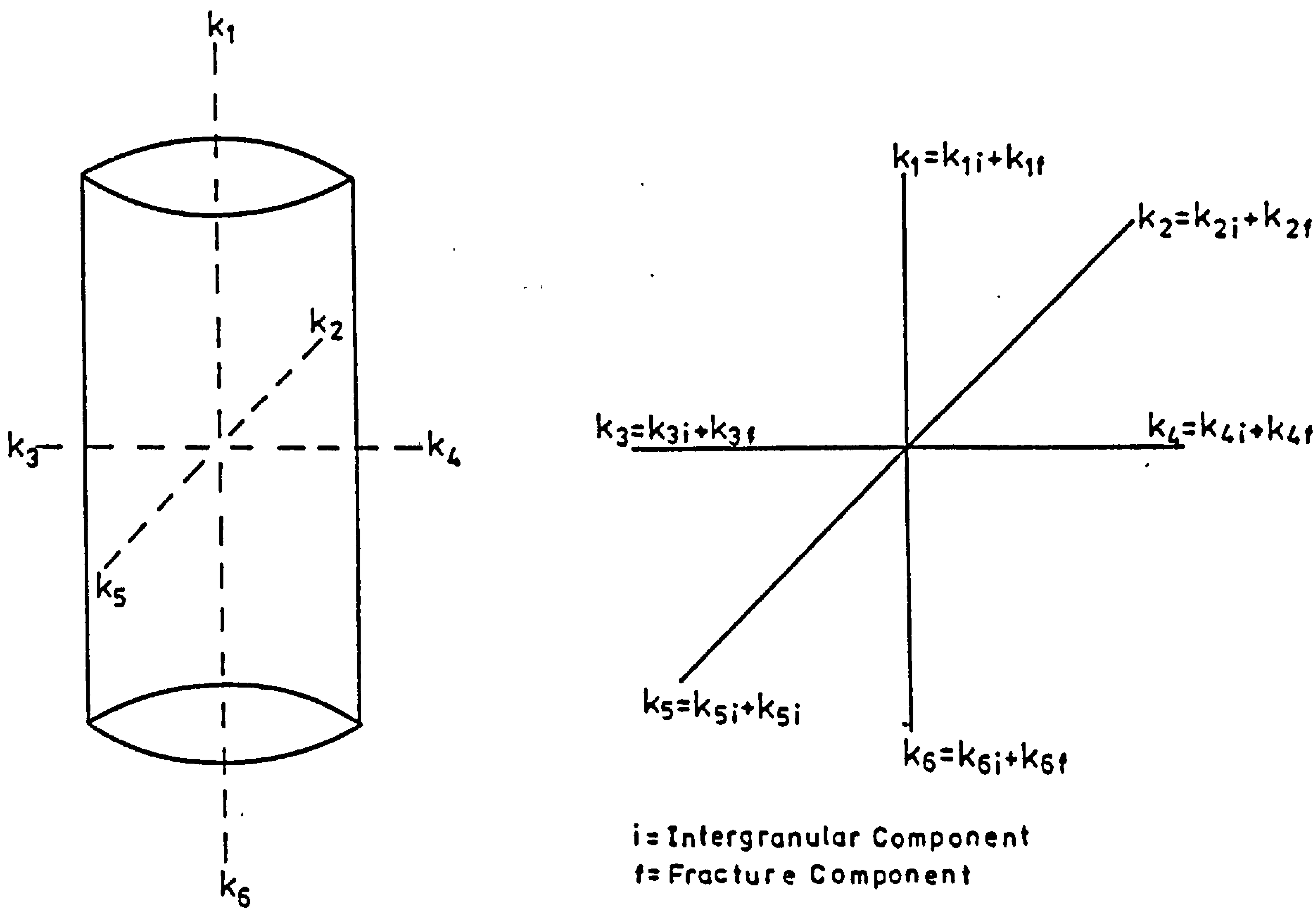


Figure 4.3 Test Cavity exhibiting Three-Dimensional Anisotropic Permeability

If a test cavity is now subjected to dynamic conditions, such as increasing face proximity, a change will occur in the relationship between intergranular and fracture permeability. A sandstone cavity under virgin strata conditions, will exhibit a much higher proportion of intergranular rather than fracture permeability provided it is not intersected by significant discontinuity systems. However, once it becomes influenced by induced stresses due to mining, separation begins to occur along planes of inherent weakness and the fracture component increases significantly. This model can now be related to the sandstone and sandstone-mudstone-coal cavities and used to explain the increasing, though fluctuating permeability values seen with increased face proximity and the marked increase apparent once the face has passed.

Low permeability values seen in composite cavities are primarily due to the presence of mudstones, siltstones and shales, all of which have very low transmissive properties unless fractured. Under normal conditions these exhibit either aquitard or aquiclude properties. The proposed model derived in Figure 4.3, can also be applied to composite cavities, but since the intergranular and fracture components of virgin strata are both very much lower when compared with 'sandstone' cavities, the resultant permeability is also much lower. It should be noted that many of the mudstone, shale and siltstone sequences contain at least a small quantity of clay minerals, which can swell when in contact with water. This in turn can produce a self sealing effect around the test cavity walls, unless either 'washing out' occurs or sufficient induced stress is exerted to rupture the seal.

4.7 Face Advance vs Permeability

It is now proposed to examine permeability changes associated with three areas of strata deformation encountered around a longwall extraction:

1) Permeability ahead of the face.
2) Permeability at the face end.
3) Permeability in the ribside pillar.

In the authors opinion, permeability changes which occur around a longwall extraction ought to exhibit a positive trend associated with face proximity. Neate (41), has analysed results using various techniques, but it was decided by the author to re-examine these and look for trends rather than attempt an interpretation based on individual or consecutive readings. This avenue of investigation is considered particularly important since Neate (41) states that:

'a generally practical approach has intrinsic problems, since with finite resources it is not possible to make tests on a large number of sites. With the practical approach therefore, one is usually placed in the position of trying to infer general conclusions from specific results.'

It is therefore inferred that no matter how accurate individual results may be from a small number of sites, the overall interpretation must be treated with care.

The difference between consecutive readings was calculated and cumulative sum values obtained, which were then used to plot a series of cumulative difference graphs for various sites. Fluctuations in the original readings have now been smoothed and the results examined to determine whether a positive or negative permeability change is associated with increased face proximity to the test site. Using this technique, a series of hypothetical fluctuating test results which exhibit a straight line cumulative sum will indicate that no overall change in permeability has occurred with increased face proximity.

4.7.1 Permeability Ahead of the Face

To date, two sites have monitored permeability changes ahead of the face.

At Lynemouth, cumulative difference graphs Figures 4.4 - 4.9, show that until the face comes within 70 m of the site strata permeability remains remarkably consistent in all test cavities, even allowing for minor fluctuations. These values can therefore be used to estimate a base or virgin insitu strata permeability. However, once 70 m is exceeded a general increase in permeability occurs, with further significant fluctuations at 30 and 10 m. Although the face line stopped 7 m in front of the site, readings were continued to monitor consolidation effects and a slow increase in permeability can be seen in all cavities. This suggests that time dependent effects are still occurring and that the strata is adjusting to stress redistribution.

Fluctuations in permeability indicate that strata behaviour is occurring in discrete units rather than as a typically elastic deformation. This will be due to the variable lithological and structural properties of the different rock types. Neate (41), observed that changes first began to occur at the top of the site and move downwards as the face advanced. Similarly, it was also shown that the onset of change appeared independent of strata type, although this only indicated when the change started to occur, rather than its actual magnitude. Both these observations can be directly related to the development of a typical subsidence profile.

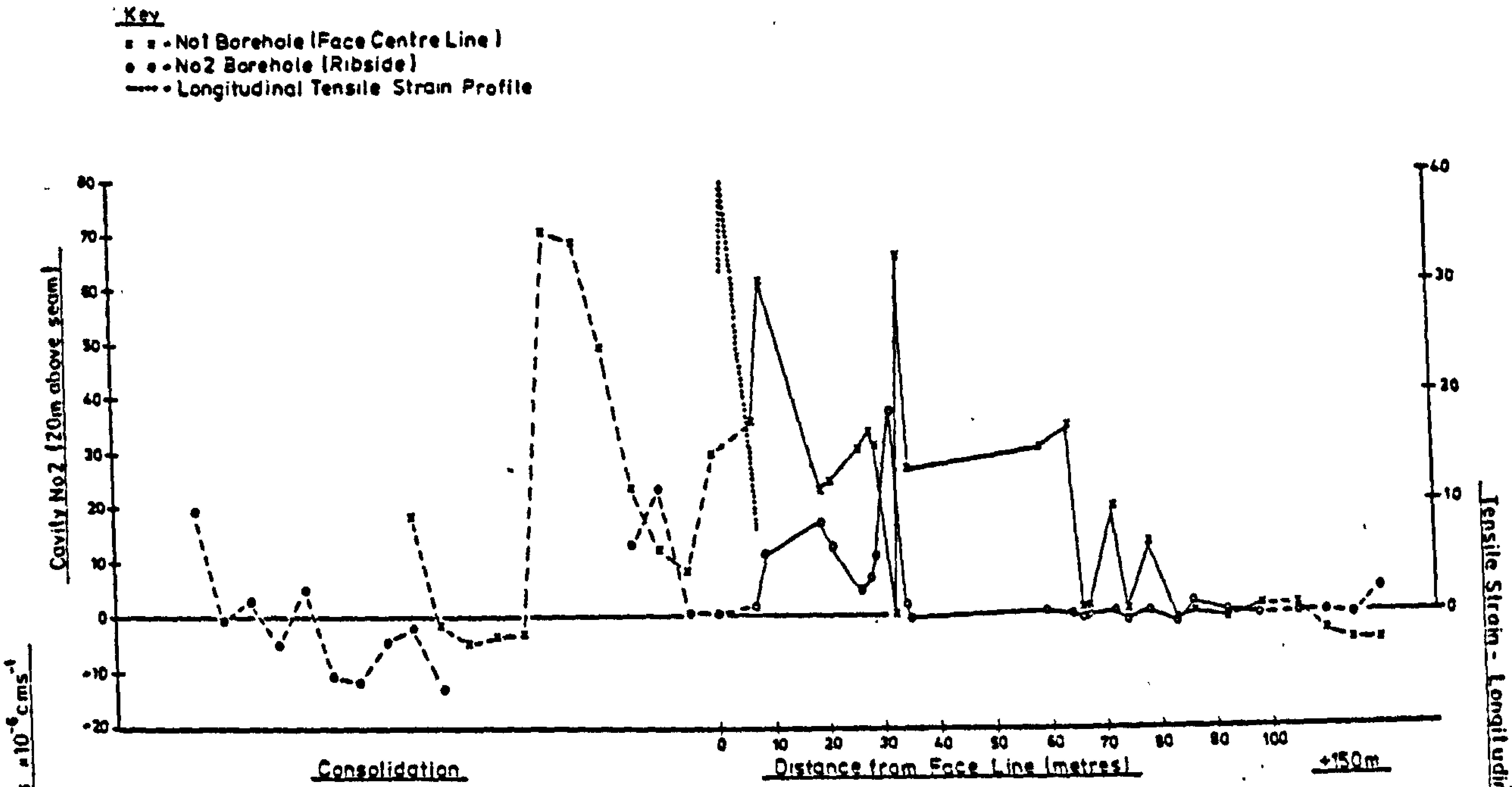


Figure 4.5

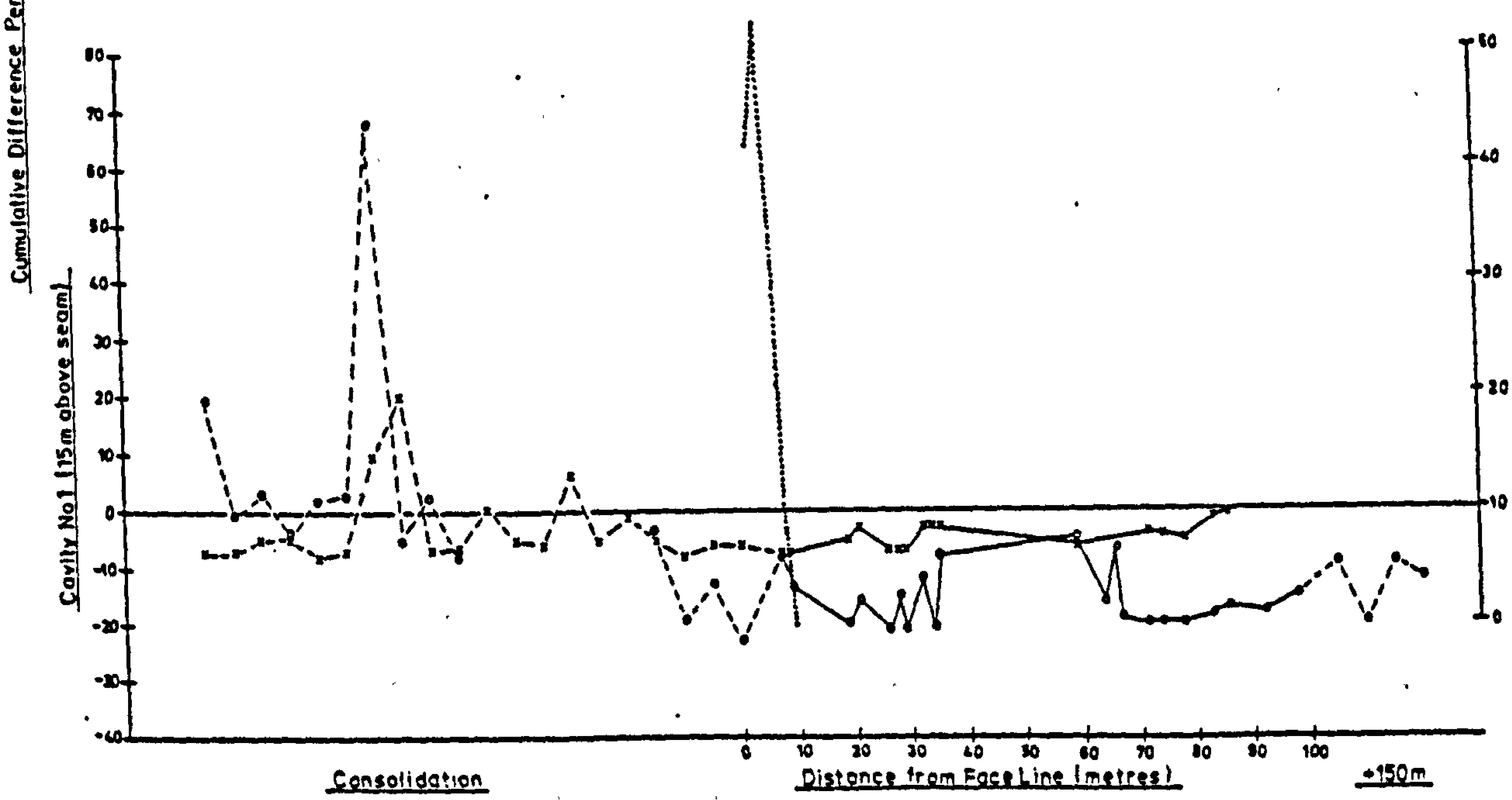


Figure 4.4

Figures 4.4 and 4.5 Cumulative Difference Permeability Values for Lynemouth Nos. 1 and 2 Boreholes, Cavities 1 and 2 against Distance from Face Line

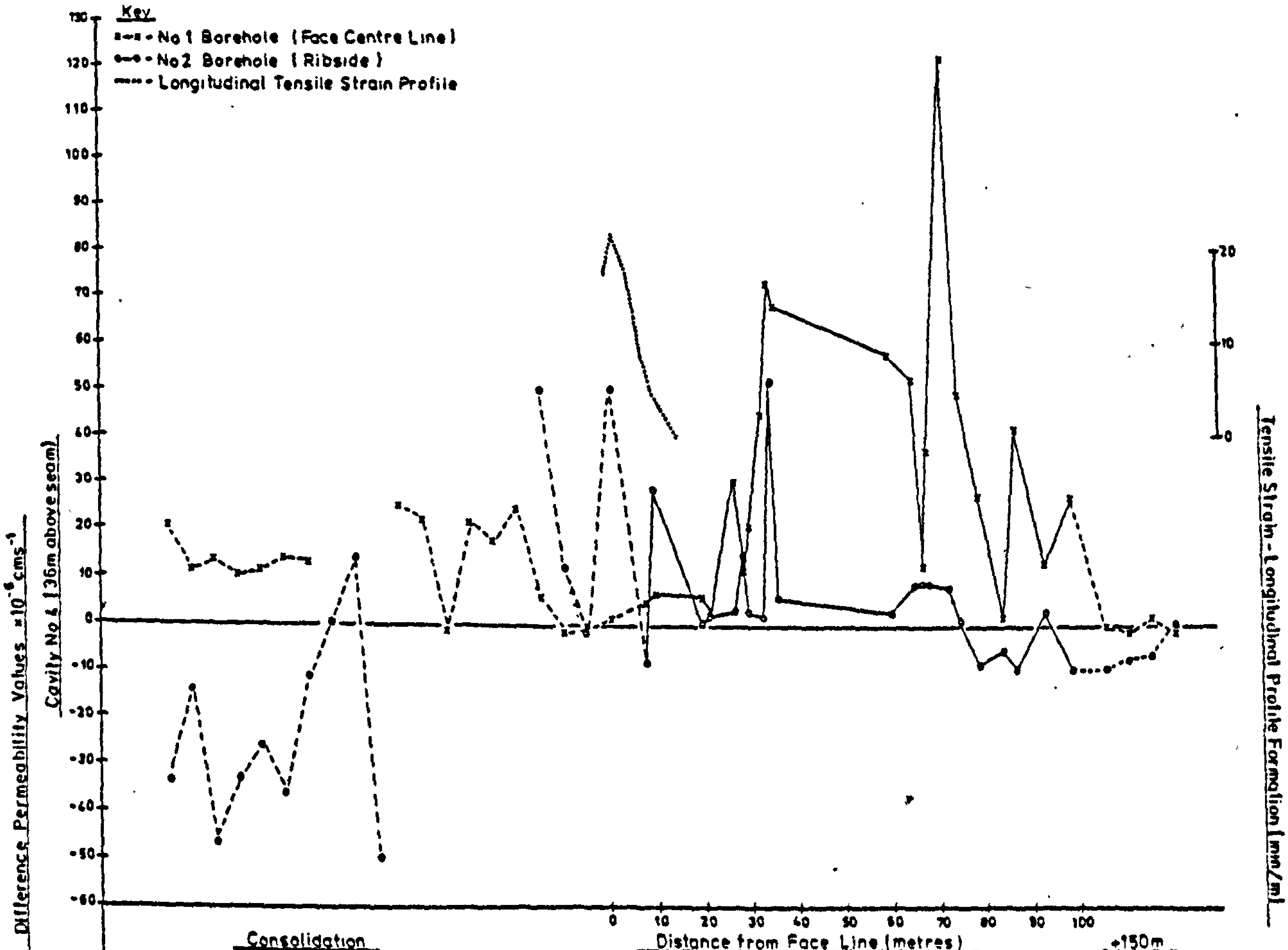


Figure 4.7

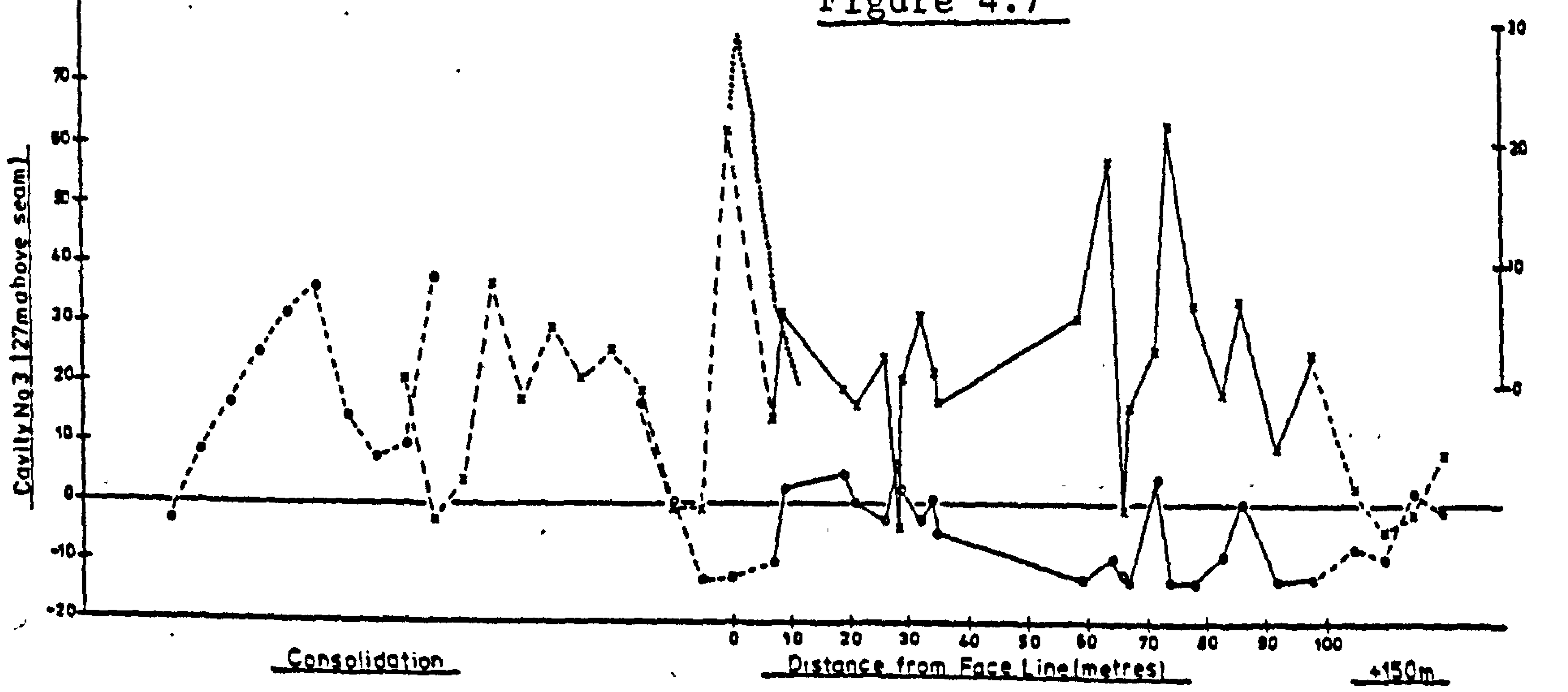


Figure 4.6

Figures 4.6 and 4.7 Cumulative Difference Permeability Values for Lynemouth Nos. 1 and 2 Boreholes, Cavities 3 and 4 against Distance from Face Line

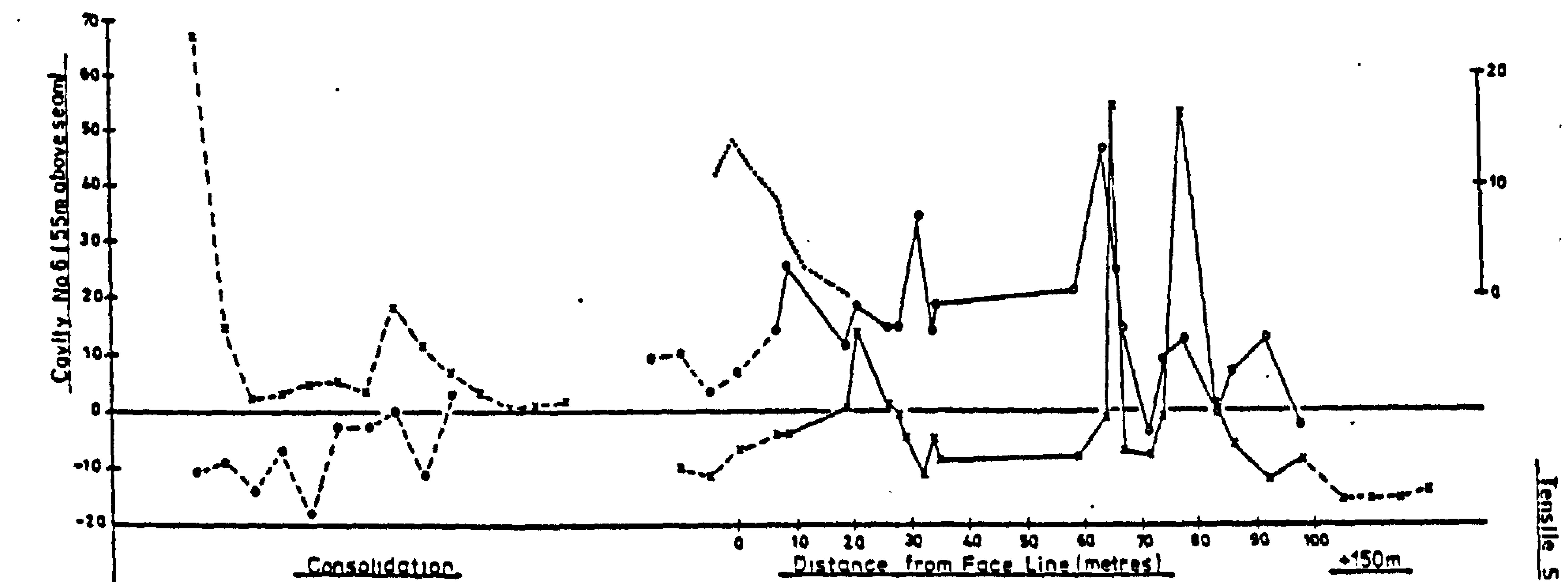


Figure 4.9

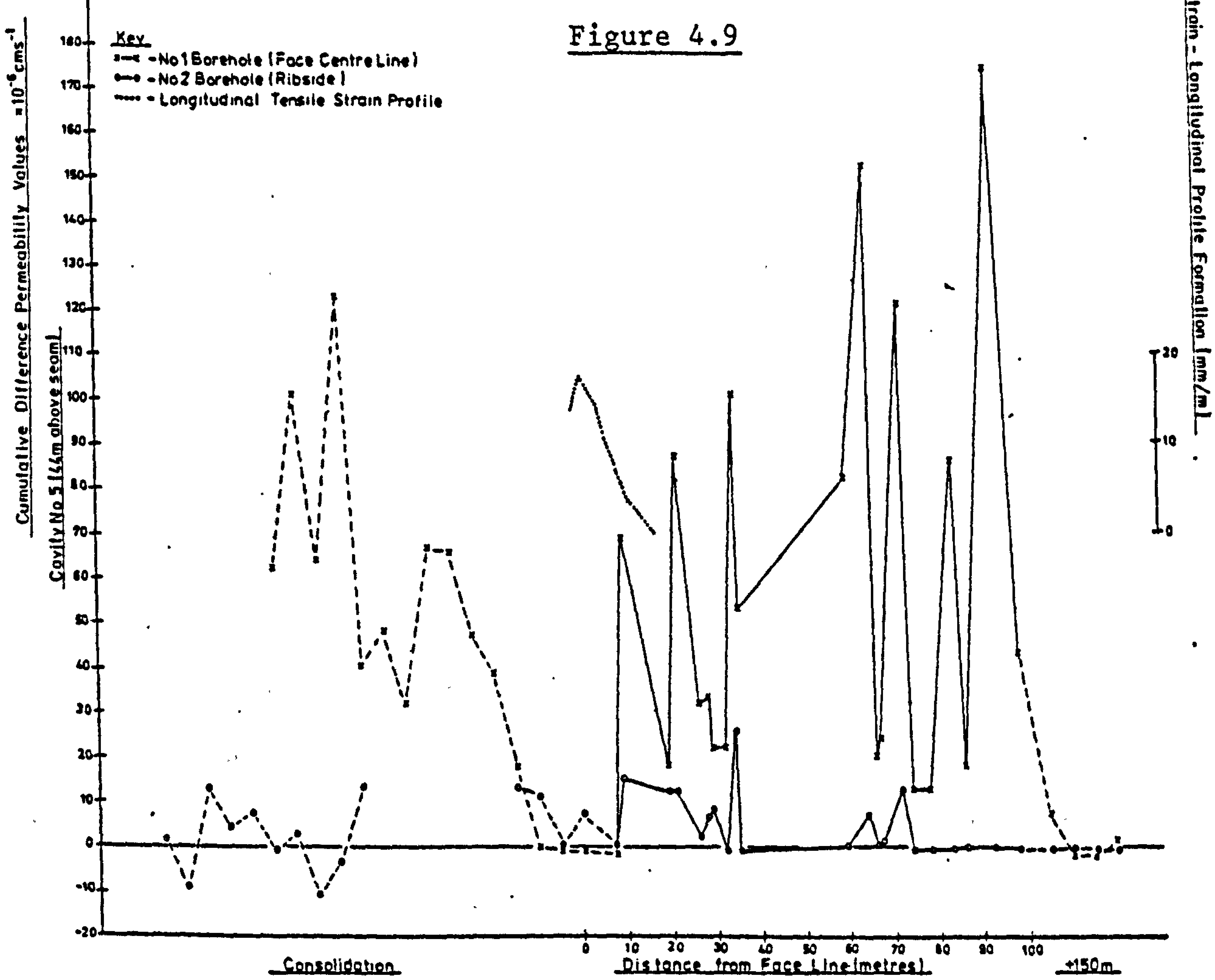


Figure 4.8

Figures 4.8 and 4.9 Cumulative Difference Permeability Values for Lynemouth Nos. 1 and 2 Boreholes, Cavities 5 and 6 against Distance from Face Line

It has also been shown, Neate (41), that permeability does not increase linearly or otherwise with increased height above the seam and that both top and bottom cavities experience changes within the same order of magnitude. Finally, changes which do occur, happen in both holes at broadly the same time, although the centre line hole experienced larger changes more rapidly.

Monitoring of vertical strain at the site was disappointing due to it not being undermined. However, Neate (41) reports an initial drift which demonstrates that permeability fluctuations were occurring without large and obvious changes in ground strain. Conventional subsidence theory predicts that the onset of longitudinal tensile strain in the centre line hole, should occur between 12 - 13 m in front of the face line, Table 4.7. When these values are superimposed onto the cumulative difference graphs, fluctuations and a general increase in permeability are seen to start well in advance of the conventionally predicted values. However, two main points must be considered when examining this evidence:

1) Predicted longitudinal strains in hole 1 will be greater than in hole 2, since hole 1 is over the face centre line. However, transverse compressive strains will also be greater in hole 1 than hole 2, since it is again over the face centre line. A contradiction therefore exists between the longitudinal tensile and transverse compressive strains experienced by each hole.

2) Predicted subsidence profile values have been calculated using methods outlined in Chapter 3 and are therefore subject to the restrictions and limitations described.

Cavity No.	Height above seam (m)								
6	55	+12.7	+5.2	+2.0	+0.8	-3.3	-7.2	-10.0	A
		0.0	2.8	5.6	8.4	11.2	14.0	11.2	B
5	44	+9.4	+3.4	+0.8	-1.4	-3.2	-6.7	-8.9	A
		0.0	3.5	7.0	10.5	14.0	17.5	14.0	B
4	36	+7.0	+2.2	+0.1	-1.7	-3.2	-6.1	-7.8	A
		0.0	4.3	8.6	13.0	17.3	21.6	17.3	B
3	27	+4.7	+1.1	-0.4	-1.9	-2.9	-5.0	-6.4	A
		0.0	5.8	11.6	17.5	23.3	29.1	23.3	B
2	20	+3.1	+0.4	-0.7	-1.9	-2.8	-4.2	-5.2	A
		0.0	7.9	15.7	23.6	31.4	39.3	31.4	B
1	15	+2.1	0.0	-0.8	-1.7	-2.4	-3.4	-4.1	A
		0.0	10.5	21.0	31.4	42.0	52.4	42.0	B

A - Distance from Face Line (metres)   B - Tensile Strain (mm/m)

(+) - in front of Face Line   (-) - behind Face Line

Table 4.7 Longitudinal Tensile Strain Profile (Centre Line), Lynemouth No. 1 Borehole.

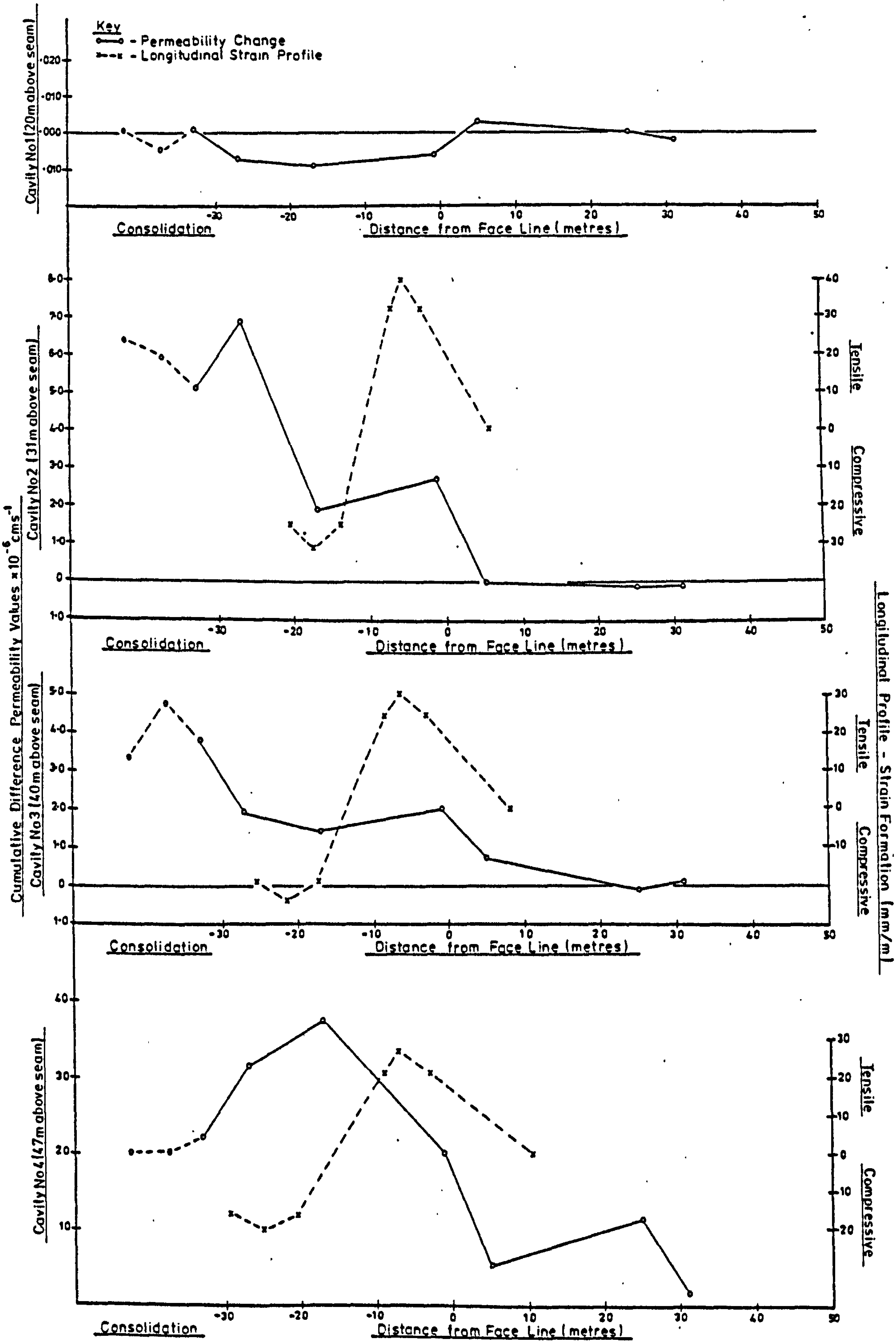


Figure 4.10 Cumulative Difference Permeability Values for the Wentworth Test Cavities against Distance from Face Line

At Wentworth, Figure 4.10, a steady increase in permeability associated with face advance is seen in all test cavities, until 20 m past the site. A general decrease then occurs which is probably due to consolidation effects, even though the face stopped 33 m past the site. Readings were continued after the face stopped in order to monitor consolidation effects.

Conventional subsidence theory predicts that in the top test cavity, No. 4, the onset of change will occur 11 m in front of the face. This figure decreases with increasing depth for the remaining test cavities. Longitudinal strain profiles have been superimposed onto the cumulative difference graphs, Figure 4.10. It can be seen that the onset of permeability change occurs well in advance of the conventionally predicted values. The extremely low values encountered in test cavity No. 1 are thought to be due to a blockage caused during installation.

Neate (41) has indicated that the onset of permeability change occurs well in advance of that observed for the onset of surface subsidence, by a distance of 40 m. However, small dilations seen on the extensometer wires cannot be confidently linked to permeability onset curves, since at this degree of precision the variable tension extensometer is at the limit of its measurable accuracy.

4.7.2 Permeability at the Face End

To date, two sites have monitored permeability changes in the face end region.

At Bagworth, Figure 4.11, consistent base permeability values are seen until the face is 20 m from site, after which a general increase occurs. However, fluctuations are seen prior to 20 m and significant ones are experienced in all three holes at 140, 85 and 50 m from site.

Neate (41) suggests that in holes 1 and 2 an increase in permeability is seen in going from the lowest to the highest test compartments, with increased face proximity. However, hole 3 shows a negative gradient in which the highest compartments are affected first.

Roadway deformation measurements have also been correlated with the onset of permeability in the boreholes. Holes 1 and 2 show no effects of increased permeability with roadway closure, while No. 3 does. To explain this Neate (41) proposes that the roadway should initially be sited in solid ground. Then as the face advances, interaction occurs in the strata surrounding hole 3 which in turn affects the bridging properties of strata overlying the roadway. This results in stress relief, subsequent loading and floor penetration of the arches and a general closure of the roadway. The initial permeability response seen in holes 1 and 2 may be due to yield zone effects in the roadway periphery aggravated by increased face end proximity.

At Annesley, the location and nature of the test site only allowed monitoring to be successfully undertaken once the face had passed. Whittaker et al (57) have shown in Figures 4.12 and 4.13 that a variation in flow exists between test cavities when compared with face position. In Figure 4.12, the test horizons monitor a

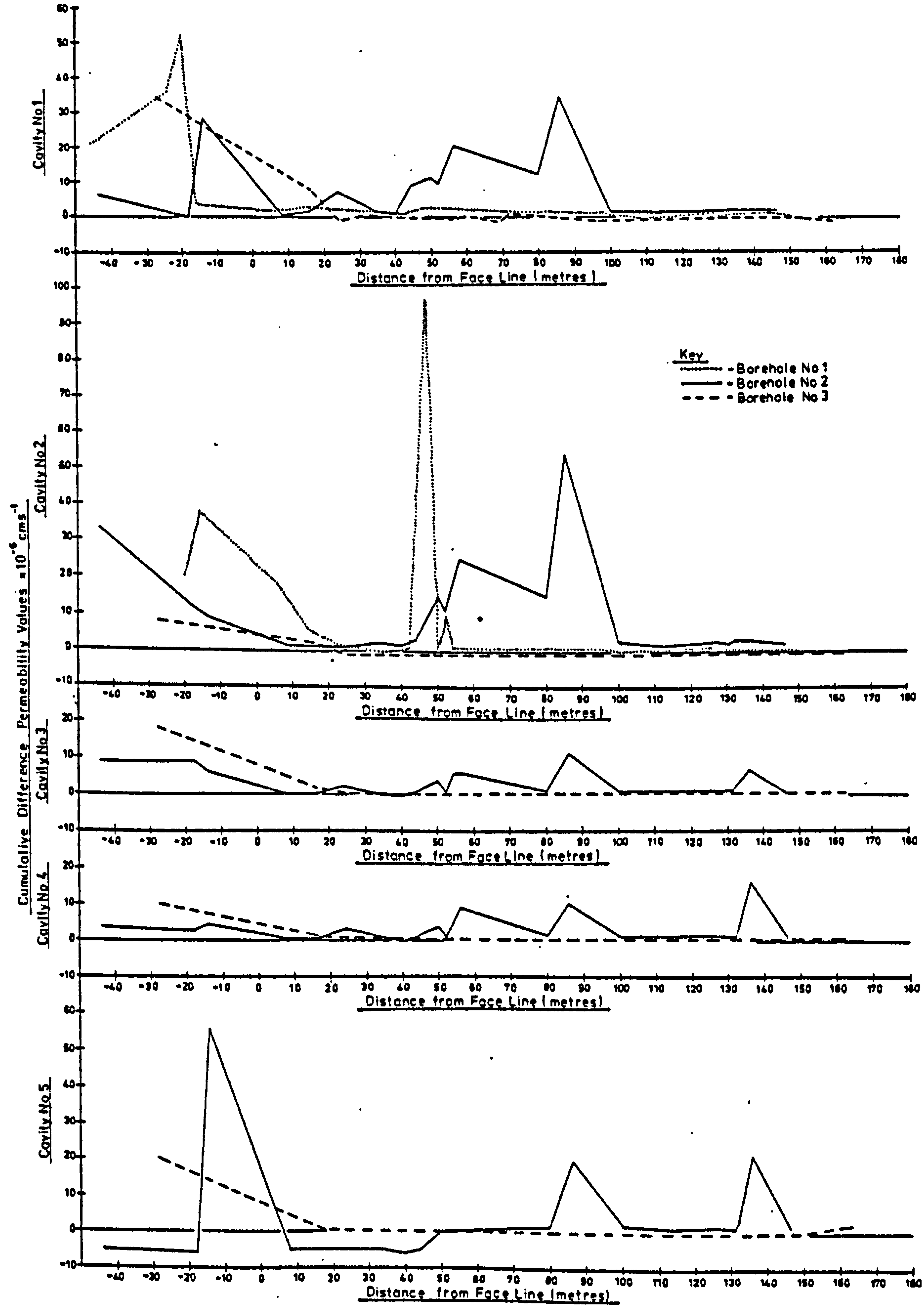


Figure 4.11 Cumulative Difference Permeability Values for the Bagworth Test Cavities against Distance from Face Line

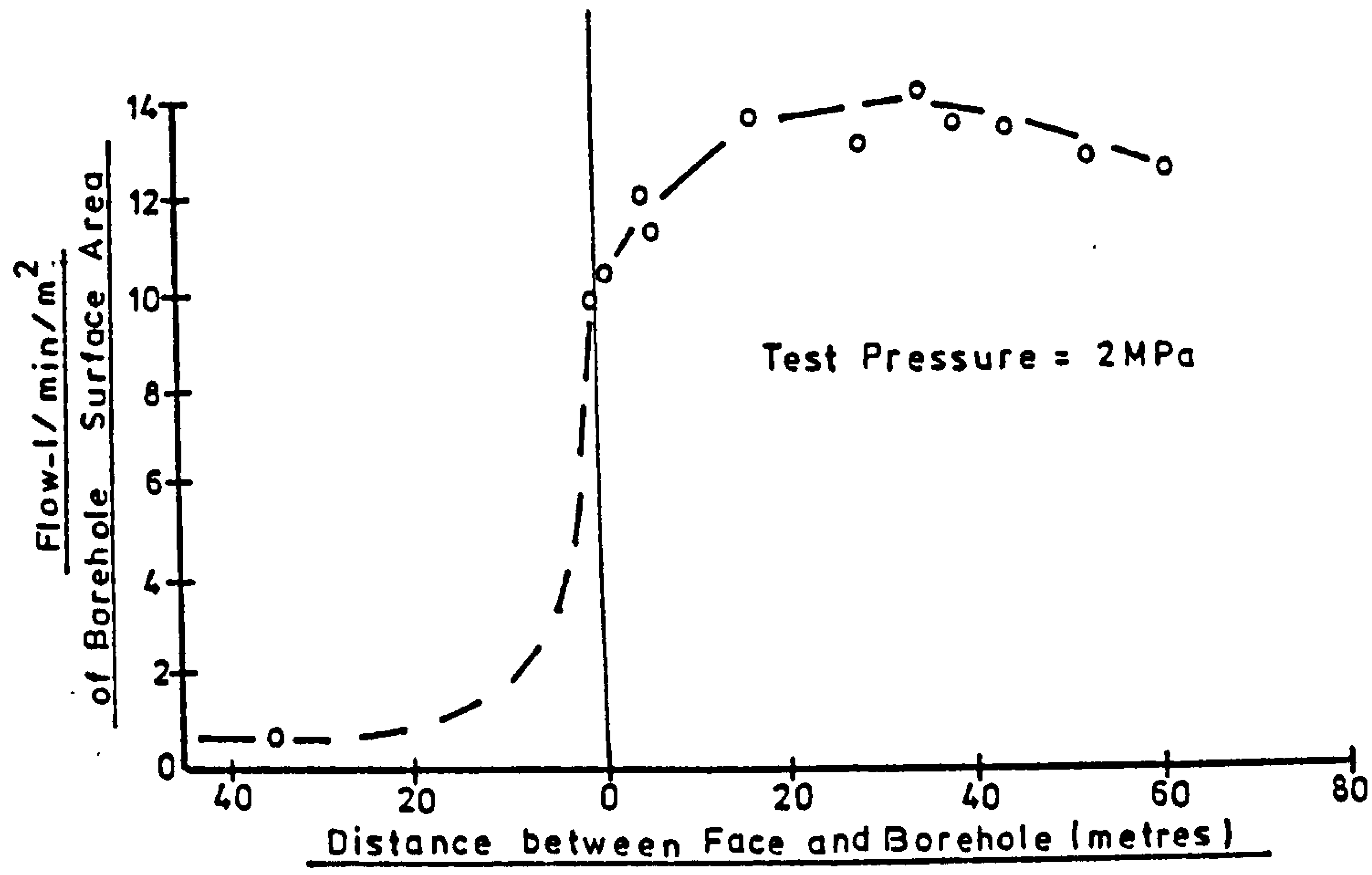


Figure 4.12 Flow Characteristics of No.1 Borehole against Face Position, Annesley
(after Whittaker et al (57))

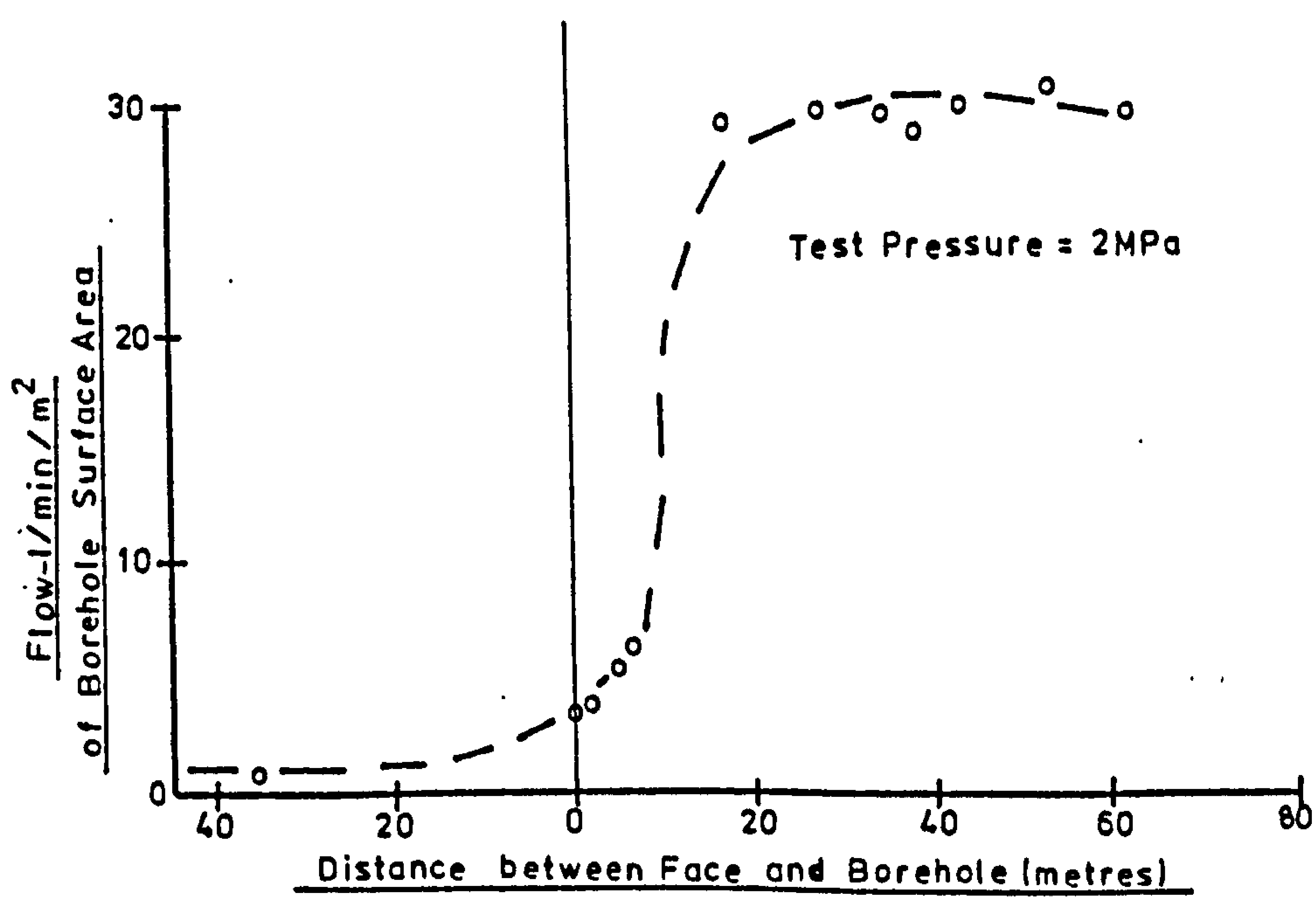


Figure 4.13 Flow Characteristics of No. 4 Borehole against Face Position, Annesley
(after Whittaker et al (57))

band of variable strata between 2 and 8 m above the seam and it is apparent that the strata becomes significantly affected immediately after undermining. Similarly, Figure 4.13 shows a test horizon between 31 and 41 m above the extraction where strata does not become significantly affected until some 15 m past site. Whittaker et al (57) proposed for the Annesley site, that a progressive upward movement in permeability occurs behind the face line which is related to the opening and closure of natural strata discontinuities and bed separation networks during undermining. However, at 40 m behind the face line a point is reached where consolidation effects begin to reduce the overall permeability changes.

Data from Annesley has been used to produce cumulative difference graphs, Figure 4.14, but only readings from two holes are given due to data availability problems and the presence of suspected instrumentation or operational errors. Unlike Bagworth, test cavities were not situated in close proximity to the face end or gate roadway, Figure 4.1. Examination of Figure 4.14 shows a general increase in permeability once the face has passed and this is interpreted as occurring at all levels in a progressively upward manner, irrespective of test cavity geology. Some fluctuations are seen, but it is unfortunate that more readings were not obtained before the face reached the site, so that changes in this region could be monitored and compared with those from Bagworth.

4.7.3 Permeability in the Ribside Pillar

Investigations have been conducted at Whitwick Colliery to investigate permeability changes in barrier and ribside pillars, Whittaker and Singh (40).

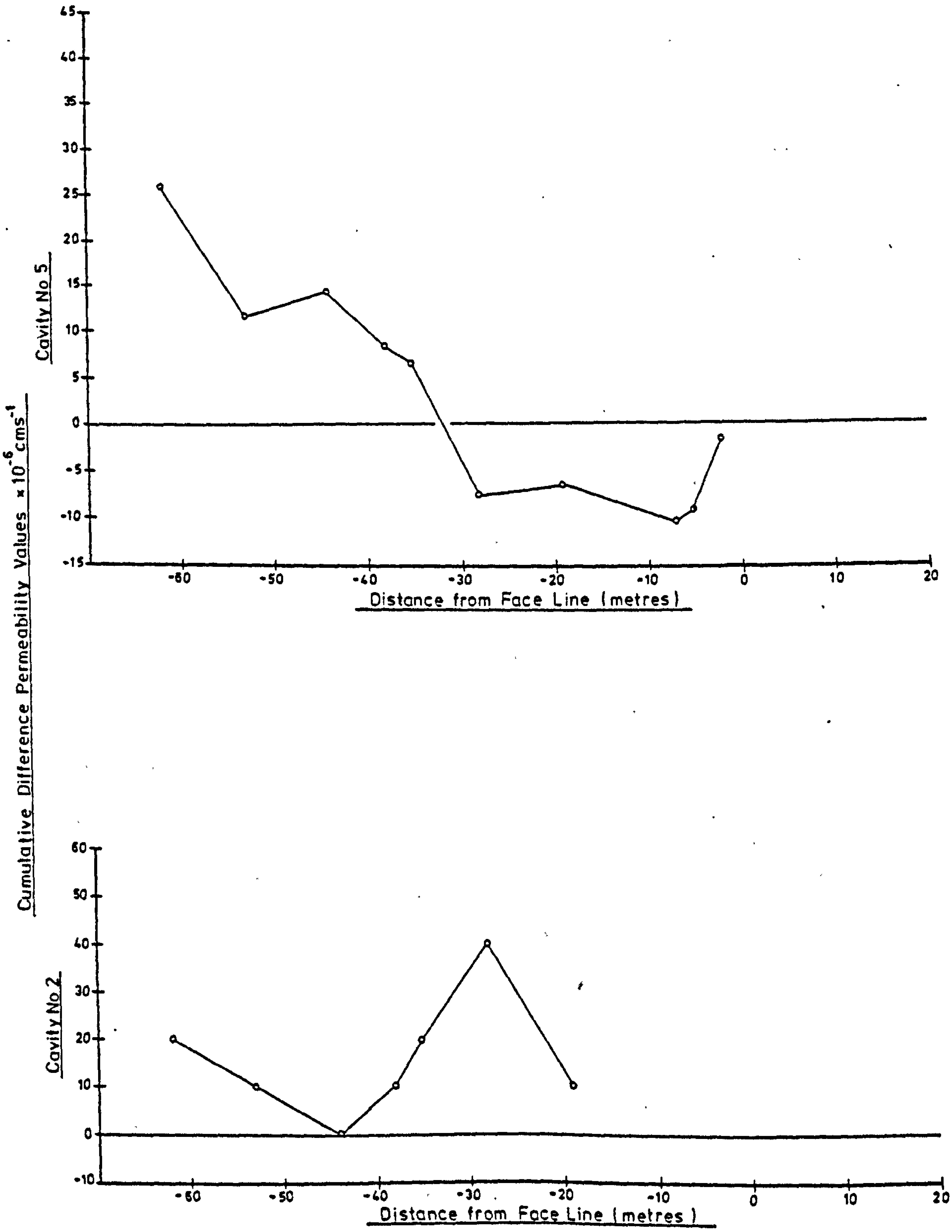


Figure 4.14 Cumulative Difference Permeability Values for the Annesley No. 2 and 5 Test Cavities against Distance from Face Line

If a pillar is constrained both vertically and horizontally along its major axis, the direction of principle stress should lie towards the extracted region and result in the formation of fissure networks running parallel to the pillar edge. Work by Whittaker and Singh (40) has shown, Figure 4.15, that a zone of fracture gradually promotes flow with increasing depth into the pillar, once the face has passed, equilibrium becomes established when the face is 120 m behind the site. Figure 4.15 can also be used to demonstrate the opening and partial reclosure of fissure networks associated with face advance. Figure 4.16 shows results for a deeper test section from which similar conclusions can be drawn. Vertical roadway closure has also been plotted and a tentative correlation appears to exist between roadway closure and the flow characteristics and hence permeability of the pillar. It is proposed, that this might be due to development of a vertical stress abutment zone near the pillar edge which is similar in nature to that experienced during longwall advance, Whittaker and Singh (58).

Figure 4.17 shows flow characteristics within a pillar represented as iso-flow lines. The flow appears to increase appreciably within the first 5 m of pillar after which it steadily decreases and becomes minimal at a distance of greater than 10 m. It is therefore concluded that a zone of fracture up to 10 m wide can occur in the pillars of shallow workings before confining pressures in the core significantly reduce the effects of increased permeability. Similarly, in deeper workings, Whittaker and Singh (40) suggest that more intense fracturing occurs which increases the flow rate and penetration depth into a pillar.

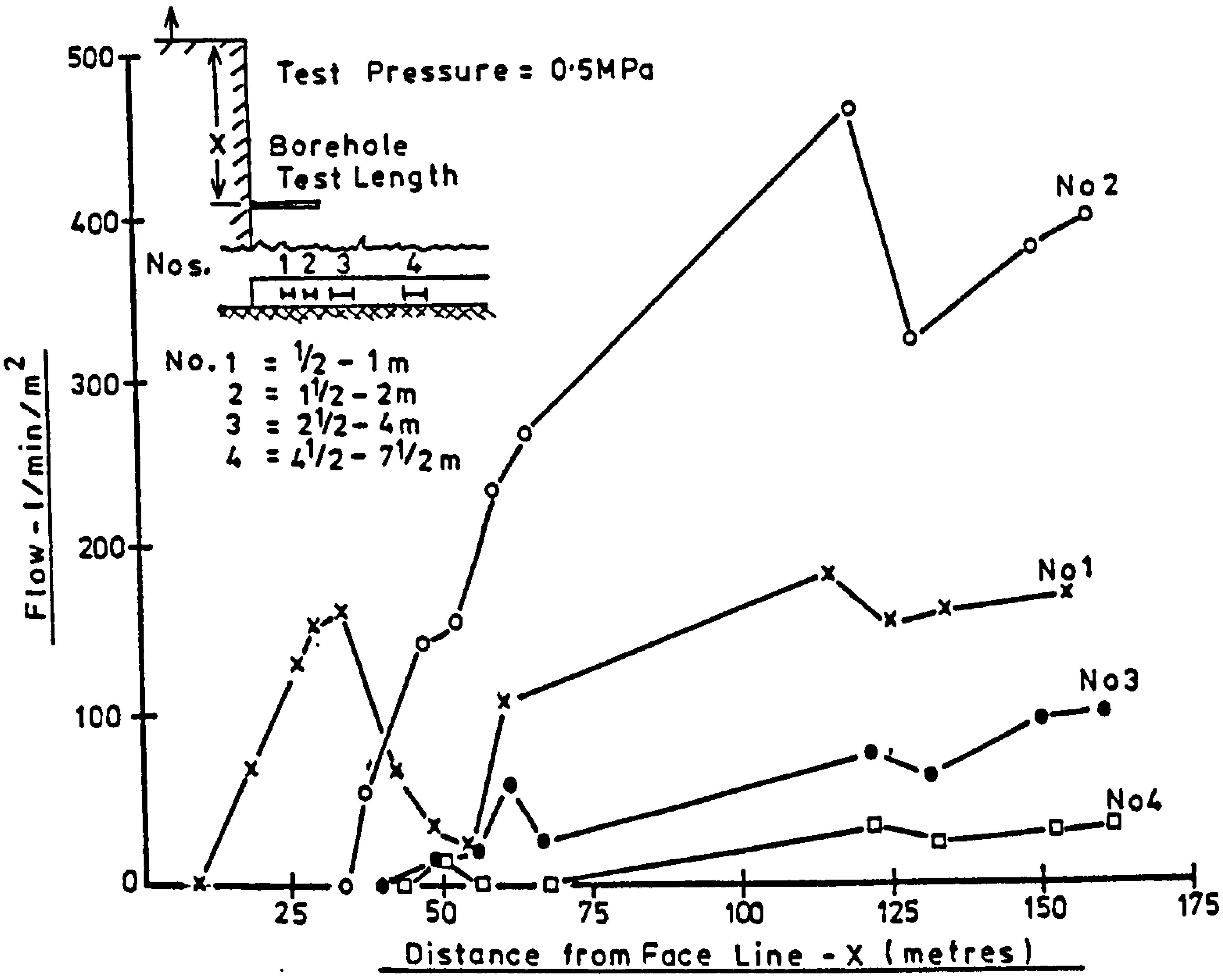


Figure 4.15 Flow Characteristics for Different Test Cavities within the Ribside Coal Pillar, Whitwick Colliery
(after Whittaker and Singh (40))

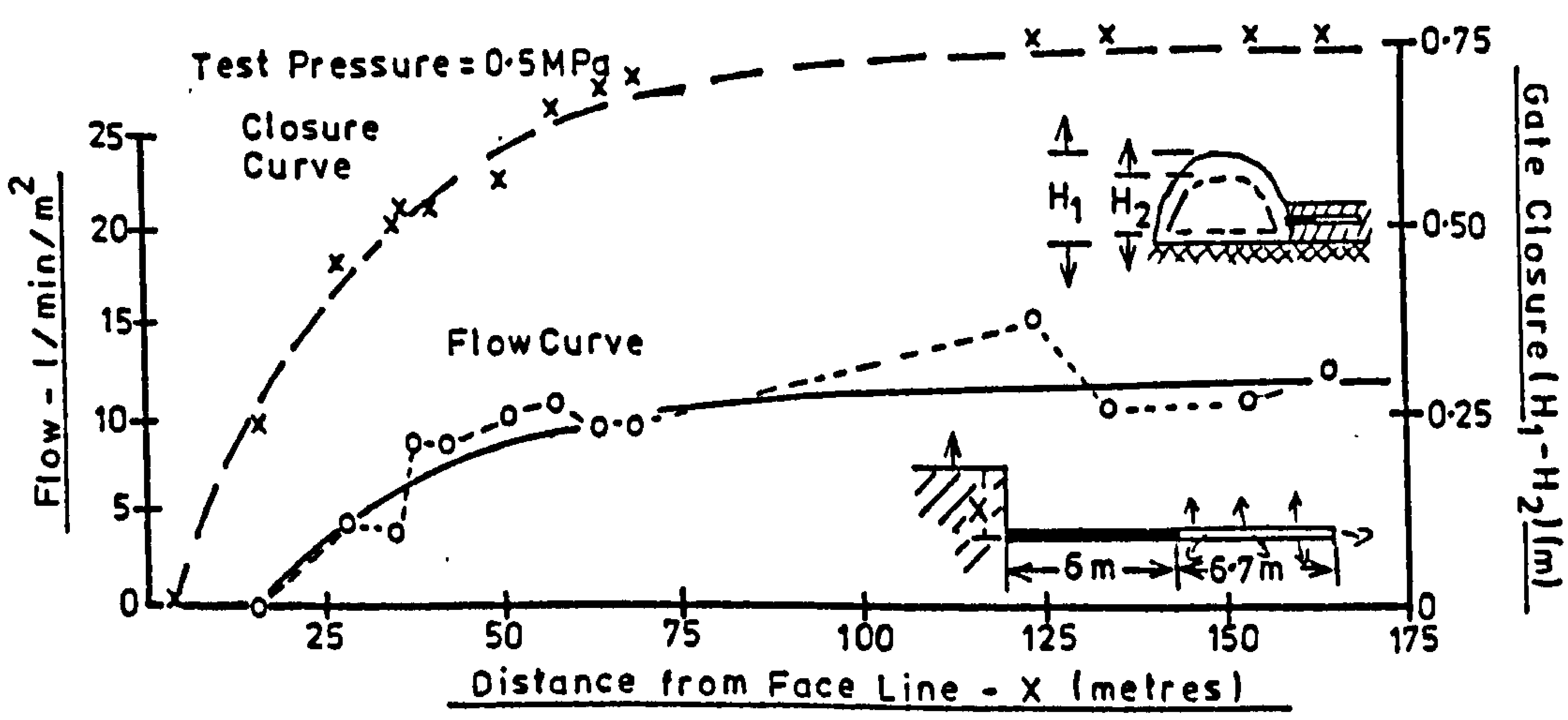
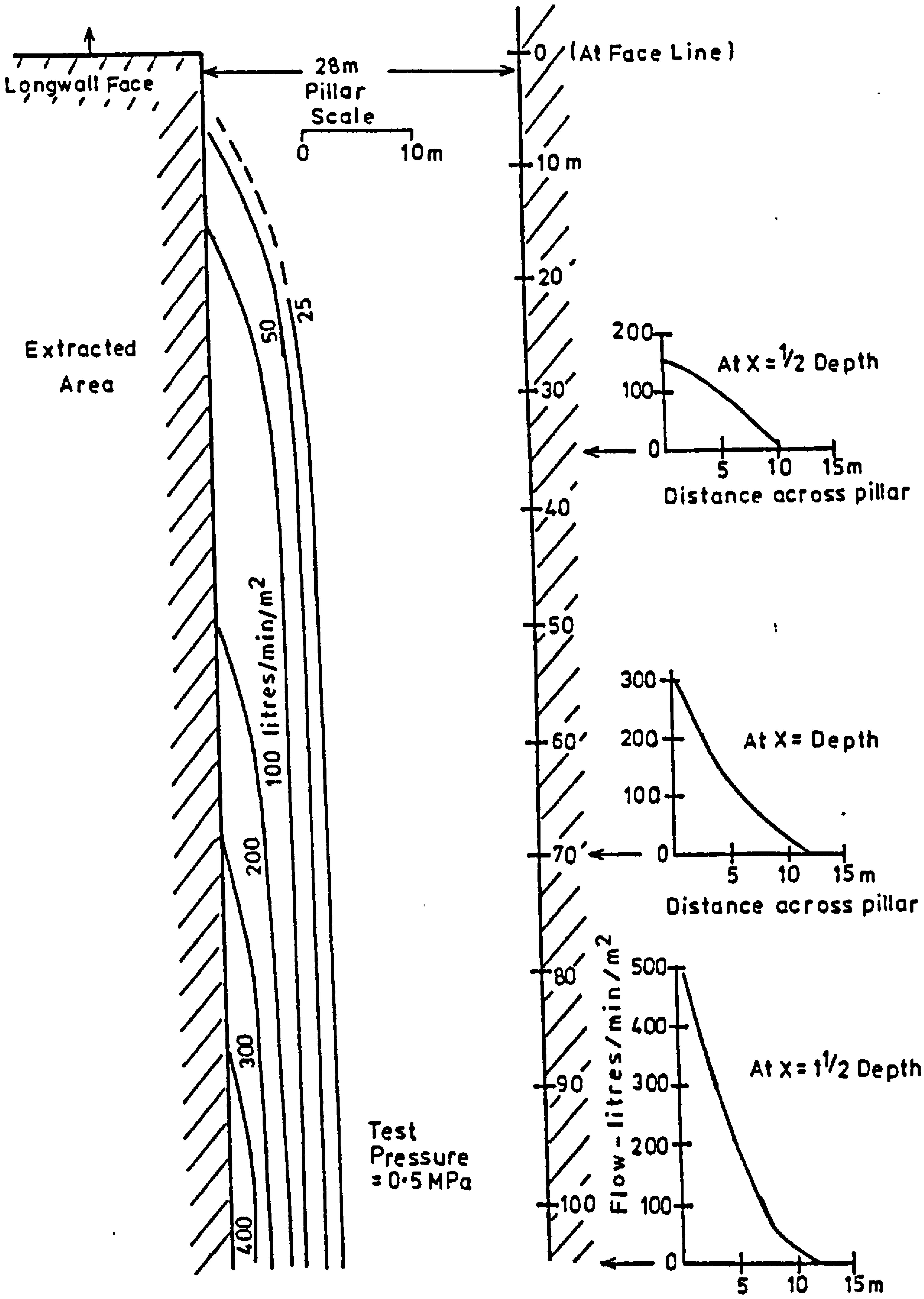


Figure 4.16 A Comparison of Ribside Coal Pillar Flow Characteristics and Roadway Closure at Different Positions behind the Face Line, Whitwick Colliery
(after Whittaker and Singh (40))



<u>Figure 4.17</u> <u>Iso-flow Lines for a Ribside Coal Pillar, Whitwick Colliery</u>

(after Whittaker and Singh (40))

4.8 Discussion of Results

The overall objective of the instrumentation was to monitor insitu permeability changes associated with subsidence profile development around an advancing longwall panel. In essence, this has been achieved although certain limiting criteria and generalised assumptions have been necessary when interpreting the results.

At Bagworth, the instrumentation was installed only on the understanding that the boreholes would not exceed a vertical height of 10 m. The resultant test cavities were therefore sited in close proximity to the roadway and may have been subject to the effects of yield zone formation, Miller (59).

Both Lynemouth and Wentworth reveal that permeability changes cannot be directly linked to conventional subsidence theory. However, permeability does appear to be a sensitive indicator of change in strata behaviour occurring around dynamic longwall extractions, which cannot be monitored by conventional instrumentation.

At Wentworth, the magnitude of permeability changes experienced was considered very small for such a shallow site. In addition, the greatest change was monitored at surface rather than nearer the seam, Neate (41). This was thought to be due to physical properties of the strata accommodating most of the induced strain. Similarly, effects in the top section of strata may have been altered by a change in physical properties due to weathering. Investigations at Whitwick concluded that current design criteria for barrier and ribside pillars based on one tenth depth provided an adequate margin of safety, even though it was not originally derived on a permeability basis.

At each of the sites monitored, potential errors exist which although assumed not to occur, may well exist in practice. Instrumentation errors, such as cement seepage from the sealing plug section into the test cavity, could result in a host of possibilities ranging from partial to complete blockage of a test cavity, as in Wentworth No. 1 cavity. Similarly, a complete or partial cement shell could form around the test cavity wall. Initially, the shell would be impermeable, but it may be sufficiently weak to become broken by the stresses induced by face advance. Cement from a seal section could also penetrate the surrounding strata via natural fissure networks and thereby reduce the inherent strata permeability. High cement viscosity, would in the majority of cases restrict strata penetration to fracture/fissure rather than intergranular mechanisms.

Operator and operational errors may also exist, since the equipment has to be assembled and reconnected for each pressurisation test. Unmeasured and under or over estimated pressure losses can therefore occur in addition to the collection of spurious readings. Careful handling and quality control checks throughout the data handling and analysis stages should also be undertaken to ensure accurate and reliable end results.

Errors can also occur during the investigation due to a lack of geological information. The test cavity strata sequence is often derived from a secondary geological source and insufficient data is usually available to distinguish between fracture and inter-granular components of permeability within the cavity. Observed fluctuations in a series of test readings may be due to errors already mentioned, but could also be associated with the 'silting up' or 'washing out' of both intergranular and fracture permeability components in the surrounding strata.

4.9 Conclusions

Changes in permeability can be linked to induced ground strains formed during subsidence profile development around longwall extractions. However, insufficient data exists at present to quantify these effects and confirm or dispute either of the model concepts discussed in section 4.2. Data from a great many sites is still required before permeability changes occurring around a longwall extraction can be either comprehensively understood or predicted by empirical methods.

The development of a continuous permeability monitoring system would in part overcome the lack of data imposed by a limited number of sites using conventional techniques. However, optimum use would have to be made of existing boreholes in order to maximise monitoring of both transverse and longitudinal profiles. At Lynemouth, in the authors opinion, it might have been more useful to have sited both boreholes over the face centre line rather than one over the centre line and one near the ribside. In this manner, instead of monitoring two sites in a zone of principally transverse compression, a comparison could have been made between the onset times and magnitudes of two sites in a zone of longitudinal tension. Finally, when considering future sites, it is important to determine beforehand the likelihood of complete undermining and optimize location of the instrumentation accordingly.

CHAPTER 5

DESIGN AND INITIATION OF PERMEABILITY INSTRUMENTATION SCHEMES

CHAPTER 5

DESIGN AND INITIATION OF PERMEABILITY INSTRUMENTATION SCHEMES

5.1 Introduction

Over the past 5 years, the Department in conjunction with the National Coal Board has undertaken a series of investigations to monitor changes in insitu strata permeability around longwall extractions. Full details of these sites can be found in Neate (41) and Whittaker and Singh (40). A comprehensive assessment of the sites and results can be found in Chapter 4.

Subsequently, the author has been involved with three more investigation sites at which the objectives were two fold:

1) Extension of existing instrumentation techniques to allow collection and analysis of insitu permeability changes for a variety of conditions. Analysis of sufficient test site data may allow the evolution of empirical predictive methods for determining permeability changes around longwall panels.

2) The application of existing monitoring techniques to a variety of rock mechanics and strata control problems, in order to establish their validity as alternative field instrumentation.

It is proposed to use details of the authors investigations to illustrate three main points, which should always be considered when planning an instrumentation scheme:

1) Choice of Test Site

2) Design of Test Site

3) Installation and Monitoring of Test Site.

It should be noted at this stage that of the three sites used as examples, only two reached the design and consultation stage (Blackhall and Whitwick), while the third Hickleton although initiated, only achieved limited success. In each case, cessation was due to a combination of circumstances beyond the control of both the author and the colliery staff. However, although success was limited, a wealth of practical experience was gained, which it is hoped will prove useful when planning future sites.

5.2 Choice of Test Site

Choice of a test site, particularly when one is lucky enough to have several available, is not necessarily easy. More often than not, the choice of site is severely restricted to that area in which the colliery is experiencing problems. A comprehensive understanding of existing conditions, why the investigation is required and what are the final objectives, is therefore essential. It is intended to use work undertaken at Blackhall Colliery, NCB North East Area, to outline the process by which a site can be chosen.

5.2.1 Blackhall Colliery

Blackhall Colliery was the most southerly of the Durham coastal collieries, but closed in the early part of 1981 due to cumulative water problems in both the workings and on production faces. Figures

5.1 and 5.2 show the workings under consideration, which are restricted to the Low Main (J) seam. A detailed description of the Blackhall workings, geology and water occurrences is given in Chapter 8.

In October 1979, the Department was asked by the NCB North East area to investigate historical and current water problems being experienced at the Colliery. The object of the investigation was to determine an origin of the water and its relationship to the extraction method. After initial discussions it was proposed that a scheme of instrumentation should be installed to monitor insitu permeability at a variety of potential aquifer horizons above a face(s) and relate the results to the water occurrences experienced.

5.2.2 J 54 Water Occurrence - November 1979

Before the full nature of the problem had been comprehended and details of the instrumentation finalised, a water occurrence occurred on the J 54 face (Figure 5.2) between November 17-18th. It reached a maximum yield of 1.4 m^3/min and resulted in abandonment of the face. Production was immediately transferred to the J 55 panel, which only required a further amount of limited development before equipping. The yield on J 54 had decreased from 1.4 m^3/min to 0.4 m^3/min by the 20th November, when the author visited the face. In addition, a meeting was arranged for the 21st to discuss with members of the colliery staff the availability of further data and assess potential instrumentation sites.

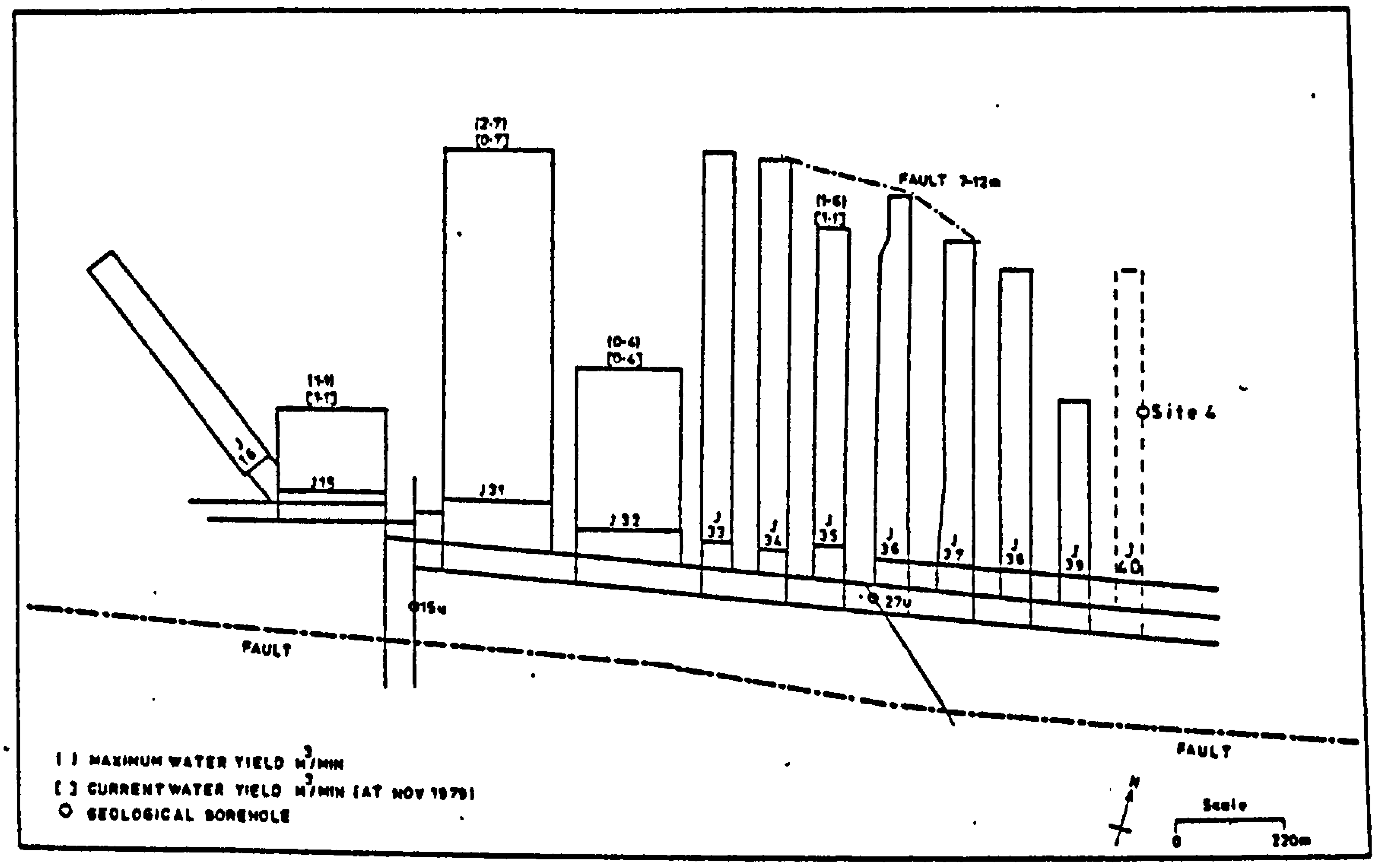


Figure 5.1 Location of workings, north of main E-W Fault, Blackhall Colliery

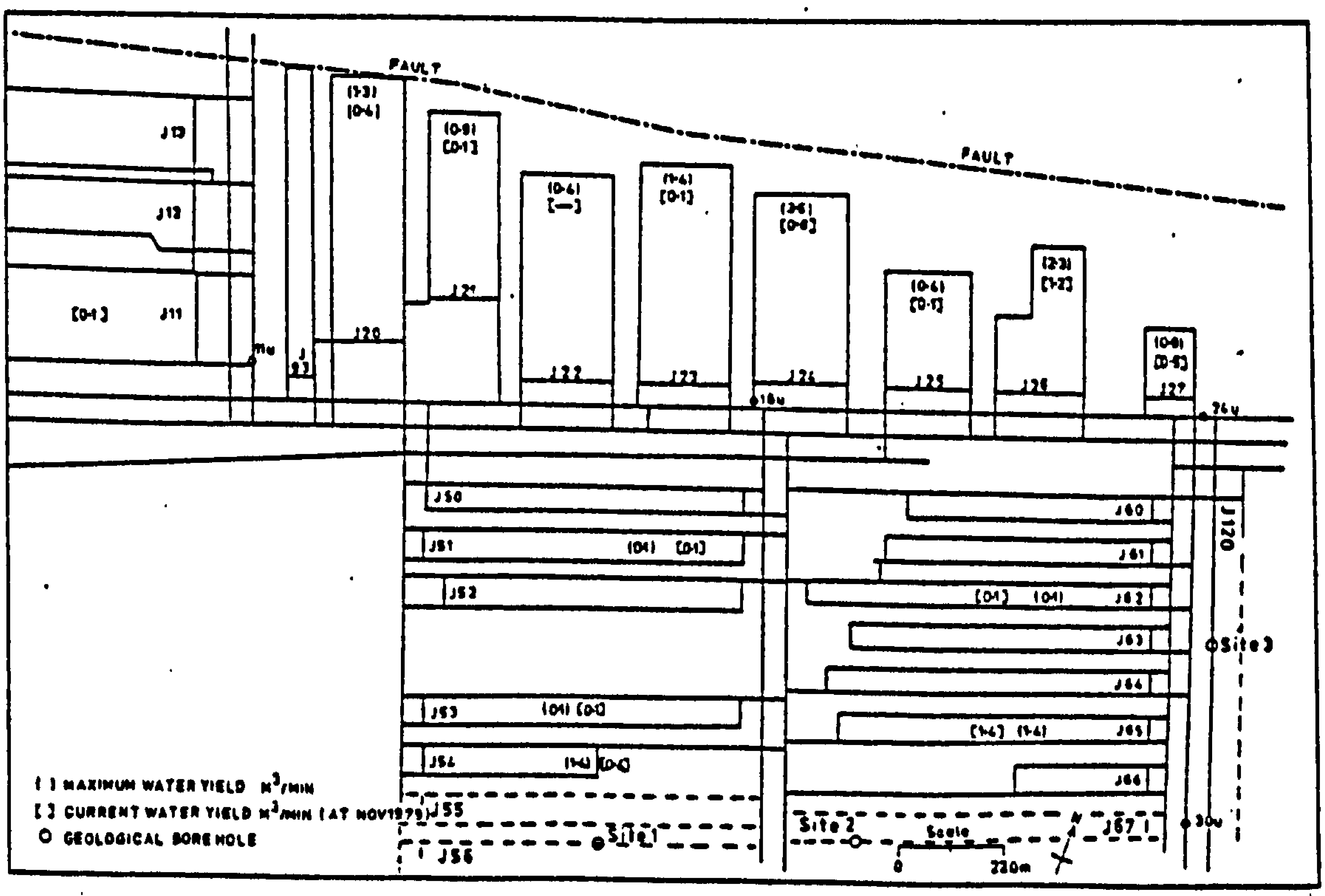


Figure 5.2 Location of workings, south of main E-W Fault, Blackhall Colliery

5.2.2.1 J 54 Water Occurrences - Face Conditions

Dimensions of the J 54 face were 54 m width, 1.47 m extracted seam height and 360 m advance before cessation due to water. It was worked by the Z-retreat method with a pre-driven tailgate and inline maingate.

Yield at the time of visit was 0.4 m^3/min, estimated from maingate pumping records. Water was seen issuing from roof cracks as heavy droppers and light feeders along the face line. In some places, water was seen flowing from the top portion of the coal face, while in others it appeared 'as if the roof was sweating'. In section, droppers could be seen emanating from the face line, over the chock canopies and back into the goaf area, although the heaviest concentration was along the face line. Spatial concentration of the roof droppers/feeders varied along the face as did the quantity yielded from each. Water was seen collecting in isolated and interconnected floor depressions forming pools upto 10 cms deep, which made working conditions very unpleasant. At the face end, some light roof droppers were seen extending upto 1 metre into the gate-road. Drainage was provided by portable pumps feeding a collector pump from which is was subsequently fed via pipe ranges to a district reservoir and surface.

5.2.3 Potential Instrumentation Sites

At a meeting on November 21st between the author and colliery staff, four potential sites were found which would be suitable for instrumentation and subsequent monitoring of insitu permeability changes around an advancing longwall face. The four sites, shown in Figures 5.1 and 5.2 are:

1) The J 55 Panel (Site 1)
2) The J 66 or J 67 Panel (Site 2)
3) The J 120 or J 121 Panel (Site 3)
4) The J 40 Panel (Site 4)

Advantages and disadvantages of each site are discussed separately.

5.2.3.1 The J 55 Panel

This was considered an ideal location; since instrumentation could be sited 360 m from the face start line, so as to correspond with the position at which the J 54 panel was abandoned. The J 56 maingate had already been driven, so drilling could commence immediately without affecting work in the roadway. Boreholes would be in the region of 100 m long, which according to the Area Driller was a reasonable length over which to expect accurate interception of a required horizon.

The main disadvantage was that the J 55 panel was expected to start production in January 1980 and in the time available, only 3 out of 6 holes could be instrumented in order to give a full range of permeability changes associated with face advance. Previous work by Neate (41), suggests that changes in strata permeability can start to occur upto 100 m in advance of the face. An alternative therefore would have been to place the site 500 m from start line, which meant that if it had to be abandoned near the same point as J 54, the site would be lost and only virgin strata permeability values obtained.

5.2.3.2 The J 66 or J 67 Panel

Instrumentation from the J 67 maingate across the J 66 panel while ideal, would not be feasible because of the lack of time available before face completion.

A scheme from the J 68 maingate across the J 67 panel while feasible in the long term, would be subject to delay because the maingate had not yet been driven. Boreholes from the J 68 maingate would be very long, 200-300 m and would cause drilling problems especially when trying to accurately intercept a predetermined strata horizon.

The main disadvantages of this site are therefore:

1) Time required for development of the J 67 panel and J 68 maingate, estimated at between 1-2 years.

2) Few water occurrences have been experienced in this area of the workings.

3) Technical difficulties in drilling inclined holes 200-300 m in length.

4) Cost of drilling in excess of 1400 m of 75 mm diameter borehole.

5.2.3.3 The J 120 or J 121 Panel

The J 120 panel was thought to represent an excellent short term site, while the J 121 would be ideal for subsequent long term investigations based on initial results. A working life of between 6 and 8 months was expected for J 120, which ought to allow adequate time for installation of an instrumentation scheme.

Disadvantages associated with these sites are:

1) No water occurrences have yet been encountered in this area.

2) The panels have been reorientated to work approximately N-S rather than E-W (compared with the J 50 and J 60 series faces).

3) Since the J 120 face is already in production, drilling difficulties might be experienced due to the conveyor position. However, on J 121 drilling and instrumentation could be done prior to conveyor installation.

It was also suggested, that since time was limited a duel scheme could be implemented. Firstly, a short term scheme, involving only 2 or 3 holes could be put over J 55. A second, longer term scheme of 5 or more holes could then be installed over J 121 at a more leisurely pace.

5.2.3.4 The J 40 Panel

The final site considered was north of the main E-W fault and would be located on the J 40 panel. Instrumentation could be installed in either the main or tailgate, but whichever gate was chosen it would have to be specially driven, since advance working methods were normally employed on these panels.

Disadvantages at this site would be:

1) The time factor in developing the panel.

2) Cost of developing the panel.

3) Unrepresentative nature of the water occurrences in this area.

4) Unrepresentative nature of the geology in this area. In particular, the amount of Coal Measures cover between the seam and

base of Permian decreases when going eastwards in the southern area. While in the north, the amount of cover increases when going eastwards.

5.2.3.5 Final Selection of Site

During final assessment of the potential sites, it was decided to abandon completely the J 66/J 67 and J 40 panels as impractical and unrealistic. Similarly, while the J 120 and J 121 sites had many advantages, no water occurrences had yet been experienced in this area. It was therefore decided to concentrate on installing instrumentation over J 55 with the possibility of a longer term site over J 121.

As a postscript, at a meeting in December it was decided that since the commissioning of J 55 was delayed, a full scheme of six holes could be completed and all ideas for J 121 were abandoned. However, in early January a change in Area policy led to cancellation of the proposed field tests and investigations were reorientated along more theoretical lines.

5.3 Test Site Instrumentation Schemes

Once a suitable site has been located, it is necessary to consider both geological and local factors in the final scheme design. Maximum use of the often limited resources is essential, particularly when considering the wider implications of a proposed investigation. Technical considerations of test site installation and instrumentation are discussed in Section 5.4. However, it is intended to use this section to illustrate variations between proposed schemes, by reference to three sites at Blackhall, Whitwick and Hickleton.

5.3.1 Blackhall Colliery Test Scheme

The object of the Blackhall scheme was to monitor insitu changes in strata permeability at potential water bearing horizons, with reference to face advance, Section 5.2 and Chapter 8. Initially, it was proposed that a fan of inclined boreholes sited in a gate-road should be used, which intersected predetermined strata horizons over the extraction panel. However, after consultation with colliery staff, a maximum vertical height of 55 metres was imposed, in order to prevent accidental formation of a hydraulic connection between a test cavity and the base of Permian.

Figure 5.3 shows the final scheme proposedfor the J 55 panel. It consists of 6 boreholes, each 75 mm in diameter, inclined over the panel at predetermined angles and staggered in the horizontal plane by 1 m separations. Data concerning these holes is summarised in Table 5.1. A 30 m test cavity would be formed at the end of each hole, which could be pressurised and allow determination of insitu strata permeabilities under dynamic conditions, Whittaker et al (57). Roadway deformation would be monitored on a regular basis, in order to determine whether a correlation exists with permeability change. Although installation of extensometers was not initially envisaged, they may have been considered at a later stage.

Actual design, construction and installation criteria for the pressurisation equipment would have been very similar to that used at the Hickleton site, which is described in Section 5.4.

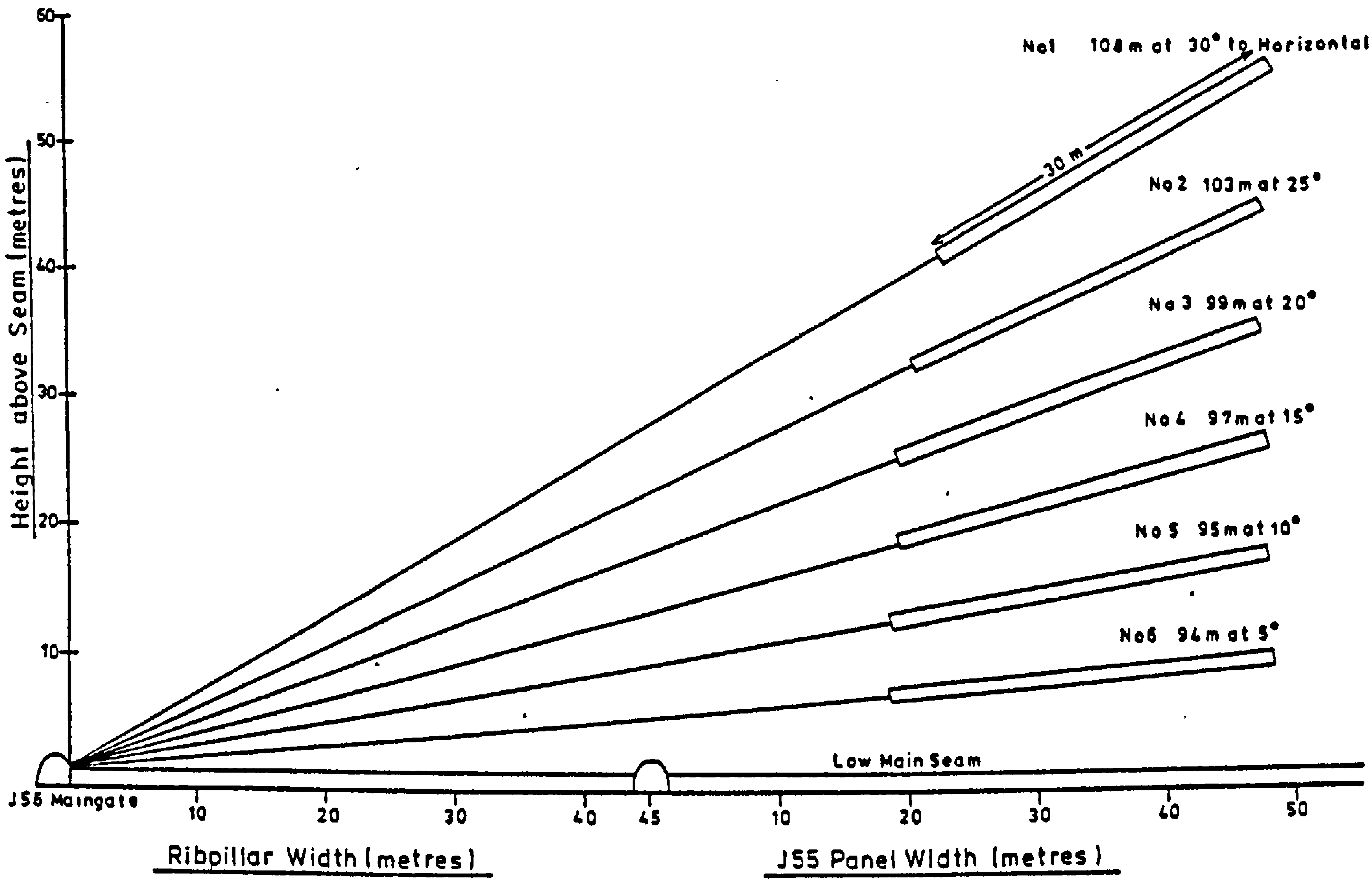


Figure 5.3 Proposed Instrumentation Scheme, Blackhall Colliery

Borehole Number	Inclination	Total Borehole Length (m)	Test Cavity Length (m)	Vertical Height above Seam of Test Cavity Mid-point (m)
1	30°	108	30	46
2	25°	103	30	37
3	20°	99	30	29
4	15°	97	30	21
5	10°	95	30	14
6	5°	94	30	9

Table 5.1 Proposed Instrumentation Parameters, Blackhall Colliery Scheme

5.3.2 Whitwick Colliery Test Scheme

A scheme of instrumentation was proposed for a site in the Nether Lount seam at Whitwick Colliery, NCB South Midlands Area. Mining details of the proposed site are:

Panel Width:	-	55 m
Panel Depth:	-	266 m
Expected Panel Length:	-	350 m
Extracted Seam Height:	-	2.0 m

Objectives of the site were to monitor by means of insitu permeability testing, the formation of fracture/joint characteristics in the overlying strata and relate this to subsidence profile formation and development of the forward stress abutment zone.

The instrumentation scheme is shown in Figure 5.4 and consists of three 35 m boreholes inclined over the panel at 20°, 45° and 70° to the horizontal. A 10 m test cavity is formed at the top of each borehole, which can be pressurised with water, while the remaining 25 m is sealed with either cement or grout/resin. Two further boreholes, one 30 m horizontally in seam, extending into the rib pillar and the other 35 m extending at 40° below the seam are instrumented in a similar manner, with each having three 5 m test compartments. In addition, two extensometers would also be installed, one 20 m long above and one 10 m long below the seam. Each contains several anchor points at pre-determined horizons. Roadway deformation measurements would also have been conducted on a regular basis.

Actual design, construction and installation of the test equipment, had the proposals got further than the initial discussion

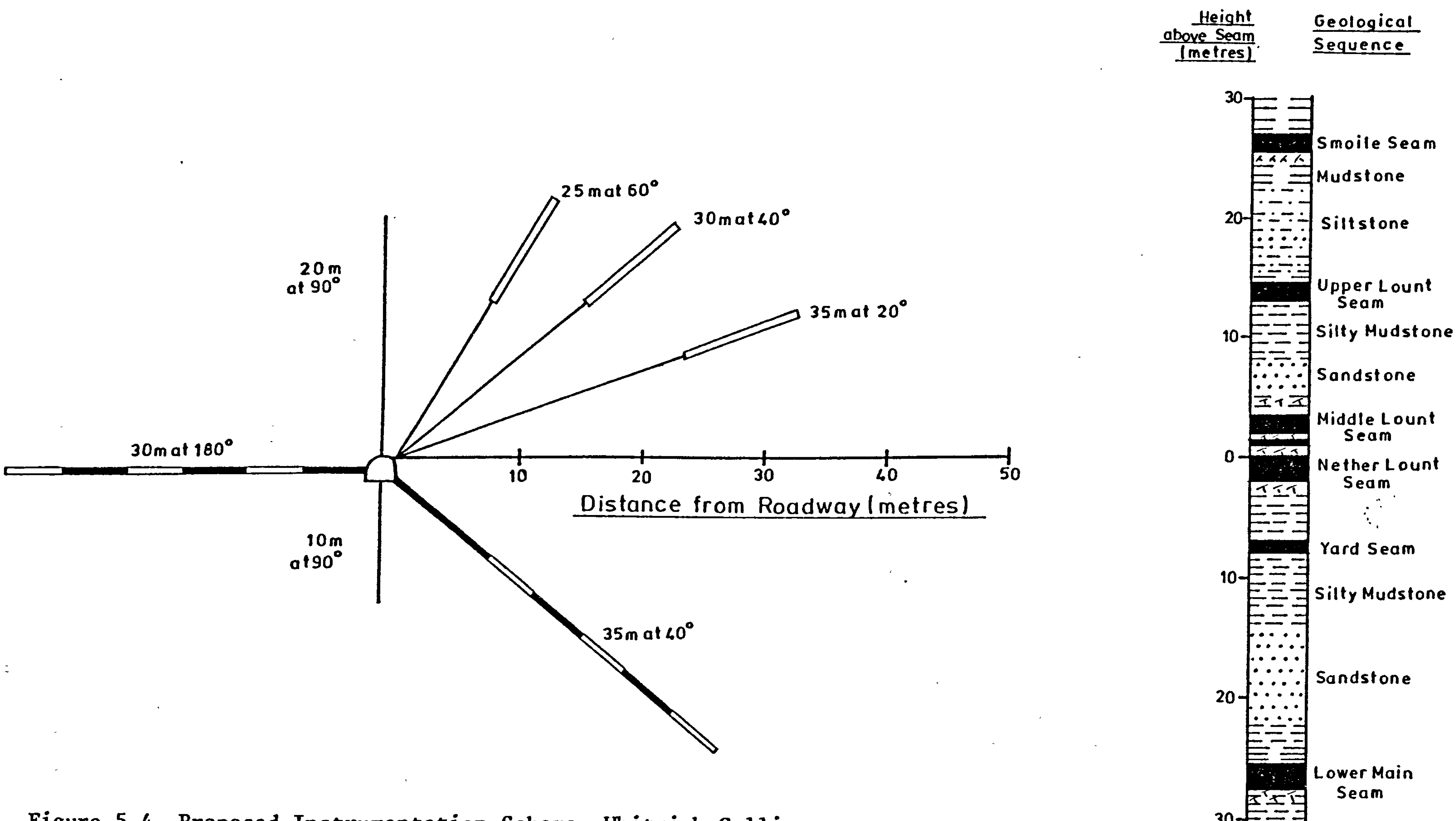


Figure 5.4 Proposed Instrumentation Scheme, Whitwick Colliery

stage, would have been similar to those used either at Hickleton, Section 5.4 or by Neate (41).

5.3.3 Hickleton Colliery Test Scheme

Permeability monitoring techniques were proposed and partially implemented in the Parkgate seam at Hickleton Colliery, NCB Doncaster Area, Section 5.4. These formed part of a comprehensive strata control investigation into the caving characteristics of the Parkgate Rock, Chapters 6 and 7. The technique was used, not primarily to determine permeability, but to use permeability changes as an indicator of caving characteristics associated with subsidence profile formation. The techniques have therefore been used as an alternative form of sensitive instrumentation in the field of strata control and rock mechanics.

The instrumentation scheme is shown in Figure 5.5 and consists of 6 boreholes, 72 mm in diameter and of predetermined length, inclined over the panel. Data concerning the boreholes is given in Table 5.2. A 10 m test cavity was to be formed at the top of each borehole for pressurisation and the remaining length sealed with cement-resin/grout. The panel was worked using the Z-retreat method and the boreholes were sited in the tailgate.

5.4 Hickleton Colliery Permeability Investigations

At Hickleton Colliery, NCB Doncaster Area, permeability determination techniques have been used as an alternative method of field instrumentation to monitor caving characteristics of the Parkgate Rock. Its application was therefore of an applied nature, rather than

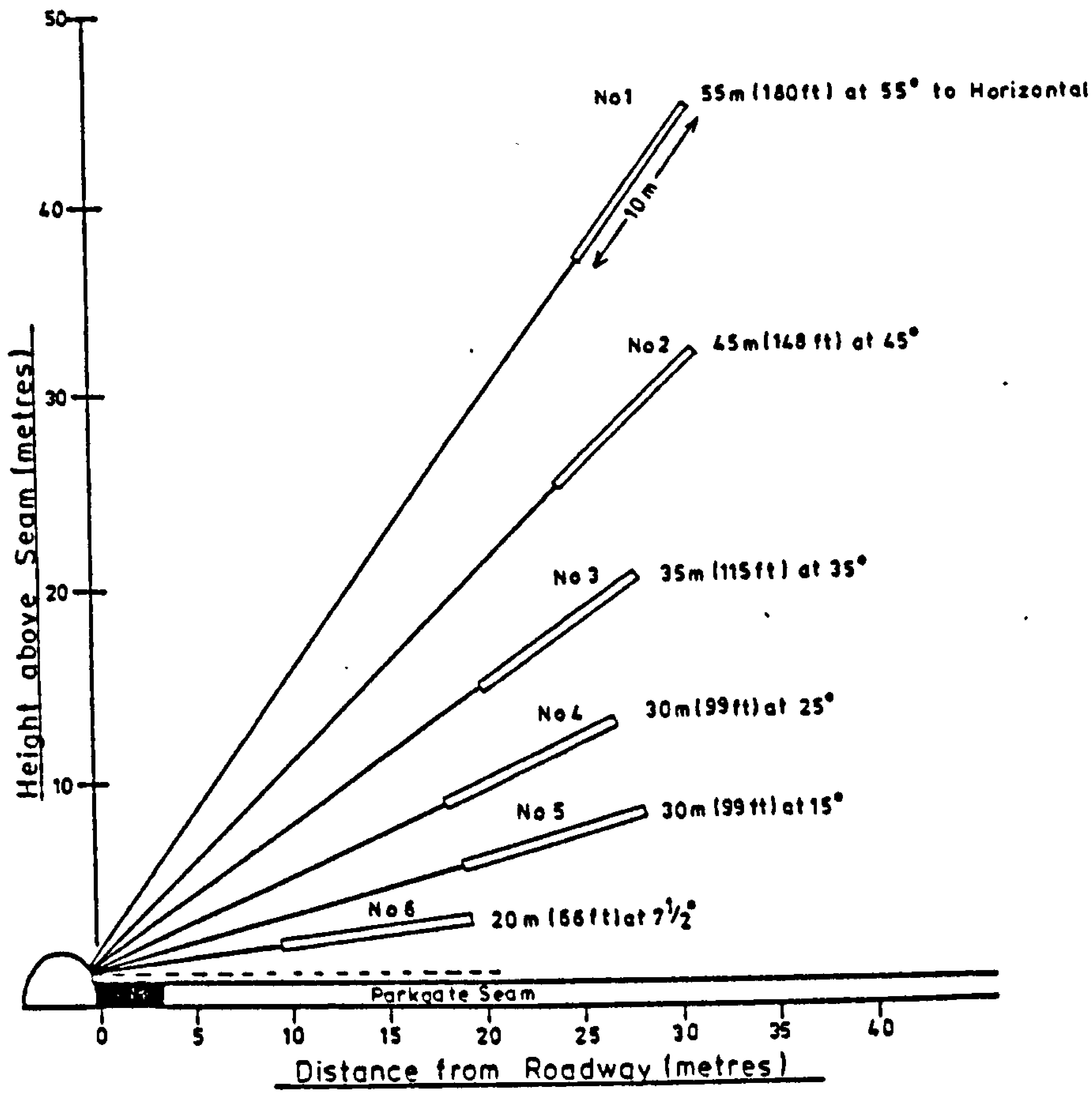


Figure 5.5 Proposed Instrumentation Scheme, Hickleton Colliery

Borehole Number	Inclination	Total Borehole Length (m)	Test Cavity Length (m)	Vertical Height above Seam of Test Cavity mid-point (m)
1	55°	55	10	41
2	45°	45	10	29
3	35°	35	10	17
4	25°	30	10	11
5	15°	30	10	7
6	8°	20	10	3

Table 5.2 Proposed Instrumentation Parameters, Hickleton Colliery Scheme

to principally determine permeability changes associated with longwall mining. Although operational difficulties resulted in only limited success, a wealth of experience has been gained which it is hoped will help and encourage future use of these techniques. This section, therefore deals with the instrumentation and installation techniques used, as well as the experience and practical difficulties encountered. Full details regarding exact site location, geology and working history can be found in Chapter 6, along with full details and analysis of the extensometer schemes. Brief details of the proposed test scheme have already been outlined in Section 5.3.3.

5.4.1 Research Ojectives

Principal objectives of the investigation were to determine caving characteristics of the Parkgate Rock, a 30 m thick sandstone sequence above the P30 panel in the Parkgate seam at Hickleton Colliery. In addition to installing extensometers in the main and tailgates, monitoring strata conditions by means of permeability changes could also determine the rate and extent of subsidence into the goaf area. The degree of pre-face breakage, associated with the forward abutment stress zone, could also be assessed and compared with the extensometer results.

Theoretically, sandstone sequences in the Parkgate Rock should in their intact and undisturbed state yield relatively low permeability values which are associated with intergranular mechanisms, Figure 5.6.1(a). However, if disruption has already occurred due to prior extraction of the P29 panel, Figure 5.7, fracture/fissure mechanisms should be more prominent and consequently make permeability values

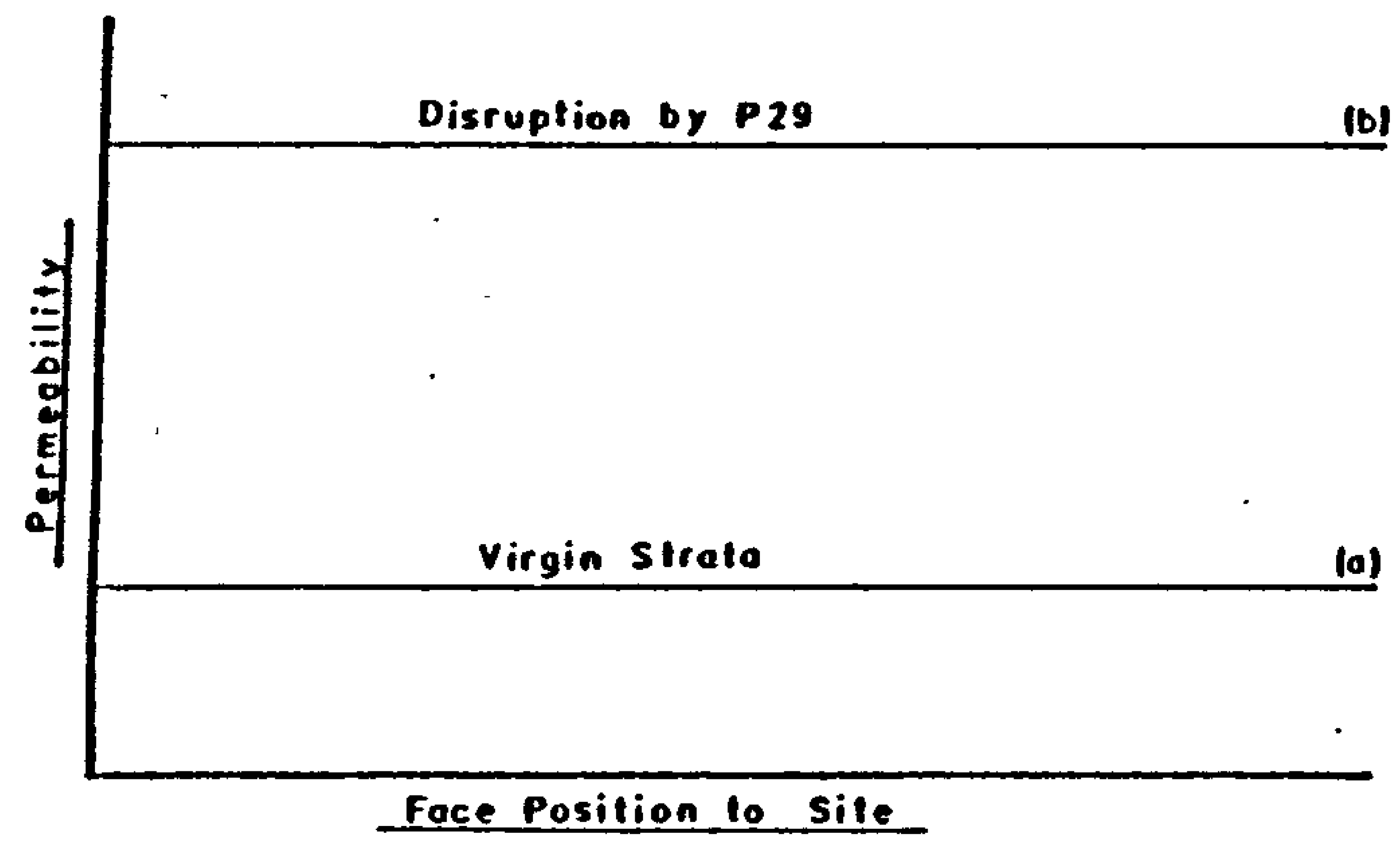


Figure 5.6.1

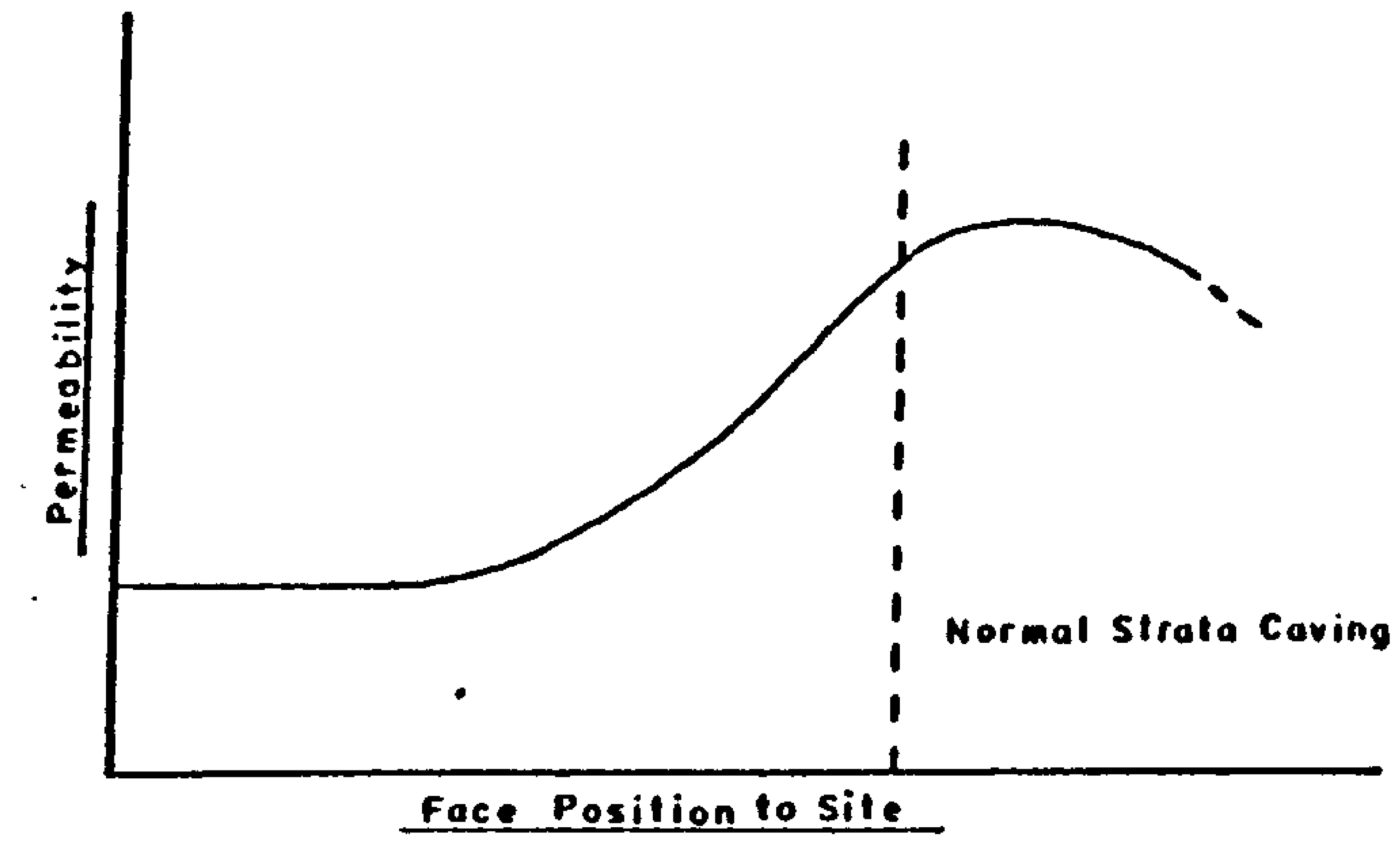


Figure 5.6.2

Figure 5.6.3

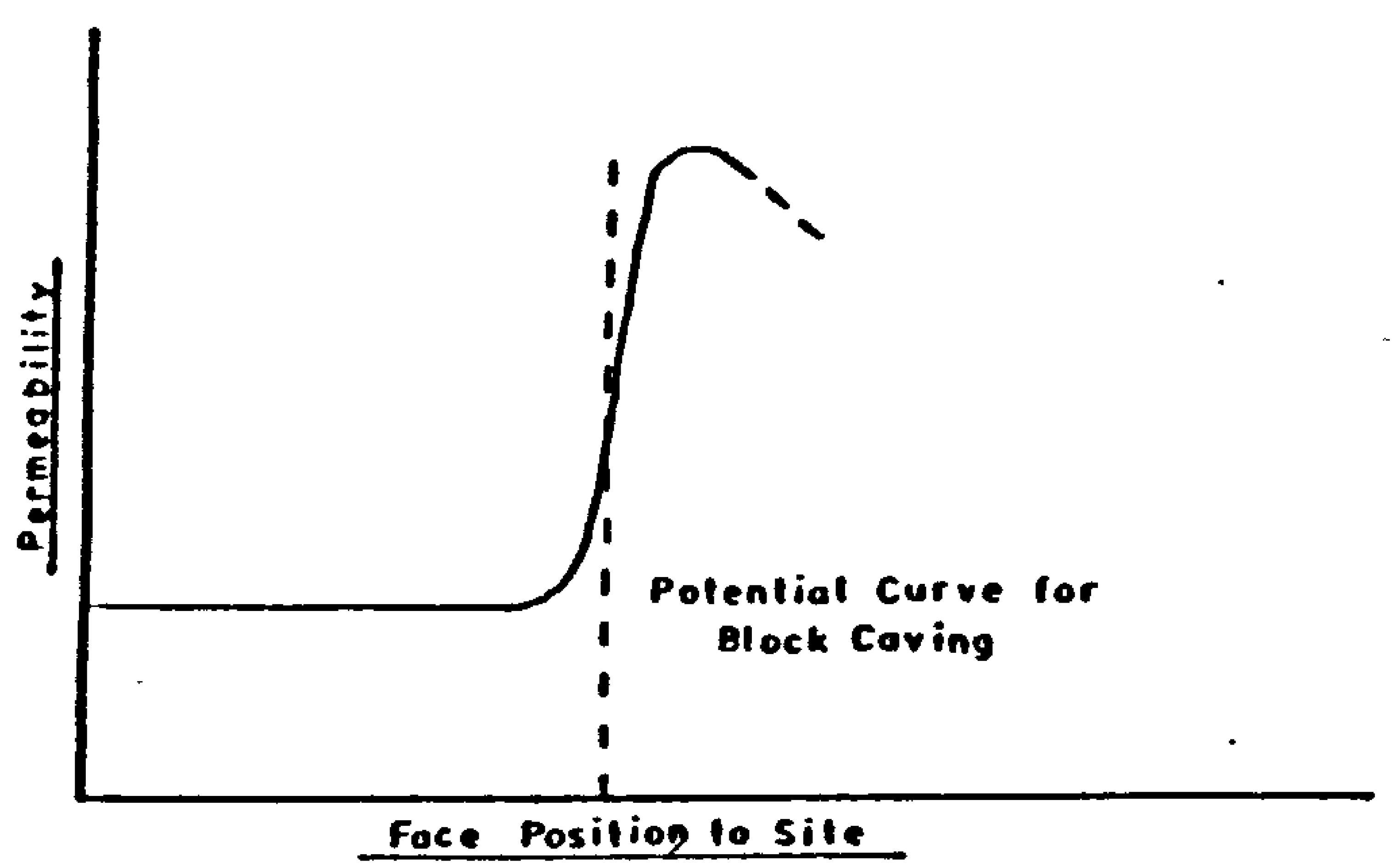


Figure 5.6.4

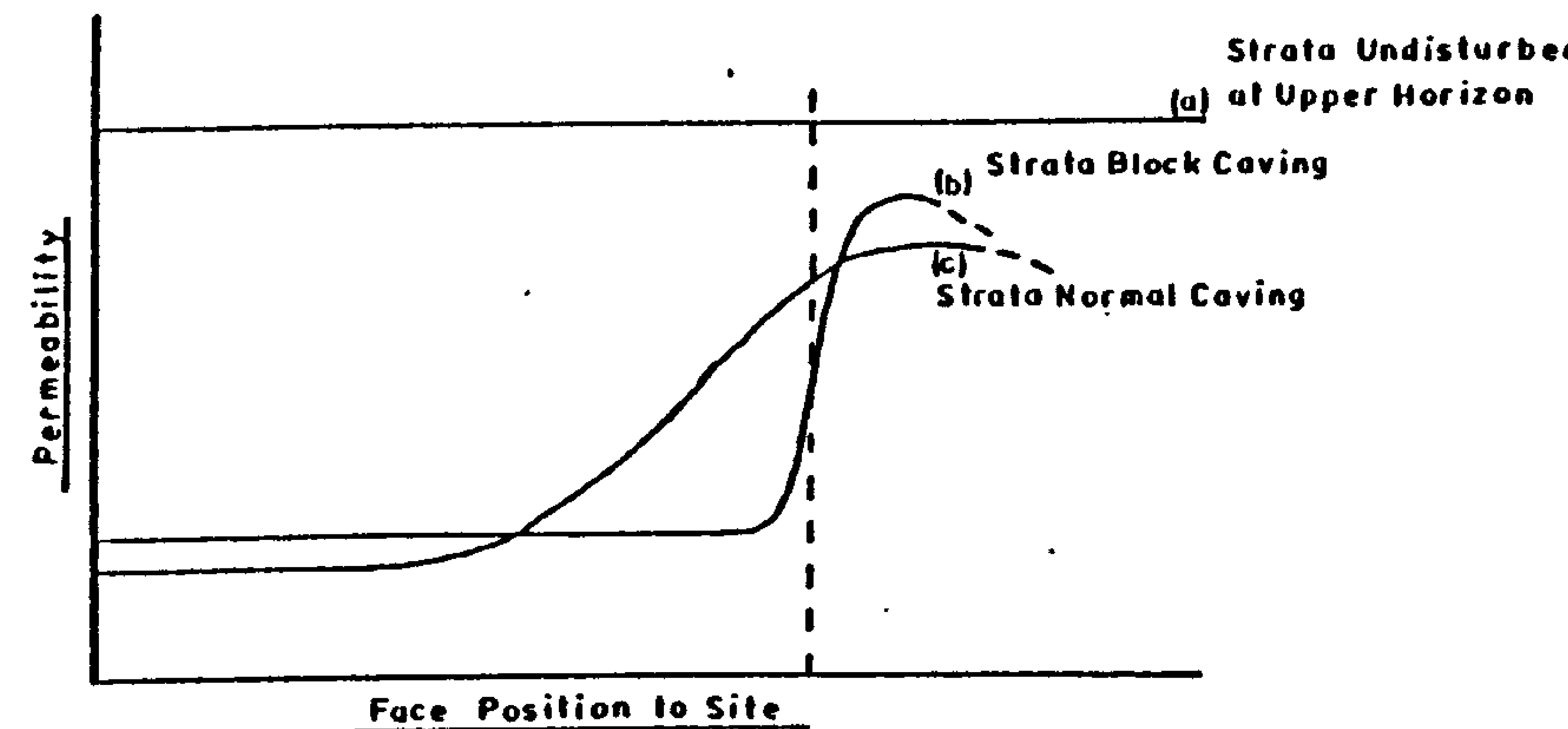


Figure 5.6 Theoretical Permeability-Face Position Curves for the Parkgate Rock

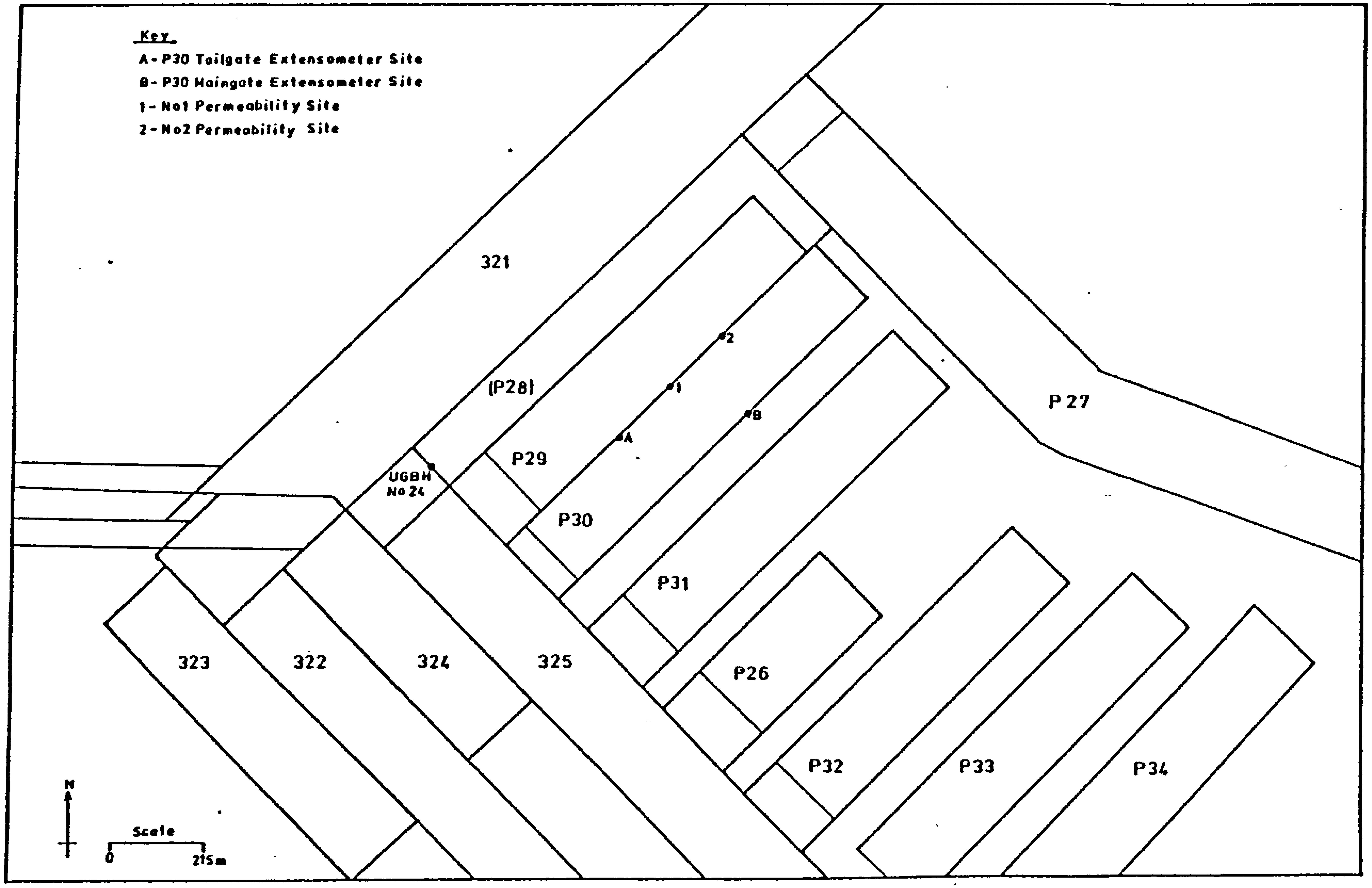


Figure 5.7 Location Plan for the East Parkgate Workings, Hickleton Colliery

higher, Figure 5.6.1(b). Dynamic conditions around an advancing face line will cause fracturing to occur with an associated increase in permeability, Figure 5.6.2. However, it has also been proposed that the Parkgate Rock might cave as 'massive blocks' because of its massive and competent nature. Permeability changes should therefore be able to determine whether or not this is happening. If blocking is occurring, low permeability values should be seen in advance and around the face vicinity, after which a rapid increase should occur due to intersection of the test cavity by fissure systems, Figure 5.6.3. Similarly, if the Parkgate Rock is sufficiently competent to remain intact above a given horizon, Figure 5.6.4(a), significant changes in permeability should be visible in strata below this horizon, Figure 5.6.4(b) (Blocking) and Figure 5.6.4(c) (Normal breakage). If however, natural breakage is occurring at all levels, a general increase in permeability should be visible in all test cavities, with increased face proximity. Results from the actual test cavities should therefore approximate to one of the theoretical conditions given in Figure 5.6. Finally, provided access to the site was permitted once the face had passed, it was hoped to assess the extent to which permeability was affected by goaf consolidation.

5.4.2 Design of Instrumentation

The proposed scheme of site instrumentation has already been outlined in Section 5.3.3. Design of the instrumentation to allow formation of the test cavities was kept as simple as possible, in order to eliminate the need for inserting mechanical sealing devices.

Design and dimensions of the No. 1 borehole instrumentation are shown in Figure 5.8. It consists of three lengths of yellow plastic

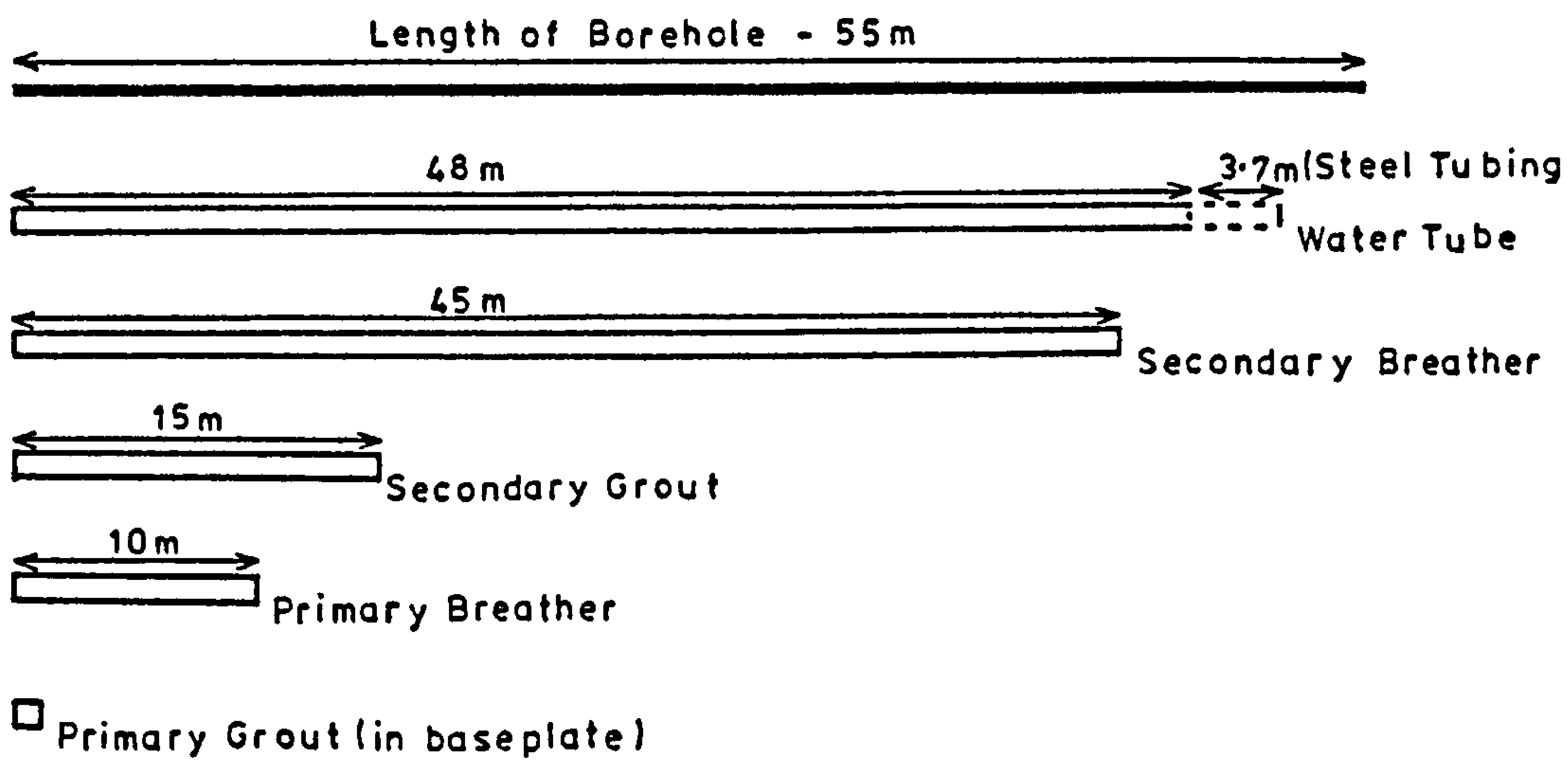
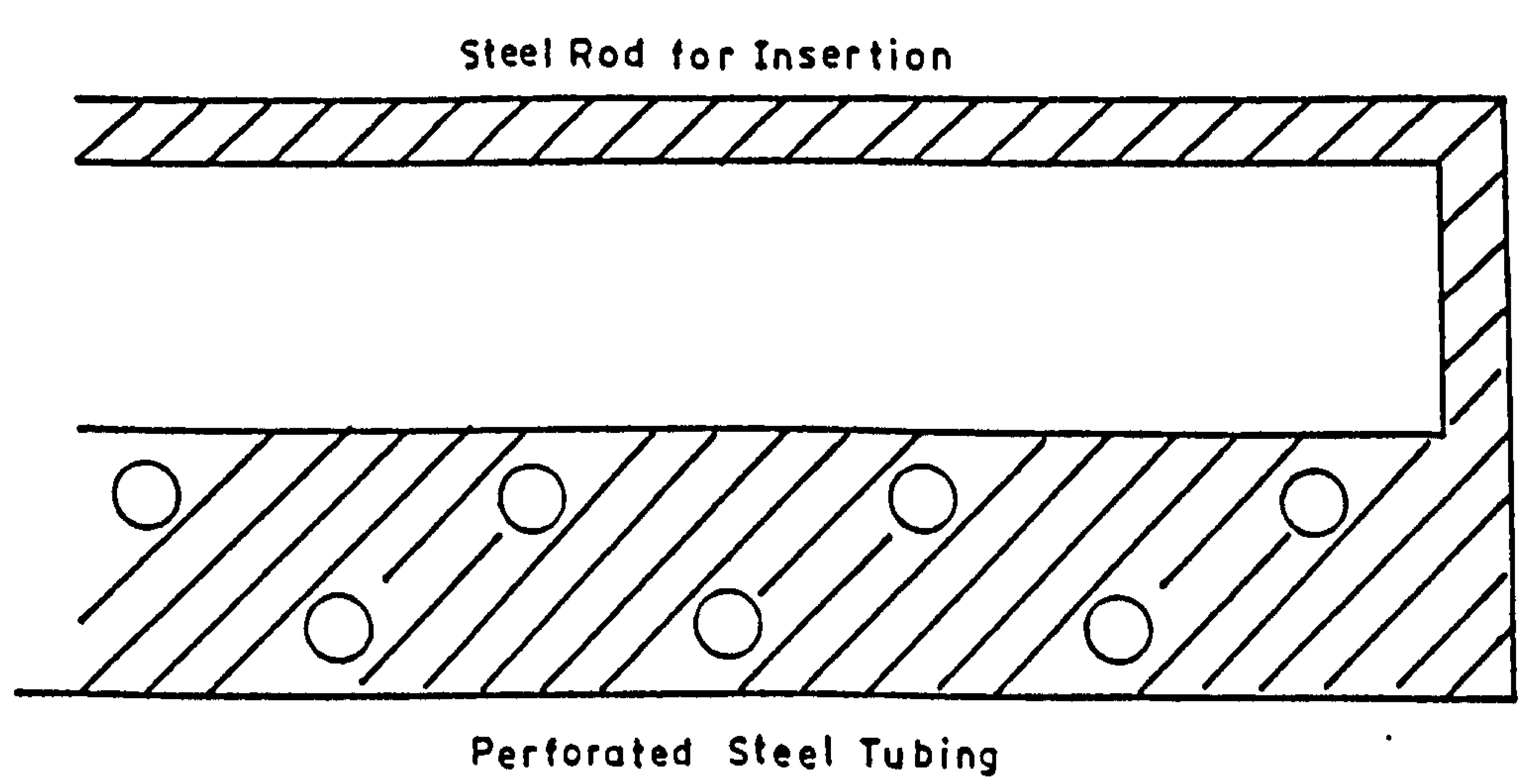


Figure 5.8 Instrumentation for Permeability Borehole No. 1

Figure 5.9 Insertion Attachment on Perforated Steel Tubing



tubing 15 mm internal diameter and 20 mm external diameter, supplied by Celtite-Selfix of Alfreton to NCB specifications. The first and longest tube, 48 m in length supplies water to the test cavity. Two perforated steel tubes, each 1.85 m long, are attached immediately prior to installation to this tube by means of two jubilee clips and tape, Plate 1. The second tube, 42 m long acts as a secondary breather and the third 12 m acts as a secondary grout tube. Nylon tubing 8 mm in diameter acts as a primary breather. The four tubes were measured out and bound together in triangular cross-section, in the Department laboratory. Numerous rolls of 12 mm x 20 m electrical insulating tape were used for binding, which is placed at 0.3 m intervals. After completion, except for the steel end tubes, the tubing was coiled and bound to allow easy transportation to both the colliery and underground site. Dimensions of the Nos. 1 to 4 borehole instrumentation are given in Table 5.3.

5.4.3 Site Installation

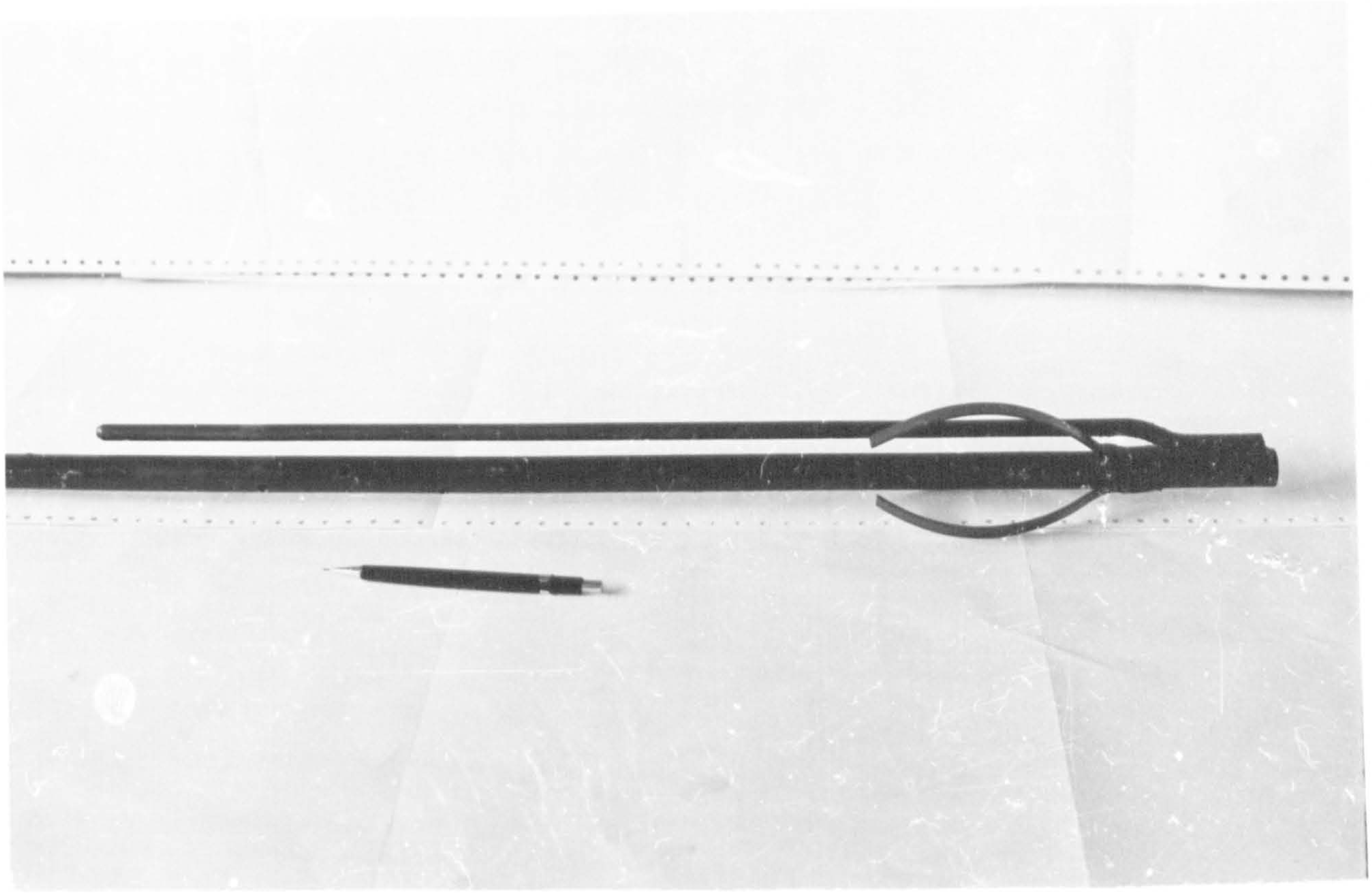
Once on site and ready to install, the tubing was unwound along the roadway and the perforated steel rods attached to the test cavity water tube. A steel rod which had previously been welded to the end of the perforated steel tube acted as an extension onto which could be slid the insertion rods, Figure 5.9. The perforated rods and tubing could then be installed into the hole by simply adding further insertion rods. When the tubing reached its desired position, the insertion rods were withdrawn. Weight of the tubing coupled with frictional forces between the tubing and borehole walls kept the instrumentation in place. Bottom members of the tubing were then threaded through a base plate and attached to the standpipe base.

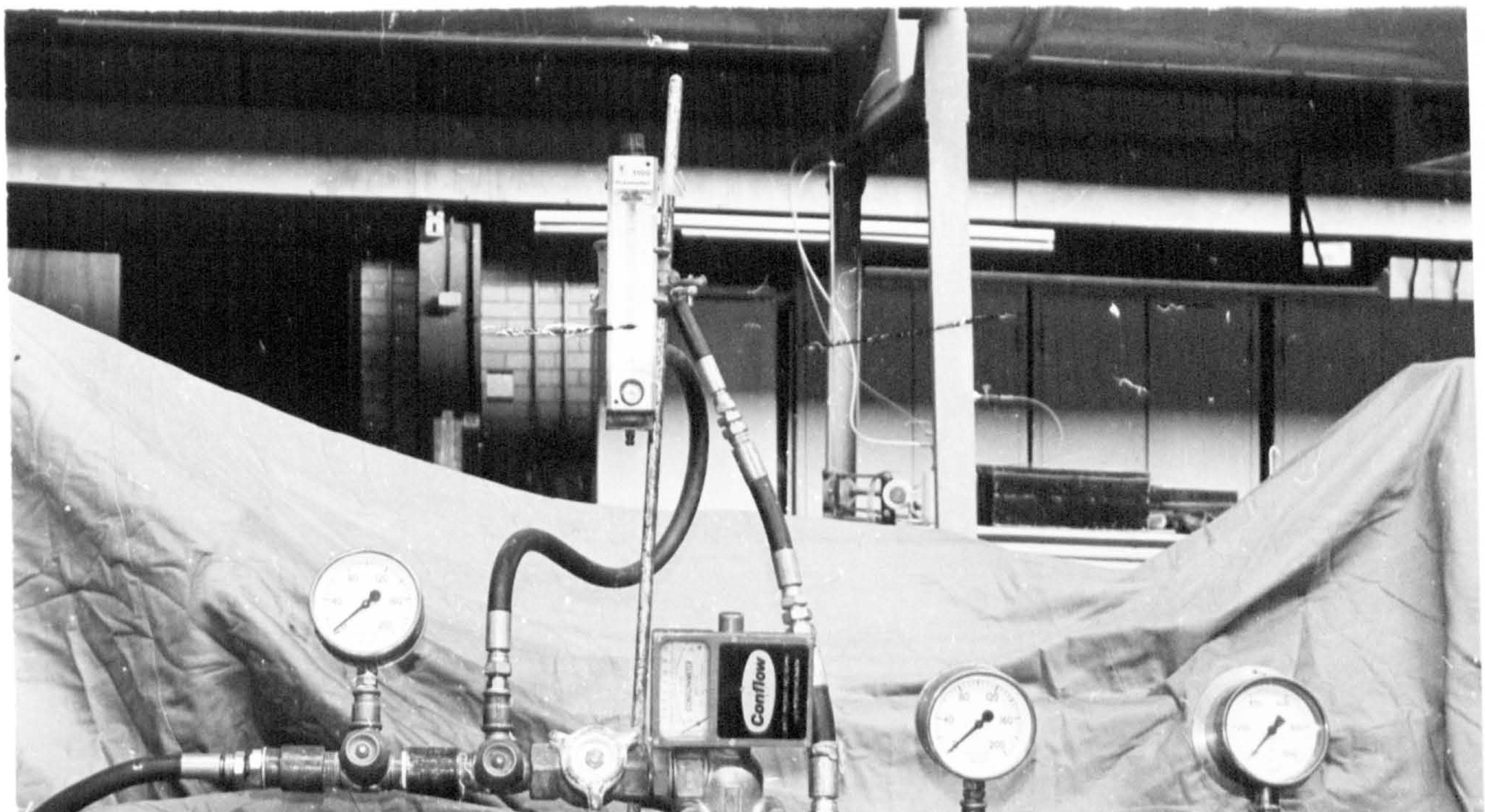
Plate 1

Perforated Steel Tubing showing
Insertion Rod and Guider

Plate 2

Mark I Pressure Testing Manifold




Conflow

			Length of Tubing (metres)			
Borehole Number	Total Length of Hole (m)	Inclination	Water Tube	Secondary Breather	Secondary Breather	Primary Breather
1	55	55°	48	45	15	10
2	45	45°	38	35	15	10
3	35	35°	28	25	12	8
4	30	25°	23	20	10	5

Table 5.3 Dimensions of the P30 Tailgate Permeability Instrumentation, Hickleton Colliery

Rubber washers were finally screwed into place over the protruding tubes to prevent leakage across the connection.

A coupling was attached to the base plate to allow grout, in this case Rotaset resin, to be pumped into the standpipe and up the hole. When grout reached the primary breather and emerged, it indicated that the hole was filled to this level. The primary breather was then sealed with a screw clip and the grout allowed to set. Next day, grout would be pumped via the secondary grout tube until it exited via the secondary breather. When a steady stream was emerging, the pump was switched off, the secondary breather sealed by means of a screw clip and the grout allowed to set. A test cavity had therefore been formed at the end of the borehole, with a tube allowing access to it.

Using this two stage grouting method, cavities and fissures around the base of hole and standpipe can be sealed before final completion of the hole. However, in practice the operation can be completed in one stage, whereby pumping is continued until grout emerges from the secondary breather, thus indicating that the hole is sealed to this level.

5.4.4 Pressurisation Equipment and Procedure

Once a cavity has been formed, it is necessary to pressurise it under steady state conditions before strata permeability can be determined. In order to achieve this, a pressure testing manifold had therefore to be designed and constructed. During testing, two manifolds were used:

The Mark I - Figure 5.10.1 and Plate 2,

and The Mark II - Figure 5.10.2.

Key

P1 - Pressure Gauge 0-13·8 MPa
P2 - " " 0-4·8 MPa
P3 - " " 0-2·7 MPa
P4 - " " 0-2·7 MPa
V - Valve

F1 - Flowmeter 4-114 l/min
F2 - Rotameter 0-4 l/min

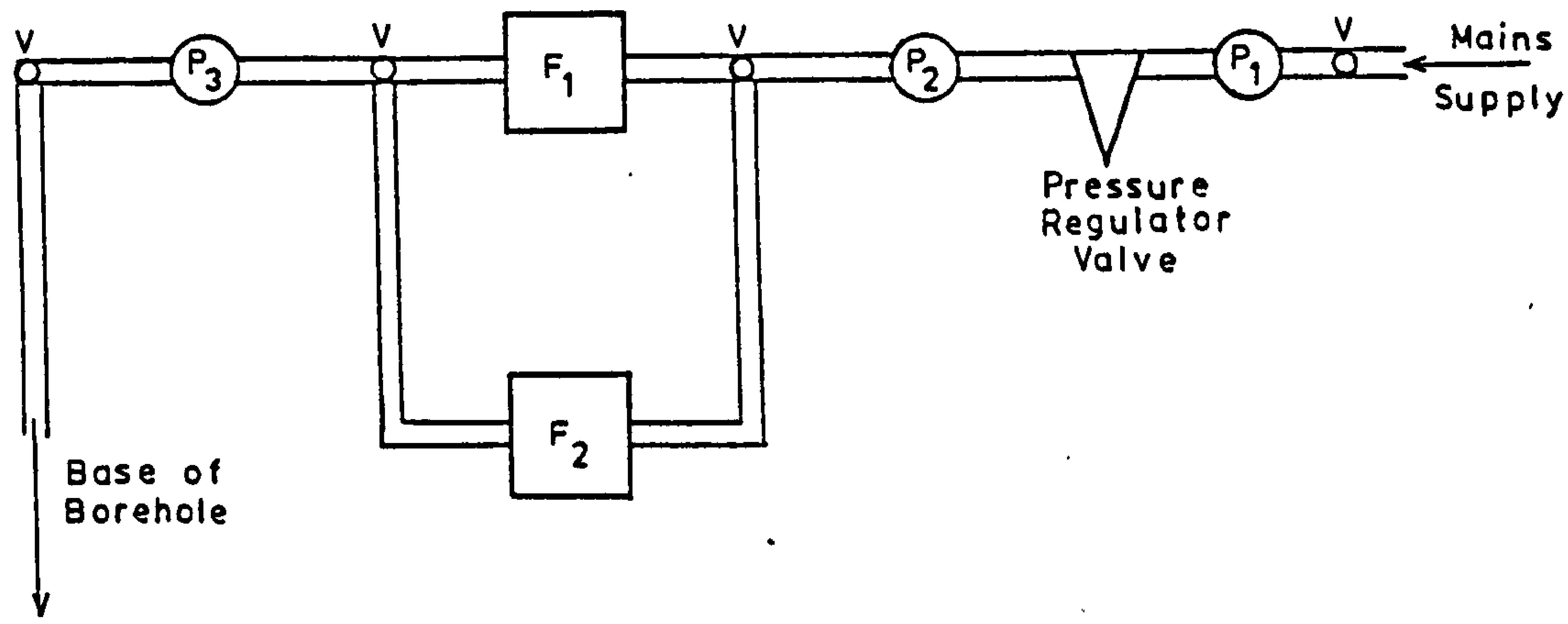


Figure 5.10.1 The Mark I Pressure Testing Manifold

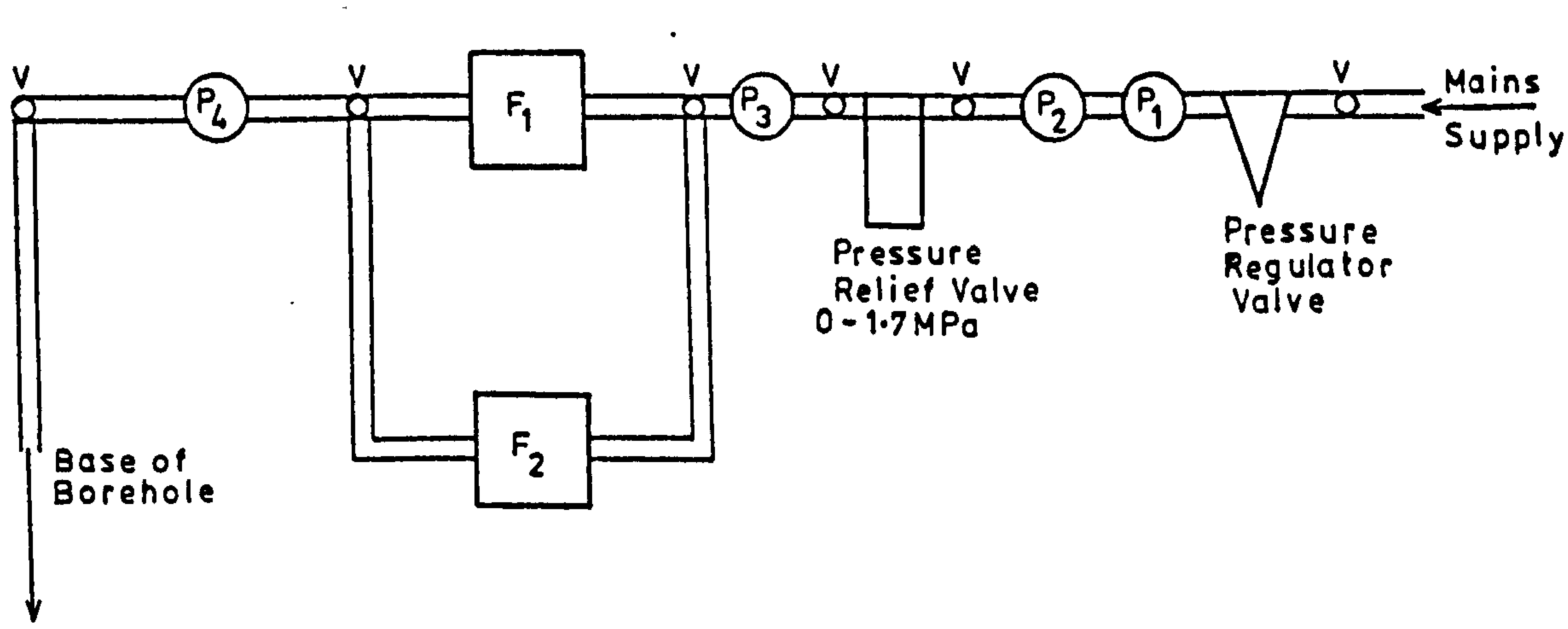


Figure 5.10.2 The Mark II Pressure Testing Manifold

Modifications were made to the Mark I after field trials on the No. 1 Borehole revealed several shortcomings. Both were similar in operation, but only the Mark II is described here.

The Mark II pressure testing manifold is shown in Figure 5.10.2. The system operates when a mains supply is connected and the stop valve opened. Mains pressure is registered on Gauge 1 (range 0 - 13.8 MPa) and reduced by means of the pressure regulator valve to about 2.1 MPa, when it is checked and further reduced to below 1.7 MPa using Gauge 2 (range 0 - 4.8 MPa). Flow is then allowed to pass through the pressure relief valve and across Flowmeter 1. Any pressure losses across the flowmeters can be checked on Gauges 3 and 4 (range 0 - 2.7 MPa). Flowmeter 1 is used to measure higher flow rates in the range 4 - 114 litres/min (1 - 25 gallons/min), while Flowmeter 2 (a Rotameter) is used for much lower flows in the range 0 - 4 litres/min (0 - 1 gallon/min). If flows are sufficiently low so as not to register on Flowmeter 1, they can be diverted through Flowmeter 2 and any pressure losses can still be determined using pressure gauges 3 and 4. The pressure relief valve has a range of 0 - 1.7 MPa and was installed to protect not only gauges 3 and 4 but also the Rotameter, should any major fluctuations in water pressure occur during testing.

In the event, water supply to the site had to be obtained from face supplies and was therefore subject to pressure fluctuations dependent upon the operation of face equipment. Insertion of the pressure relief valve was therefore a wise precaution, particularly after experiences during the No. 1 Borehole trials.

5.4.5 Borehole Permeability Testing at Hickleton Colliery

In this section, it is proposed to outline practical aspects encountered during installation and pressure testing of three inclined boreholes sited in the P30 tailgate, Parkgate seam at Hickleton Colliery.

5.4.5.1 The No. 1 Borehole Test

Equipment in No. 1 Borehole was installed by methods already described in Sections 5.4.3 and 5.4.4. However, initial pressurisation tests revealed two serious deviations from the expected final condition:

1) Water was found to issue from the secondary breather and grout tubes as well as in considerable quantities from strata surrounding the borehole.

2) Potential pressure fluctuations were underestimated and at one point a pressure of 6.9 MPa and flow of 36 litres/min (8 gpm) was entering the borehole. This in turn seriously damaged existing pressure gauges and required the acquisition of a new set.

Subsequent regrouting of the borehole resulted in the water tube becoming blocked, presumably with grout. Two questions therefore arise:

1) What happened to the grout initially pumped into the borehole?

2) Why did the water tube become blocked?

In answer to the first question, since the grout was obviously not in the borehole and very little appeared in the roadway, it must have entered the surrounding strata. It is therefore suggested that the borehole was intersected by either numerous or several large fissure systems which drained off grout at only a slightly slower rate than that at which it was being pumped in.

As regards the second question, two possible answers exist. Firstly, the water tubing was ruptured by excess pressures experienced during the initial test, allowing the ingress of grout. Secondly, the rate of rise of grout was sufficient during the time lag between it entering and emerging from the secondary breather tube, to continue climbing the borehole and enter the water tubing.

A redesign of the pressure testing manifold to the Mark II version, eliminated excess pressure. Similarly, two theoretical methods were devised for preventing blockage of the water pipe:

1) Fill the water pipe with water before grouting.

2) After grouting, pass compressed air up the water tube until it comes back out through the secondary breather tube. An open circuit therefore exists, which can be closed by adding a small quantity of grout to the lower section of the secondary breather tube and sealing it by means of a screw clip.

5.4.5.2 The No. 2 Borehole Test

Drilling was completed in the No. 2 Borehole on a Friday and the rig moved into position for the No. 3 hole, before leaving the site to stand over the weekend. Installation of the No. 2 Borehole was scheduled for first thing on Monday morning. However, on Monday the No. 2 Borehole equipment became inextricably wedged and due to a potential industrial relations problem had to be rapidly abandoned.

The equipment became wedged at 6.5 m and subsequent investigations revealed that with care insertion rods could be pushed past the perforated tubing and hence the blockage. It was therefore concluded, that strata movement over the weekend, possibly in the form of a small 'bump' had resulted in partial closure of the borehole.

5.4.5.3 The No. 3 Borehole Test

Experience on No. 2 Borehole, resulted in the drill being run up No. 3 Borehole immediately prior to installation. The hole was found to be clear and the equipment installed without problems. Grouting was then started from the base of the hole, but due to time and pumping difficulties, it was decided to seal the complete hole in one operation.

Almost immediately grout started to seep in increasing quantities from the surrounding strata. A significant quantity was yielded from a small fissure 0.6 m above the standpipe. Pumping ceased and 'hard stop' cement used to seal most of the leakages. Pumping was then continued until grout emerged from the appropriate breather tubes. However, when grout started to emerge from the secondary breather tube, leakage from the strata increased rapidly both in quantity and extent, with a significant amount occurring 1.5 m outbye of the hole. Calculations revealed that about 400 litres (90 gallons) of grout were pumped into the hole and that approximately 100 litres (20 gallons) seeped back into the roadway. At this point, it is worth noting that while drilling No. 3 Borehole water could be seen seeping from the strata upto 4 m outbye and 2 m inbye of the hole.

Another significant feature which occurred during grouting was the very small quantity of grout which emerged initially from the secondary breather tube. No amount of subsequent pumping, could again achieve grout emerging from this tube. This caused concern, particularly in view of the roadway leakage and experiences on No. 1 Borehole. It was rapidly concluded that grout was being lost to the surrounding strata almost as fast as it was being pumped into the hole. A calculated risk was therefore taken and the secondary grout tube opened, since a full head of grout should exist not only in the secondary grout tube but also in the hole between the secondary grout and breather tubes. Opening the tube produced only a very small quantity of grout, even though at the time of sealing a steady stream was emerging. It was therefore concluded that a significant loss of grout to the strata must be occurring below the top of the secondary grout tube, even though significant losses of grout to the roadway were not occurring. It was also thought that pressurisation of the hole would lead to a massive leakage of water from the strata into the roadway and this is in fact what happened.

The following day, pressurisation produced massive leakage from the strata both inbye and outbye of the hole as well as in the roadway roof. An attempt was made to measure borehole inflow using No. 1 Flowmeter and a stopwatch and strata leakage (primary feeders or droppers only) using a bucket and stopwatch.

Values in Table 5.4 should be treated with caution, since the figures are based on estimates and only intended to act as an indicator for the volume of water lost to the surrounding strata.

	Time (minutes)	Flow Rate (litres/min)	Total Volume (litres)
Borehole Inflow	3	45-64	135-192
Strata Leakage after cessation of pumping	5	4 to 0.3	20 to 1.5
Potential Loss to Strata	-	-	115-190

Table 5.4 Borehole No. 3 Inflow and Strata Leakage Volumes P30 Tailgate, Parkgate Seam, Hickleton Colliery

During pumping, water outflow from the strata was similar to torrential rain, while after cessation the quantity decreased to light rain and eventually droppers. Number 3 Borehole was then abandoned, since the drillers were required in another area of the pit for methane drainage problems and subsequently for installation of the Maingate Extensometer site.

5.4.5.4 Additional Borehole Testing

A high priority was given to installing the maingate extensometers, which resulted in cessation of further permeability work in the form of additional boreholes. Discussions with Area staff resulted in a revised Site 2 scheme for Boreholes 4-6, Figure 5.11. Similarly, experience with Boreholes 1-3, suggested that a 4.8 m standpipe was inadequate for penetrating broken ground around the roadway periphery. An increase in standpipe length to 12 m was therefore proposed for Boreholes 4-6. Rapid face advance also necessitated moving Site 2, a further 150 m outbye of Site 1 (Boreholes 1-3).

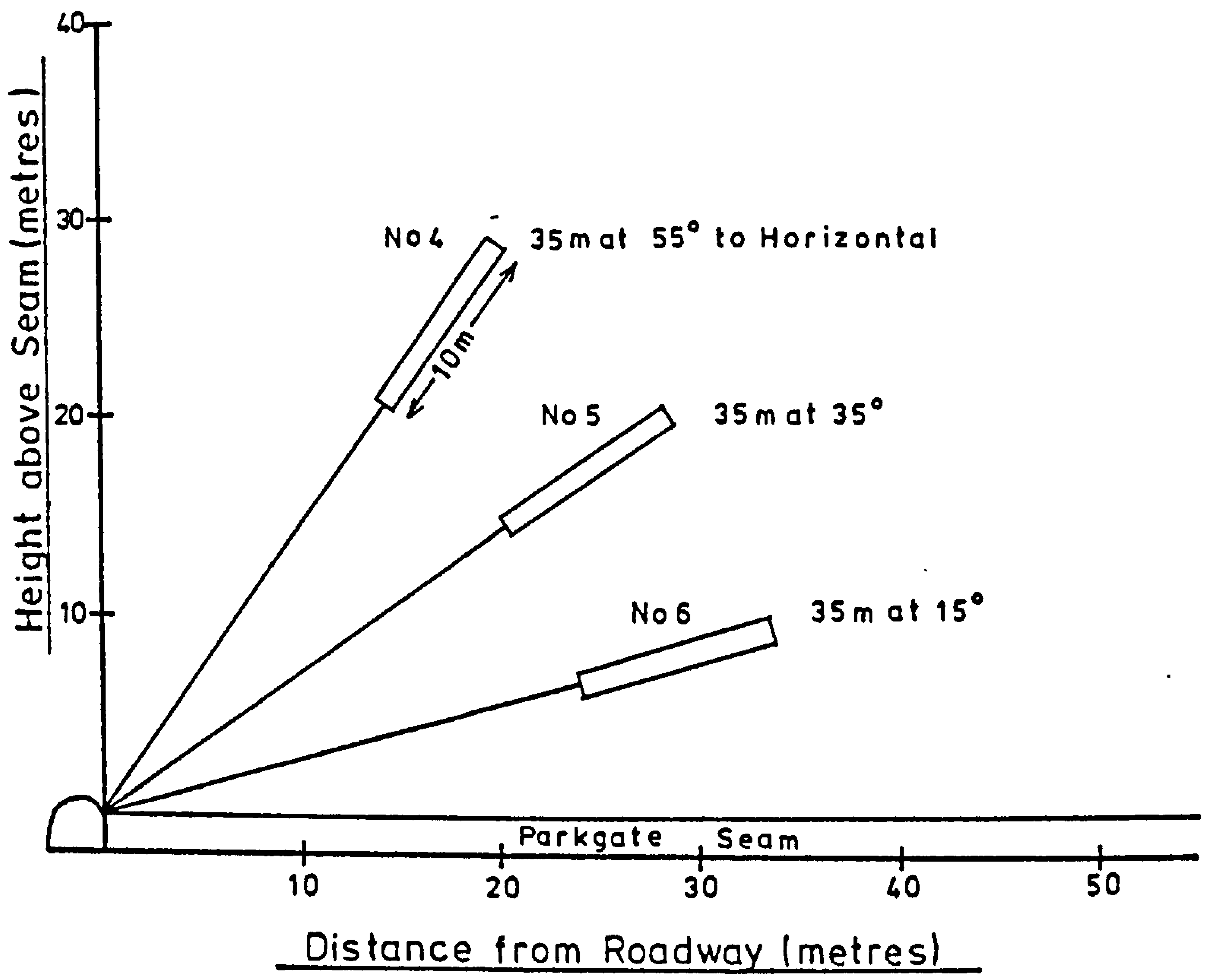


Figure 5.11 Permeability Site No. 2 (Revised Scheme) Boreholes Nos. 4-6, P30's Tailgate

However, before any work started on Site 2, it was decided as Area policy to abandon coal production in the East Parkgate and move all future operations across to the West Parkgate. As a result, all plans for continuation of the permeability work also had to be abandoned.

5.4.6 Evaluation of Additional Investigative Techniques

While at Hickleton, the author encountered several techniques which it is thoughtif used in conjunction with permeability testing, could provide valuable additional information. It is therefore proposed to outline these techniques briefly and highlight the beneficial affects obtainable when used for this type of investigation.

5.4.6.1 Drilling Operations at Site 1

Drilling rates experienced at Site 1 (Boreholes 1-3) were exceptionally low and although no detailed figures exist, records indicate weeks rather than days for the completion of each hole. This can be attributed mainly to the type and age of the machine used.

The drilling machine used, was in the region of 40 years old and driven by compressed air. However, the nearest available supply was in a service main in the P32 tailgate, Figure 5.7. Compressed air therefore had to be piped from the P32 tailgate, along P27's tailgate and down P30's tailgate to the site, with a resultant drop in final operating pressure. Similarly, water had to be taken directly from face supplies, which resulted in fluctuating rates and frequent disagreements with face operatives.

It is interesting to compare the tailgate drilling rate of weeks per hole with that for 3 holes sited in the maingate. Two 33 m upholes and one 15 m downhole were completed in the space of 3 days, using a modern electrically operated machine, driven by mains supplies available in the maingate. If such a machine had been available in the tailgate, then not only would the Site 1 holes have been completed sooner, but also in the light of eventual experience at Site 1, these holes could have been redrilled at Site 2 with little additional effort. Similarly, instead of a final proposed scheme of 6 boreholes, the final scheme could have envisaged a comprehensive investigation of up to 12 boreholes.

Drilling operations can therefore play an important role in the eventual success or failure of a given investigation.

5.4.6.2 Dye-Tracing Techniques

It was noticed during initial pressure testing of Borehole No. 1, that water was seeping from the partially completed Borehole No. 2. A connection was therefore thought to exist between these boreholes, even though they had different spatial locations. Existence of such a connection could prove invaluable not only in assessing the fractured nature of the strata, but also the validity of numerical techniques used to calculate permeability at these sites.

To prove the existence of a connection, it was proposed that dye should be added to the pressurisation water and a watch kept for its emergence from the No. 2 Borehole. Fluorescein dye was to be used and would be added by placing a large quantity inside the pressure monitoring system before pressurisation. Fluorescein is a particularly persistent and easily noticeable dye, even in very low concentrations, due to its fluorescent green colouration. Its inert nature (non-toxic and incombustible) make it ideal for use underground, although this was ascertained by consultation with Area Scientific staff.

In addition to simply determining whether water was penetrating from No. 1 to No. 2 Borehole, it was also hoped to determine the approximate horizon at which this connection occurred. This could be achieved by inserting the insertion rods to the top of the No. 2 hole prior to testing and then withdrawing during testing by known lengths at discrete time intervals, until dyed water was seen coming from the base of the rods.

As already described in Sections 5.4.5.1 - 4, this experiment proved impossible to conduct. However, it should be seriously considered for future investigations of a similar nature, where connections between adjacent test cavities can be proved by a simple and inexpensive technique.

5.4.6.3 Borehole Deviation Monitoring Techniques

It is normally assumed that actual and theoretical locations of a test borehole are one and the same. However, borehole deviation during drilling operations usually occurs, which results in the actual intersected test horizon being different from the predicted. A comprehensive knowledge of the test area geology combined with detailed logging of either borehole cores or cuttings can often accurately locate the final intersected horizon, although this method is subject to potential errors.

An alternative method is to log the borehole using an orientation and deviation monitoring instrument. Test Borehole No. 1 was logged using such a device, in order to determine whether a difference existed between the actual and predicted test cavity locations. The survey was undertaken by the Area Survey Office, using a Sperry-Sun Magnetic Multishot Directional Survey Instrument, NCB Doncaster Area Survey Department (60).

Basically, the equipment obtains a complete directional survey in an uncased borehole, free from extraneous magnetic effects. The instrument consists of a compass-angle unit, film magazine, actuating mechanism, electrical power supply and a precision time control device, all of which are housed in a protective, non-magnetic

Frame No.	Compass Brg.	Inc °	Grid Brg.	True Inc °	Mean Inc °	Slope Dist.	Diff in Height	Plan Dist	Mean Brg.	Defl. from set out	Defl. from Course	Dist along Course
			46.83°SE	55.00°		19.67'	19.67'	11.28'	46.83°SE		0'	11.28'
376	35 NW	37	43 SE	53.00		19.67						
					53.375		28.00	5.97	43.5 SE	3.33°	0.35	17.24
384	36 NW	36¼	44 SE	53.75		29.67						
					53.75		36.06	5.91	44 SE	2.83°	0.64	23.14
396	36 NW	36¼	44 SE	53.75		39.67						
					54.50		44.20	5.81	44 SE	2.83°	0.93	28.95
404	36 NW	34¾	44 SE	55.25		49.67						
					56.00		52.49	5.59	44 SE	2.83°	1.20	34.53
414	36 NW	33¼	44 SE	56.75		59.67						
					57.375		60.91	5.39	44 SE	2.83°	1.47	39.91
422	36 NW	32	44 SE	58.00		69.67						
					58.50		69.44	5.22	44 SE	2.83°	1.72	45.13
432	36 NW	31	44 SE	59.00		79.67						
					59.50		78.06	5.08	44 SE	2.83°	1.97	50.20
441	36 NW	30	44 SE	60.00		89.67						
					60.00		86.72	5.00	44 SE	2.83°	2.22	55.19
450	36 NW	30	44 SE	60.00		99.67						
					60.00		95.38	5.00	44.5 SE	2.33°	2.42	60.19
459	37 NW	30	45 SE	60.00		109.67						
					59.875		104.03	5.02	44.5 SE	2.33°	2.63	65.21
468	36 NW	30¼	44 SE	59.75		119.67						
					58.375		112.54	5.24	46 SE	0.83°	2.70	70.45
477	40 NW	33	48 SE	57.00		129.67						
					56.375		120.87	5.54	48.5 SE	-1.67°	2.54	75.98
486	41 NW	34¼	49 SE	55.75		139.67						
					55.125		129.07	5.72	49.75 SE	-2.92°	2.25	81.70
495	42½ NW	35½	50.5 SE	54.50		149.67						
					53.750		137.14	5.91	50.25 SE	-3.42°	1.90	87.60
504	42 NW	37	50 SE	53.00		159.67						
					52.750		145.10	6.05	50 SE	-3.17°	1.56	93.64
513	42 NW	37½	50 SE	52.50		166.67						
					52.50	171.67	149.06	3.04	50 SE	-3.17°	1.39	96.68

End of hole

Note: Difference in height is from top of seam.
Distance is from standpipe.
All distances in feet.

Table 5.5 Inclination and Deviation Data for Permeability Borehole No. 1, P30's Tailgate

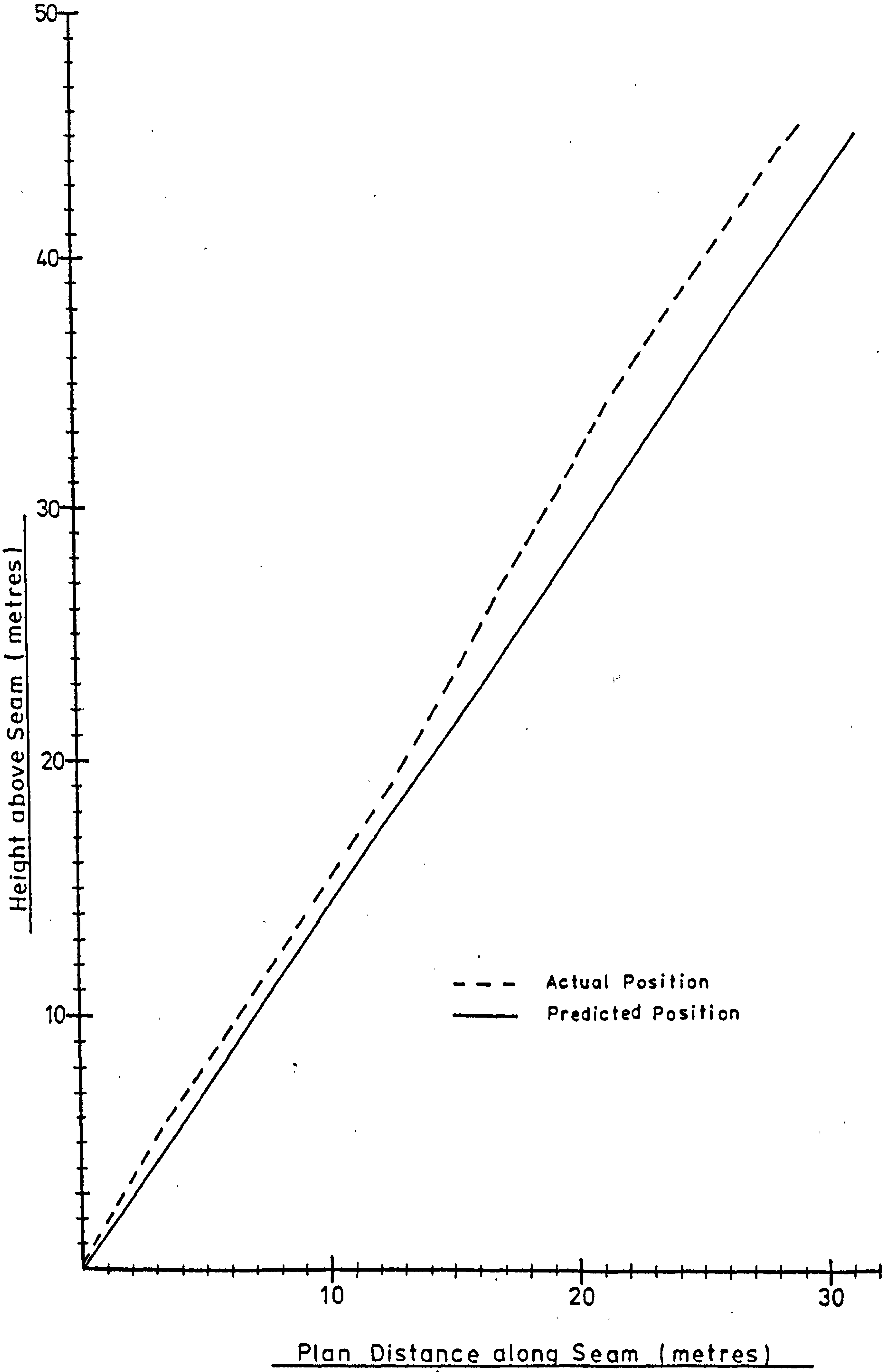


Figure 5.12 Actual and Predicted Positions of Permeability Borehole No. 1, P30's Tailgate

casing, Plate 3. Electrical power is supplied by batteries and the time control device is designed such that with a variable programmer, it can record for periods of up to several days. The instrument can be inserted into boreholes in a number of ways, but with horizontal and vertically upholes, it is normally attached to the end of drill rods. The film magazine (or camera) takes pictures of a compass unit face, Plate 4, at regular predetermined time intervals, when a lighting mechanism is operated. An actuating mechanism then moves the film onto the next frame. Records are taken at regularly timed intervals within the borehole, with the operator recording the distance and time of each survey point. This information is essential for reading and interpreting the film at a later date. After completion of a survey, the film is developed, read by means of a projector and the data analysed. This procedure ensures an accurate correlation between distance along the hole and the inclination and deviation of each survey point.

Results from a survey in No. 1 Borehole (Section 5.4.5.1) are presented in Table 5.5, while a plot of actual and predicted positions is given in Figure 5.12. Examination of Table 5.5 reveals that the borehole inclination (True Inclination) has remained remarkably consistent throughout drilling operations. Similarly, Figure 5.13 shows good agreement between the actual and predicted locations of the borehole and hence test cavity horizons.

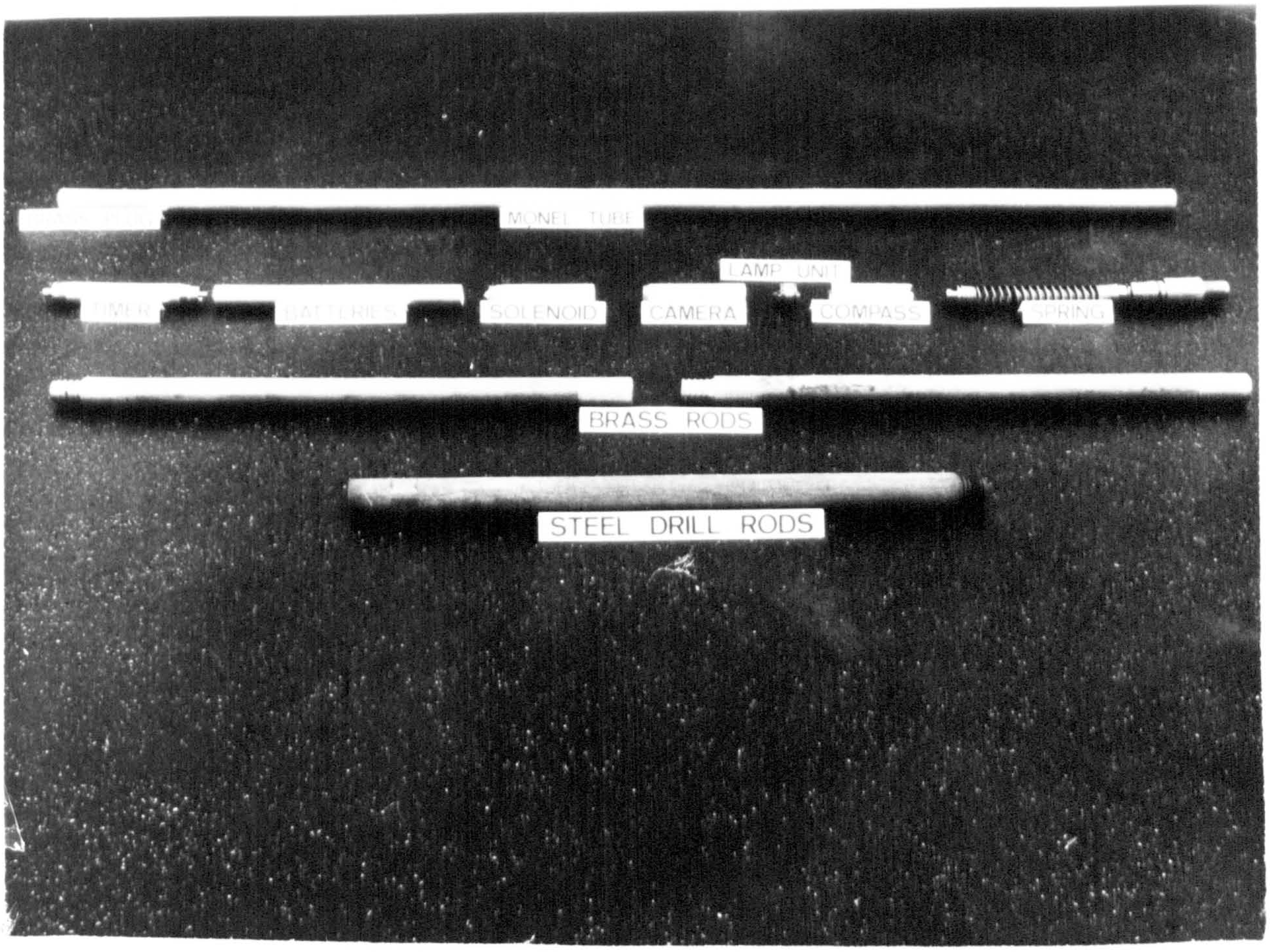
Ideally, this type of survey should be conducted on all boreholes where an accurate location is essential for final data interpretation. Although this procedure is not always possible for practical or operational reasons, it should be attempted on at least one borehole

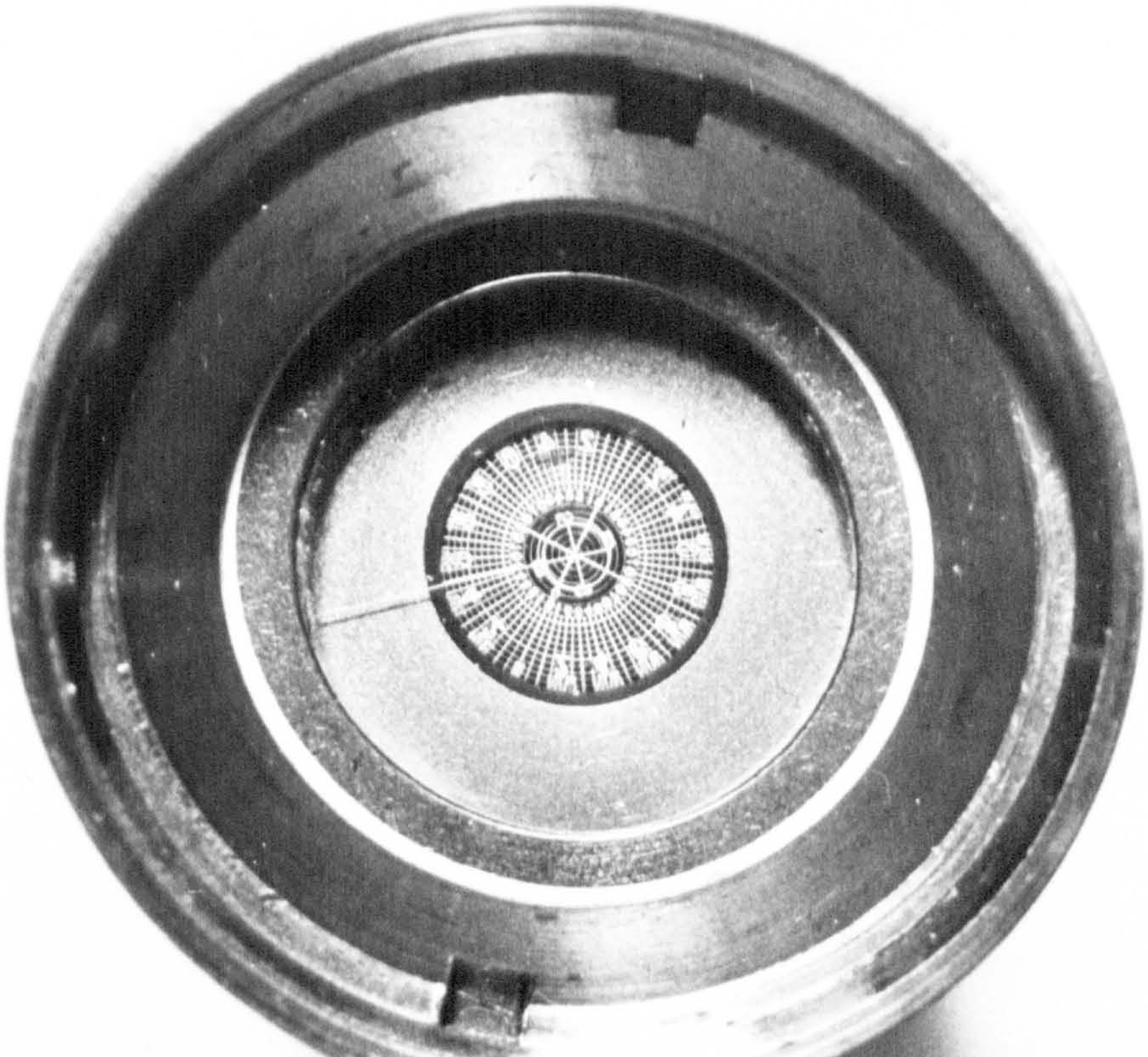
Plate 3

Components of the Sperry-Sun Magnetic
Multishot Directional Survey Instrument

Plate 4

Compass Unit of the Sperry-Sun
Directional Survey Unit


MONEL TUBE
LAMP UNIT
TIMER
BATTERIES
SOLENOID
CAMERA
COMPASS
SPRING
BRASS RODS
STEEL DRILL RODS



in the scheme, in order to assess the amount of potential deviation occurring during drilling operations.

5.5 Conclusion

Usually, the choice of an instrumentation site is restricted to that area of colliery workings where problems are being experienced. However, if one is fortunate enough to be able to choose from one of several potential sites, the final choice is not necessarily easy. Full account of the investigation objectives as well as any practical difficulties likely to be encountered during installation and monitoring, should also be considered.

Similarly, when designing an instrumentation scheme it is essential to maximise use of the often limited resources. Wider implications of the proposed investigation should be considered, in addition to identifying and solving the immediate problems. All work so far undertaken by the Department has examined permeability changes above an extraction panel, as well as in adjacent ribside pillars. None has yet attempted to monitor potential changes in strata permeability beneath the seam, which it is felt could have important implications. The proposed Whitwick scheme was designed with this shortcoming in mind.

Once a suitable site and investigation scheme has been finalised, it is necessary to explore fully potential problems likely to be encountered, before deciding on the final equipment design. Ideally, trials should be conducted beforehand or a design adopted which has already been proved successful. However, unspecified site and variable geological conditions can often significantly affect the final outcome. At Hickleton, it is thought with hindsight that loss of grout from the boreholes to the surrounding strata could have been overcome by using 12 m instead of 4.8 m standpipes. Similarly, the pressure monitoring manifold originally used at previous sites, proved unsuccessful during initial trials on the No. 1 Borehole.

Finally, external factors such as drilling operations can significantly influence the final success or failure of an investigation. Similarly, new or normally unrelated techniques, such as dye tracing and borehole deviation monitoring, should be used in conjunction with existing schemes, in order to provide additional information and establish their use as alternative field instrumentation techniques.

CHAPTER 6

CAVING CHARACTERISTICS ASSOCIATED WITH A MASSIVE SANDSTONE ROOF

CHAPTER 6

CAVING CHARACTERISTICS ASSOCIATED WITH A MASSIVE SANDSTONE ROOF

6.1 Introduction

In the authors opinion, insitu permeability changes are inextricably linked with the caving characteristics of strata surrounding longwall extraction panels. It is therefore necessary to fully understand both the mechanics of insitu permeability change and strata subsidence into a goaf area before empirical methods can be developed to predict the eventual changes experienced.

A detailed strata control investigation was undertaken by the author at Hickleton Colliery, NCB Doncaster Area to determine caving characteristics of the Parkgate Rock (a 30 m sandstone sequence) and relate this to the historical occurrence of 'weight' bumps* in the area. This work therefore provides a unique opportunity to examine and assess the caving properties of a massive sandstone sequence lying immediately above the working seam. Insitu permeability monitoring techniques were also undertaken during the investigation and these are described in Chapter 5.

Initially, it is proposed to examine the historical background which led to initiation of the investigation, since this lays the foundation on which the final research objectives were developed.

---

* A weight bump is the violent release of strain energy from the surrounding strata, which results in sudden forward displacement of the seam or the projection of coal. Shock waves, bump-like noises and spalling or breaking of the coal and/or strata can also occur.

Instrumentation and monitoring techniques are described as well as analysis and interpretation of the results with respect to the final research objectives.

6.2 Historical Significance of 'Weight' Bumps

A history of 'weight' bumps in the Parkgate seam of the Doncaster area and in seams of other NCB areas, has long been associated with workings immediately overlain by thick sandstone sequences. During the mid-1960's, a series of 'bumps' in the Parkgate seam at Hickleton Colliery, coupled with a major accident in the same seam at the nearby Barnborough Colliery in 1942, Humphrys (61), brought renewed interest and concern about the nature of these events, particularly in the light of future planning proposals.

6.2.1 'Weight' Bump Characteristics

'Weight' or 'rock' bumps are generally associated with the sudden release of high stress concentrations in seams at depth (greater than 700 m) which are overlain by a thick sequence of competent roof strata, such as sandstone or strong siltstones. Bump conditions are not usually found in British mining conditions, but are frequently encountered in North America, West Germany and South Africa, Bryson (62) and the Rock Burst Prevention Development Committee Symposium, Essen (63). Other examples of British bump seams apart from the Parkgate, are the Thick Coal of South Staffordshire (64) and the Great seam at Monktonhall, Scotland.

Characteristic features of a bump are the sudden and/or violent spalling of coal from the face, floor upheaval, air blasts and an increase in gas emission. This however, should not be confused with an 'outburst', which is also related to abnormal stress conditions but not with competent roof strata. 'Outbursts' tend to be restricted to the emission of coal and gas, while 'weight' bumps can also result in the displacement of supports and face machinery, Shepherd and Kellet (65) and Plates 5 and 6.

The magnitude of a bump is dependent primarily on the strength of roof and floor strata confining the seam. Coal extraction results in a build up of stress, which because of the competent strata cannot be transferred or relaxed by normal caving mechanisms as would occur in softer more normal strata.

6.2.2 The Barnborough Bump

In 1942, at Barnborough Colliery, a bump in the Parkgate seam led to the death of 4 men. It took the form of violent floor upheaval, which partially closed many of the roadways bisecting an area of bord and pillar working.

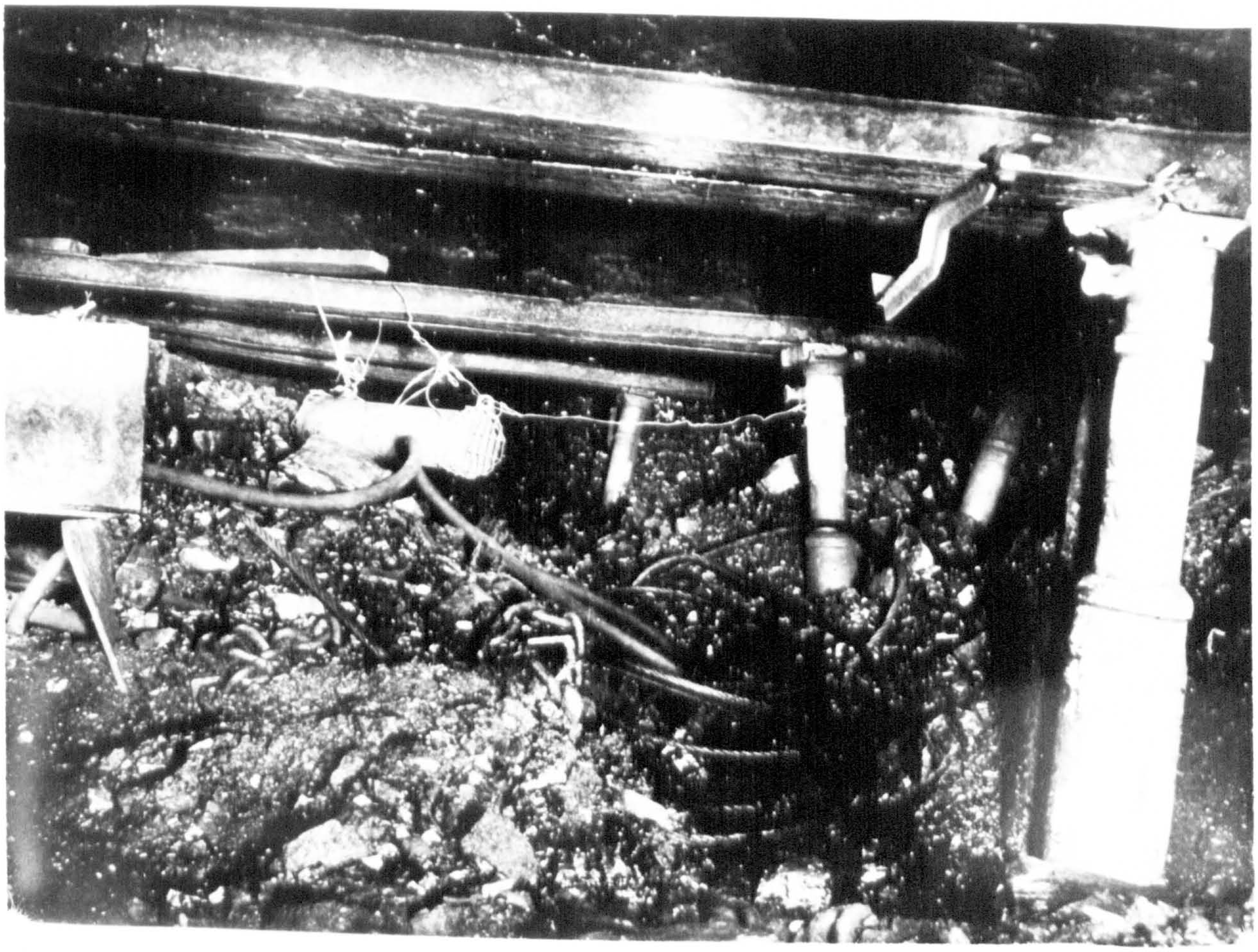
Prior to the bump, the workings consisted of a principal coal pillar 340 m by 265 m which had been divided into secondary pillars by the drivage of roadways at 20 m centres, Figure 6.1. Complete extraction had also occurred in the south-east portion of the area. An irregular pillar, varying in width from 10 m to 60 m extended some 180 m from the southern edge of the principal pillar and a smaller triangular pillar also existed 150 m south-east of the principal pillar, Johnson (66).

Plate 5

Effects of the Weight Bump on 324 Face.

East Parkgate Area, Hickleton Colliery

Plate 6



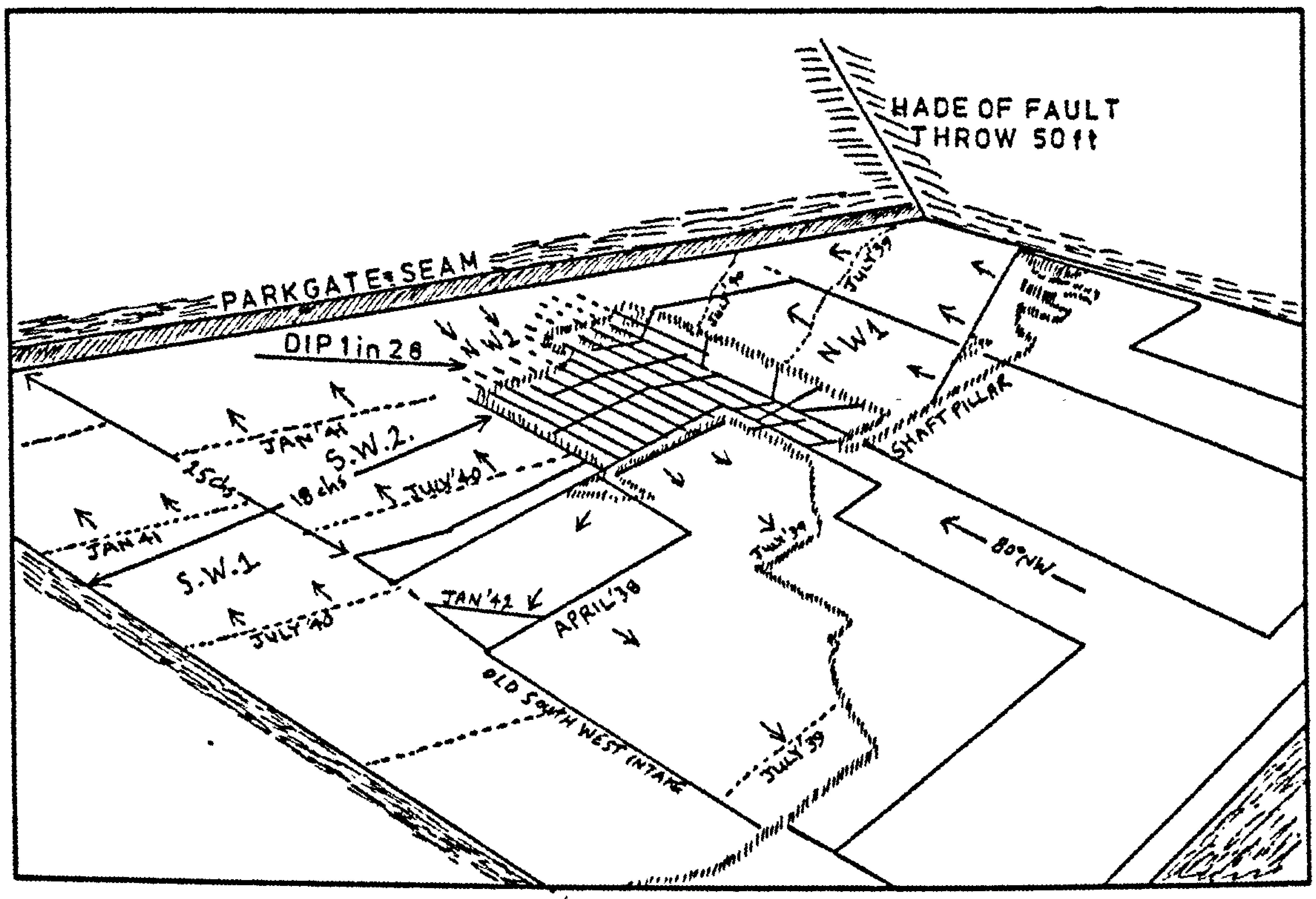
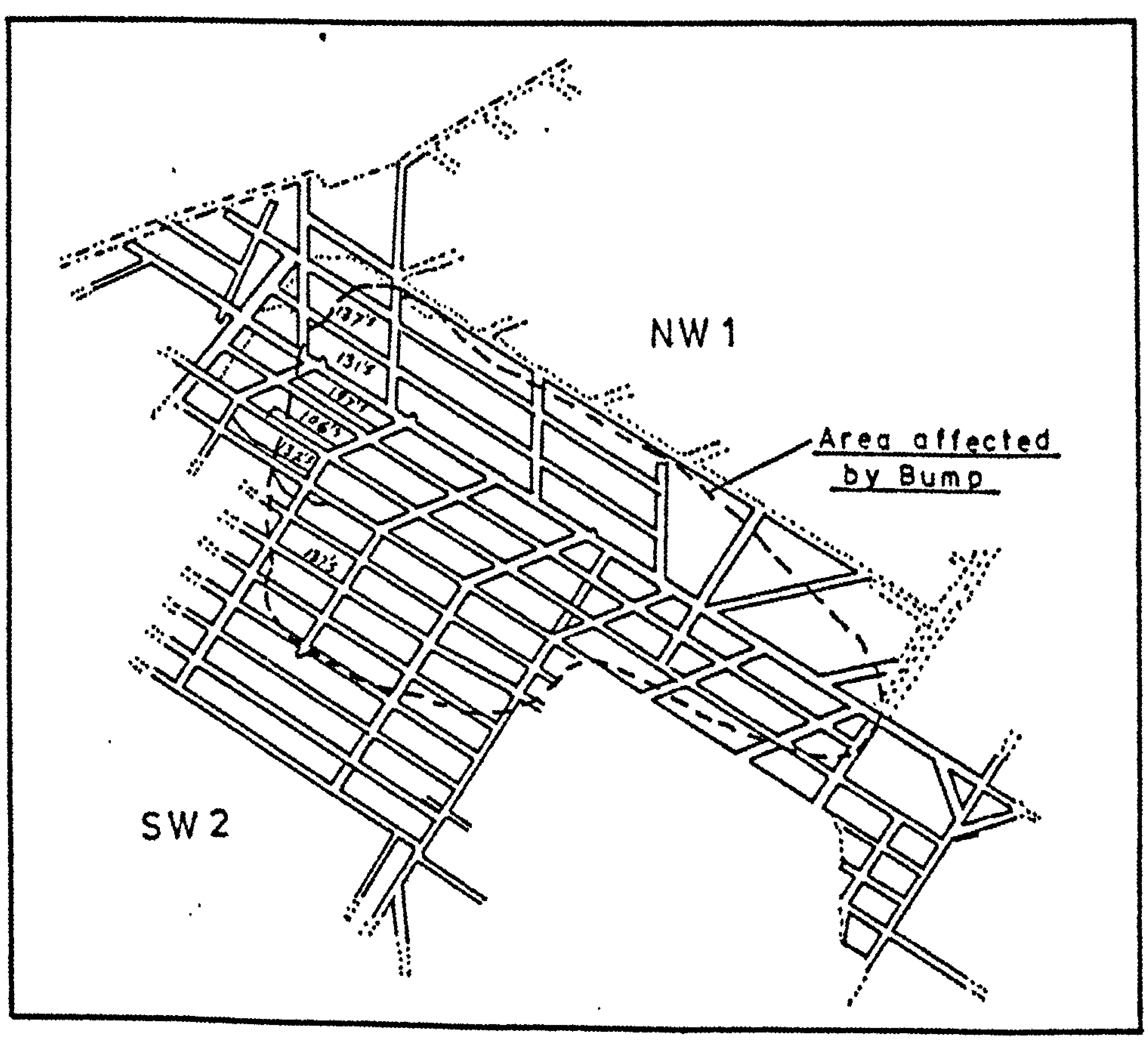


Figure 6.1 Plans of Barnborough Colliery showing the area where the Weight Bump occurred
(after Phillips (68))



Apart from the principal pillar being well bisected by roadways, other contributing factors were thought to be: the depth of working (700 m), competent roof strata (35 m of Parkgate Rock), a strong coal and roof, and the presence of remnant pillars.

The official inquiry concluded that the bump resulted from sandstone strata subsiding into goaf areas surrounding the principal pillar. This resulted in a cantilever action which caused upward deflection of strata immediately overlying the principal pillar and the formation of a shallow dome like structure. Eventually, fracture of the Parkgate Rock surrounding the principal pillar resulted in a rebound of the domed strata, which caused violent upheaval of the floor but neither disruption nor outburst of the coal. The inquiry therefore deduced that the bump was not caused by pillar failure but by a rebound mechanism associated with the overlying domed Parkgate Rock, Humphrys (61).

On reflection, it is interesting to speculate whether the mechanism put forward by the inquiry in 1942 is in fact valid. Current thinking, Johnson (66), suggests that the incident was primarily the result of reducing the main pillar size, with a resultant increased load being placed upon it. The two smaller remnant pillars could also have played an important role, since with increasing load from the periphery workings, these would have become highly stressed. If one of these then failed, an instantaneous transfer of load to the main pillar would occur, which although insufficient to cause complete failure could result in massive floor upheaval. This theory is supported by the fact that the remnant pillar to the south of the main pillar had been severely crushed, which suggests failure.

6.2.3 The Hickleton Bumps - 324 and 325 Panels

Workings in the East Parkgate at Hickleton Colliery, Figure 6.2, consisted of an initial 321 panel, followed by reorientated 322, 323, 324, and 325 panels which advanced in a south-easterly direction. In 1964, a series of bumps occurred on the 324 panel and these were subsequently investigated by MRDE and the results reported by Shepherd and Kellet (65).

Bumps on the 324 panel occurred at the face end adjacent to the 322 goaf. Initially a 6 m pillar separated the 324 and 322 units, but after the first bumps this was reduced to 4 m and extra 50 ton Desford supports introduced. However, the bumps continued and the face was finally abandoned.

Initial investigations revealed that the bumps may be due to high stress concentrations resulting from superimposed flanking abutment pressures associated with the 322 and 324 panels. In addition, the 324 panel was also found to be undermining a remnant pillar in the overlying Barnsley seam and this could be contributory factor. Finally, by cutting 324's face towards the 322 goaf, a steadily decreasing web of increasingly highly stressed coal could have acted as the trigger for the final 'major' bump. The bumps caused violent coal spalling from the face, dislodgement of face supports (props and bars) and displacement of the coal cutter from the face conveyor, Plates 5 and 6.

Commissioning of the 325 unit coincided with investigations carried out by MRDE, as well as replacement of the 'prop and bar' face support system with Gullick six-leg 50 ton chocks and subsequently Dobson six-leg 30 ton chocks, Shepherd and Kellet (65). The MRDE

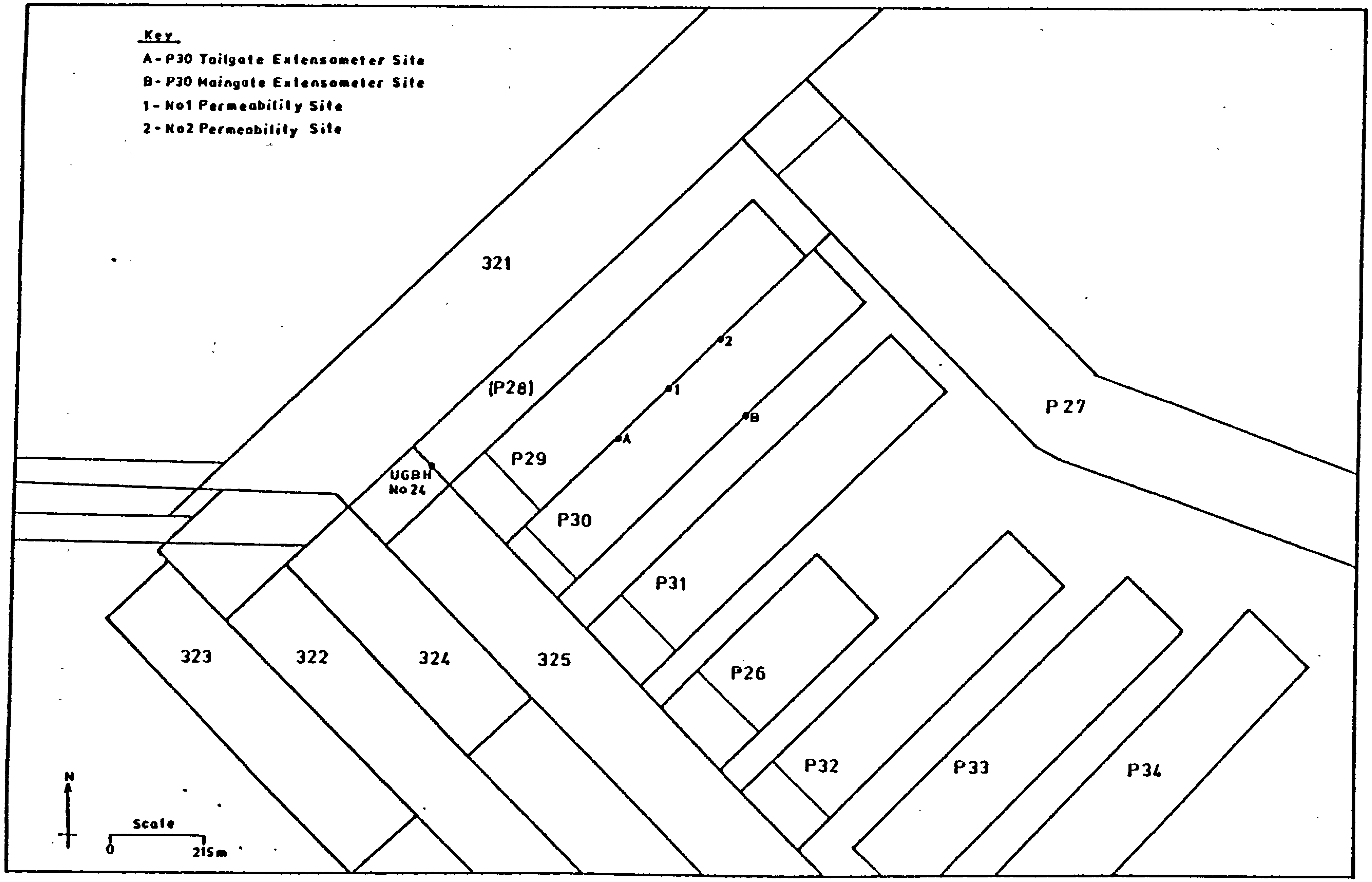


Figure 6.2 Location Plan of the East Parkgate Workings, Hickleton Colliery

investigation concluded that the 325 panel fulfilled most of the conditions necessary for weight bumps to occur, namely a massive sandstone roof, strong coal and a face advancing parallel to a goaf area, resulting in high cumulative stress concentrations along the boundary. Some minor bumps did occur on 325's until it passed beyond the influence of 324's, after which a period of uneventful working occurred. No bumps, were experienced on the P27 or subsequent panels. Remnant pillars in seams overlying 324's, but not 325's may have been a contributory factor. Similarly, the effect of replacing the 'prop and bar' face support system with mechanised powered supports could not be fully evaluated. It was therefore concluded, that when working beneath a massive sandstone roof, coal pillars of sufficient width should be left between panels, in order to alleviate the cumulative effects of stress caused by adjacent flank abutment pressures.

6.2.4 The Proposed P30 Panel

During 1973-74, it was proposed that a pillar of coal existing between the P29 and P31 goaf's should be utilised and a production face installed, Figure 6.2. However, a total pillar width of 255 m and planned face width of 162 m would leave a 91 m pillar against one of the adjacent goaf areas. This caused concern, particularly after the bumps experienced on the 324 and 325 panels, Section 6.2.3.

A reappraisal of work undertaken on 324 and 325, revealed two further factors which may have contributed to the bump conditions. Firstly, direction of cleat within the seam, and secondly, the width of the adjacent goaf when bumps were experienced. After considering both factors it was concluded that:

1) since P30 would be a 'bord' face, heavy spalling could be expected along the face line which would constantly displace stress concentrations into the solid and effectively de-stress the face.

2) to the north-west of P30 (Figure 6.2) lies a 183 m goaf (P29), a 120 m pillar and a 256 m goaf (321). To the south-east lies a 183 m goaf (P31), a 64 m pillar and a 201 m goaf (P26). At a depth of 860 m it was thought that the 64 m pillar might have crushed and resulted in the formation of a 448 m goaf. To work the P30 panel along side the P31 goaf would therefore create potential bump conditions based on the 324 and 325 'wide goaf' experience. However to the north-east, the 120 m pillar between P29 and 321 should still be stable and therefore eliminate the 'wide goaf' conditions which apparently cause bumps.

A Barnsley seam pillar exists above the north-west edge of the proposed working, but is not thought to significantly effect the situation.

It was finally decided that the most favourable layout for the P30 panel, in order to avoid bump conditions, would be to work a 162 m panel immediately adjacent to the P29 goaf and leave a 90 m pillar between the P30 panel and P31 goaf, Internal MRDE Report (67). Early in 1979 production started on the P30 panel and from its location in Figure 6.2, is obviously based on the previously mentioned criteria.

6.2.5 The Proposed P28 Panel

In 1979, further plans were drawn up for working a remnant coal pillar between the 321 and P29 goaf areas. However, many reservations were expressed concerning the advisability of such a proposal, especially in the light of bumps on the 324 and 325 panels and the considerations agreed for safe working of the P30 panel.

The argument against working the P28 panel, centres around the fact that although two adjacent panels can be safely worked beneath the Parkgate Rock, the extraction of a third results in the formation of bump conditions. Additional evidence for this theory was put forward with work done by Phillips (68) who suggested that the amount of stored strain energy in a cantilever is 36 times greater than that for a beam of the same dimension. Therefore, if it is assumed that the Parkgate Rock acts as a beam, bridging one normal face width (200 m) and caving across two adjacent face widths, the working of a third will affect the abutment zone supporting a cantilever, which in turn is capable of storing or dissipating vast amounts of energy, Internal MRDE Report (67).

It was also suggested, that not only does the Parkgate Rock act as a cantilever, but so too does one or possibly two more thick sandstone sequences situated above the Parkgate Rock. At Barnborough, of 180 m of strata above the Parkgate seam, over 60% was either sandstone or siltstone. It is therefore possible that two or three cantilevers existed above the goaf at both Barnborough and Hickleton and contributed to the bump conditions. An idea of the size of these cantilevers is given by Humphrys (61):

'It is certain that the rock beds overlying the adjacent areas would not break off close alongside the pillared area, but there would be considerable overhang - possibly between 30 m (100 ft) and 60 m (200 ft).'

If such strata conditions exist, it is possible that such a cantilever system could exist over the south-east side of the P28 panel, even if full subsidence is recorded between the P29 and P30 goafs. This situation would be even worse if no or only partial subsidence occurred above the P29 and P30 goaf areas. Similarly, no consideration has yet been given to possible effects caused by the large 321 goaf to the north-west of P28.

In addition, two further points also need to be carefully considered. Firstly, when P28 starts it will be removing a pillar on which one, two or possibly three strong sandstone sequences, each some 30 m thick, are resting. These sandstones are probably hogged over the pillar or bent into the adjacent goaf areas by the rock mass resting on their flanks. Also, the onset of first weight will result in an ever increasing length of sandstone beam, which at some point must break. This in turn could trigger a bump in the highly stressed sandstone and coal ahead of the face.

Secondly, as the face nears the end of its life, a situation will arise which is similar to that which occurred at Barnborough. A T-shaped pillar will result, where one limb is being removed until there is no longer sufficient support to contain the stresses imposed on the other two limbs, Internal MRDE Report (67). This theory is supported by work done in 1935 by Rice (69), who warned that bump conditions could be expected whenever: cover depths exceeded 300 m; a massive roof exists along with strong roof strata and coal; and the extraction of too much coal results in greatly increased loads on the remaining pillars. Phillips (68) also refers to 7 bumps in the Sonnenschein seam during the First World War. Each occurred during the mining of small remnant pillars completely surrounded by goaf, which but for the increased output caused by the war, would have been left. Pillar sizes varied between 45 m and 137 m in length and 18 m and 55 m in width. The seam was immediately overlain by 20 m of sandstone and in a total sequence of 112 m above the seam, 99 m were sandstone.

Although some mitigating circumstances do exist, in that there are no roadways cutting the proposed P28 panel, it was concluded that the attendant risks in working P28 could not be justified. Similarly, the view that floor lift in gateroads allows slow release of high stress concentrations and therefore a decreasing risk of bumps may be erroneous. Humphrys (61) states that:

'two men were dinting a roadway in the affected district to make height',

which suggests that roadway closure was occurring prior to the bump, Internal MRDE Report (67). Also, since no methods are currently available for measuring the state of stress in a rock mass, in order to give imminent warning of a bump, the plans to work P28 panel should be abandoned.

However, at this point two important factors came into play. Firstly, at a meeting on 11th October 1979, it was stated that the area needed to work this face and in the light of previous bumps on

324 and 325 how could this be most successfully achieved. Secondly, the Colliery Manager asked MRDE to re-examine the situation, because P30's waste appeared to be caving normally and heavy weighting was becoming apparent in the P30 tailgate behind the face. A visit by MRDE staff to the face and gateroads confirmed the behaviour as more typical of normal conditions and suggested that perhaps some attempt should be made to monitor the amount of subsidence within the Parkgate Rock and overlying sequences. At this point, the Department of Mining Engineering, Nottingham University was asked to conduct a comprehensive strata control investigation into caving properties of the Parkgate Rock above the P30 panel, in order to determine conditions likely to be encountered over the P28 panel.

6.3 Research Objectives

Strata control investigations initiated on the P30 panel at Hickleton were implemented around four main research objectives:

1) Analysis of strata behaviour on P30, in order to assess conditions likely to be encountered on P28. If it was decided to work P28, the necessary precautionary measures could be established at this stage.

2) It has been stated, Internal MRDE Report (67), that 'As there are no methods available to measure the state of stress in the rock mass in question, in order to give imminent warning, of catastrophe (a weight bump), we would urge ....'. It is therefore necessary to devise and establish the instrumentation techniques required for monitoring behaviour and caving characteristics of massive sandstone roof conditions.

3) A long history of bump conditions in European Coalfields has led to the development of techniques for assessing potential hazards associated with high stress concentrations. In particular, the West German method of 'in-seam' drilling could lead to its immediate application for the prediction of bumps in British mining conditions.

4) It is currently recognised that future workings in British mines will be going deeper into seams overlain by thick sandstone sequences. If research is initiated to understand the strata mechanics involved and develop suitable instrument monitoring techniques, when these seams are worked, considerable amounts of time, money and possibly lives will be saved.

6.3.1 Research Procedure

Early in 1980, a four phase investigation scheme was drawn up for discussion and eventual approval by the various parties involved in the P28 proposal, namely: Hickleton Colliery, NCB Doncaster Area, MRDE and Nottingham University. However, by the end of 1980, a change in area planning policy terminated all future extraction in the East Parkgate at Hickleton, which effectively curtailed all existing and planned research activity. Fortunately, by this time all four phases had been completed, either successfully or otherwise and supplementary schemes were being planned.

The four phase scheme is listed in chronological order:

1) tailgate extensometers - P30.
2) tailgate permeability measurements - P30.
3) maingate extensometers - P30.
4) geological investigations - P28.

The tailgate permeability work has already been discussed in Chapter 5. Similarly, geological and geotechnical investigations carried out on P28, as well as secondary investigations on P30 and an assessment of West German 'in-seam' drilling techniques are discussed in Chapter 7. It is therefore proposed to concentrate the remainder of this chapter on extensometer work carried out in the P30 main and tailgates and assess strata behaviour above the P30 panel.

6.4 East Parkgate Location and Geology

6.4.1 The P30 Location

The location of Hickleton Colliery is given in Figures 6.3 and 6.4. The P30 and P28 panels under investigation are situated in the Parkgate seam on the eastern side of the colliery workings (East Parkgate). A diagram of the East Parkgate, showing all relevant workings is given in Figure 6.2.

Two main panel series have been worked. The first trends north-east to south-west and comprises panels 321, P29, P30, P31, P32, P33 and P34, while the second trends north-west to south-east and comprises panels 322, 323, 324, 325 and P27.

Panel P30 has a face length of 165 m, extracted seam height of 1.5 m, depth of working of 900 m and was worked by the Z-retreat method. The old P29 maingate (production ceased in 1973) was refurbished and used as the tailgate, while the maingate was driven in-line with the face. On the maingate side, the panel runs

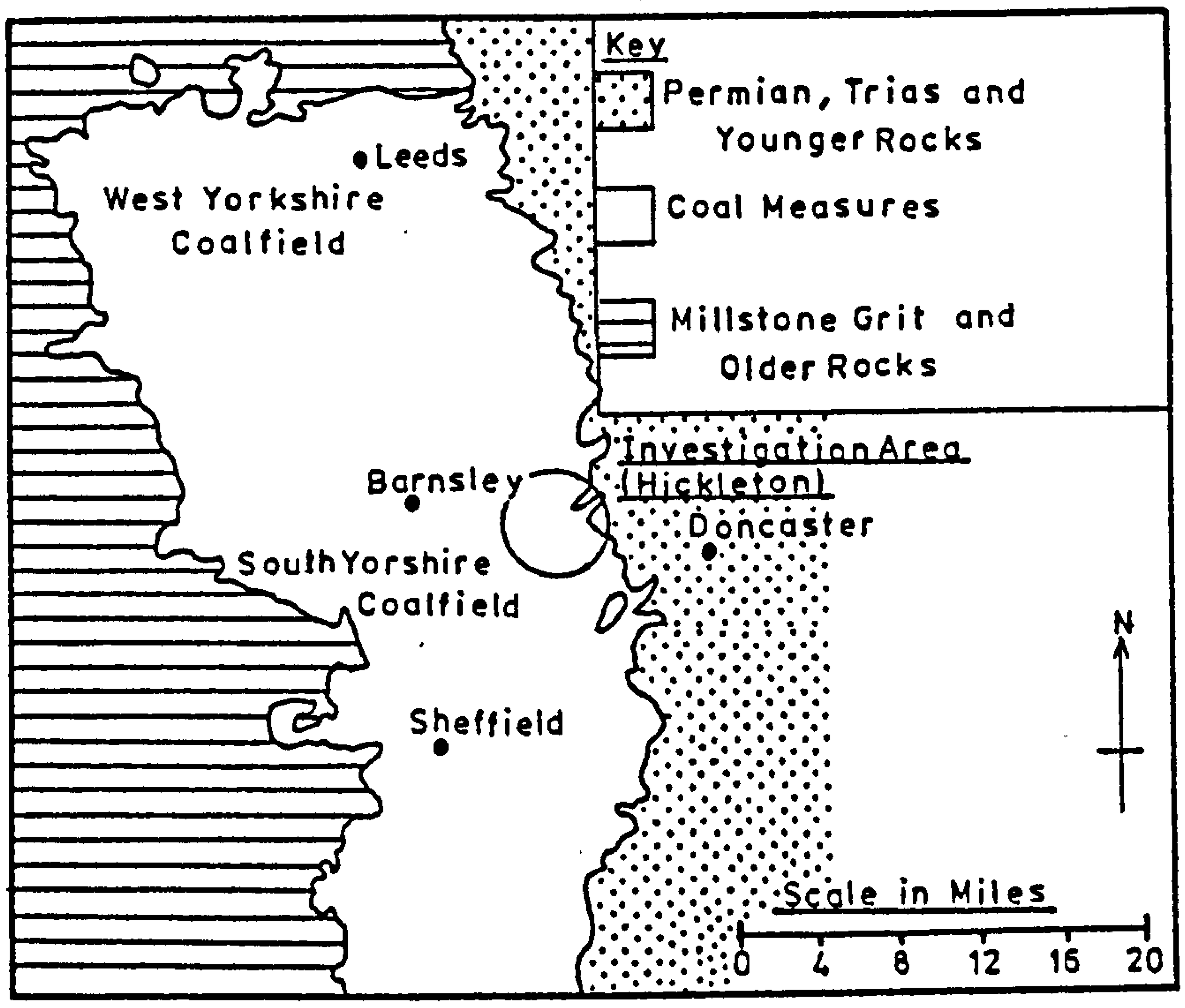


Figure 6.3 Location of the Investigation Area (Hickleton) in relation to the South Yorkshire Coalfield

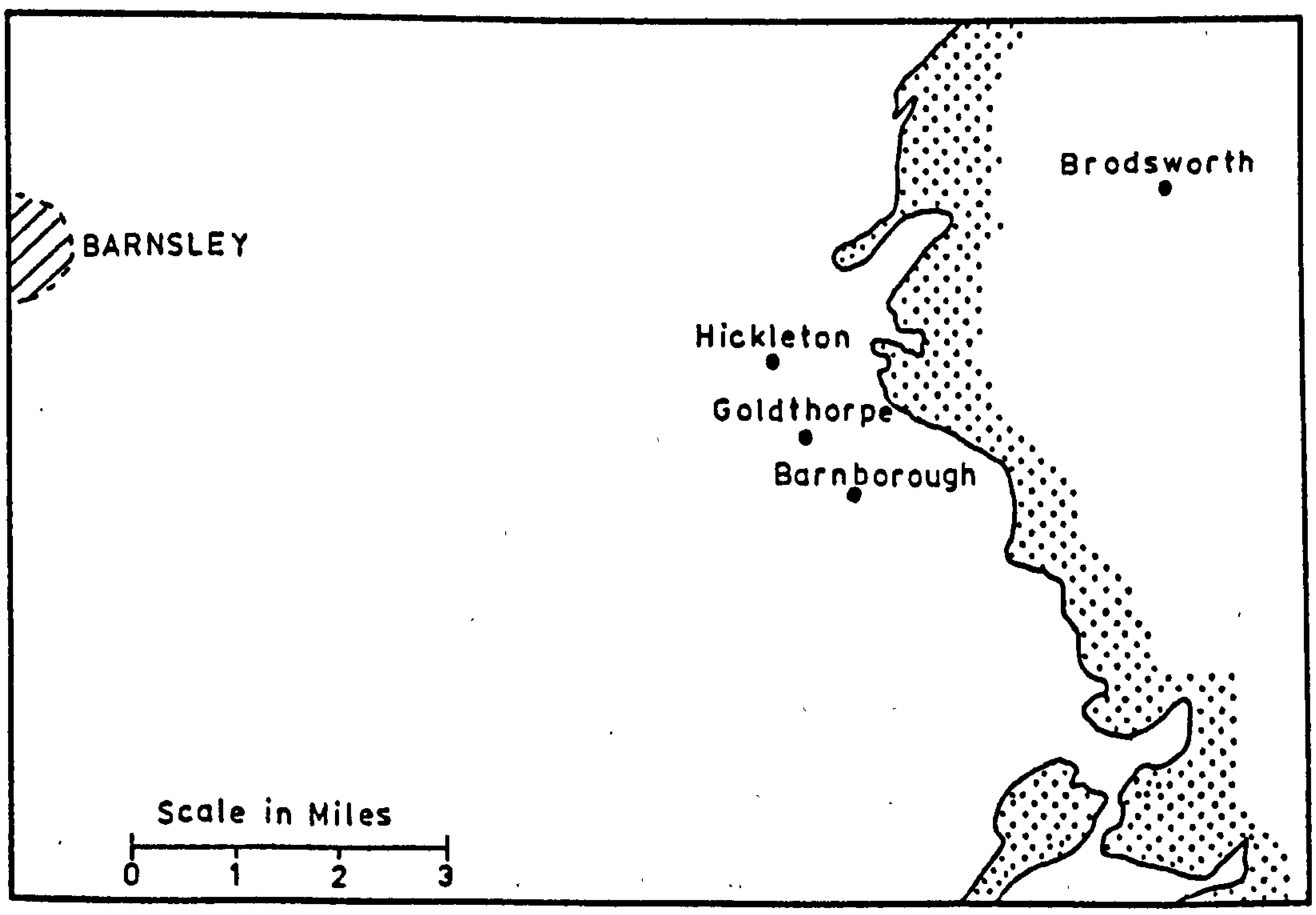


Figure 6.4 Location of Hickleton Colliery

parallel to a 90 m pillar which separates it from P31, which ceased production in 1974. The goaf width of both P29 and P31 is 183 m. Between the 321 and P29 goaf areas lies a 123 m pillar (the proposed P28 panel).

6.4.2 Test Site Geology

Hickleton Colliery lies on the eastern edge of the exposed South Yorkshire Coalfield, Figure 6.4, with the Parkgate seam in the eastern workings lying at a depth of some 900 m, Mitchell et al (70).

Figure 6.5 shows geological information available at the start of the investigation in the vicinity of the P30 and P28 panels. To the west of 321 lies a major washout, which separates the eastern and western areas of the colliery workings. The seam dips gently to the north.

The seam is initially overlain by a shaley sandstone containing coal streaks, which varies in thickness from 0.3 to 2.5 m. This is followed by a further sandstone sequence (the Parkgate Rock) between 15 and 25 m thick which grades upwards into muddy siltstone and a seatearth associated with the Fenton seam. Above the Fenton seam, lies a series of mudstones and siltstones which grade upwards into further interbedded sandstones, which underlie the Flockton Thick seam, Figure 6.5. This generalised strata sequence above P30 has been interpreted by the author from Figure 6.5. A detailed geological section of strata above P28 is given in Chapter 7 and a detailed log of the Parkgate Rock in Figure 6.6.

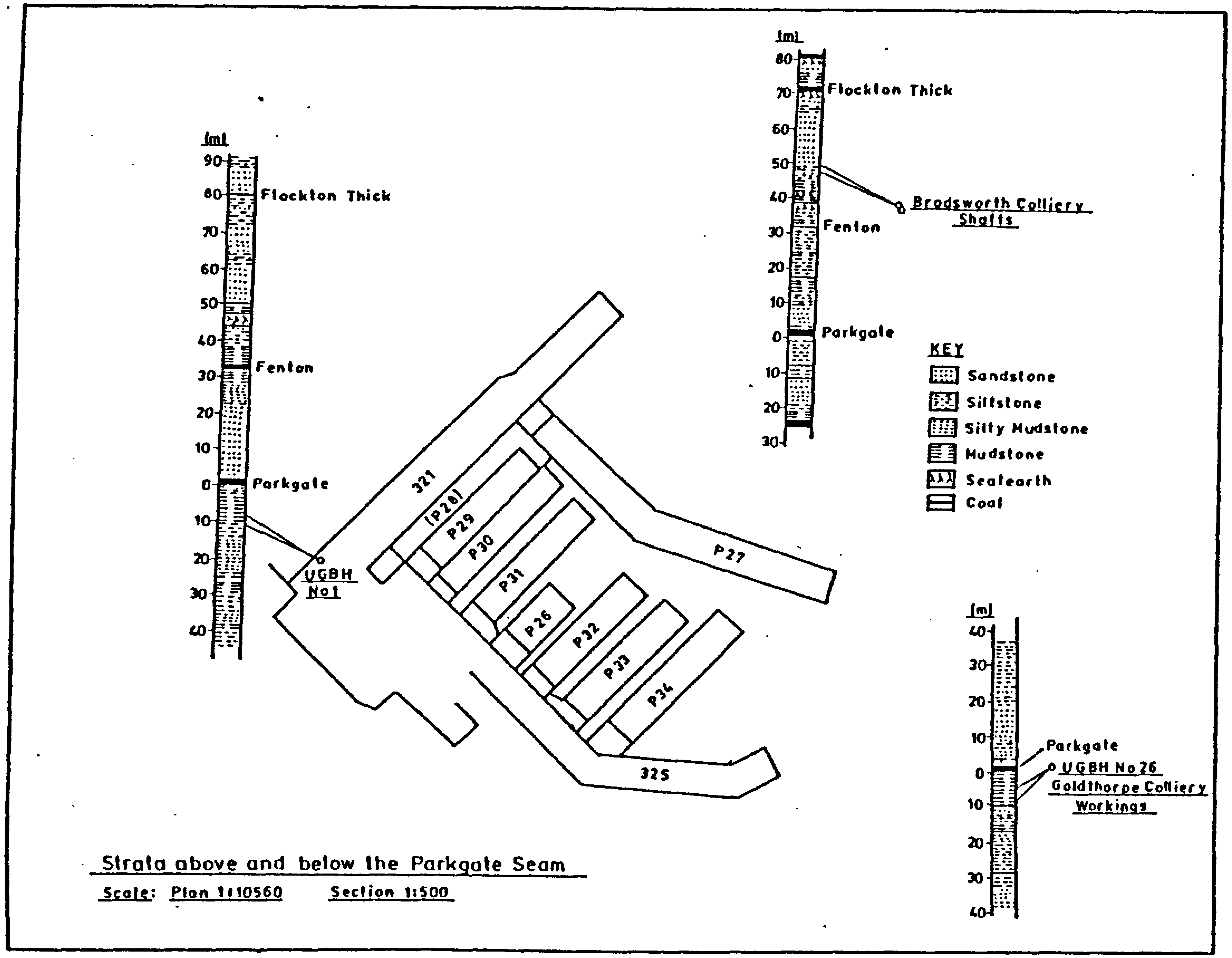


Figure 6.5 Geological Information available at the start of the Investigation, East Parkgate Area.

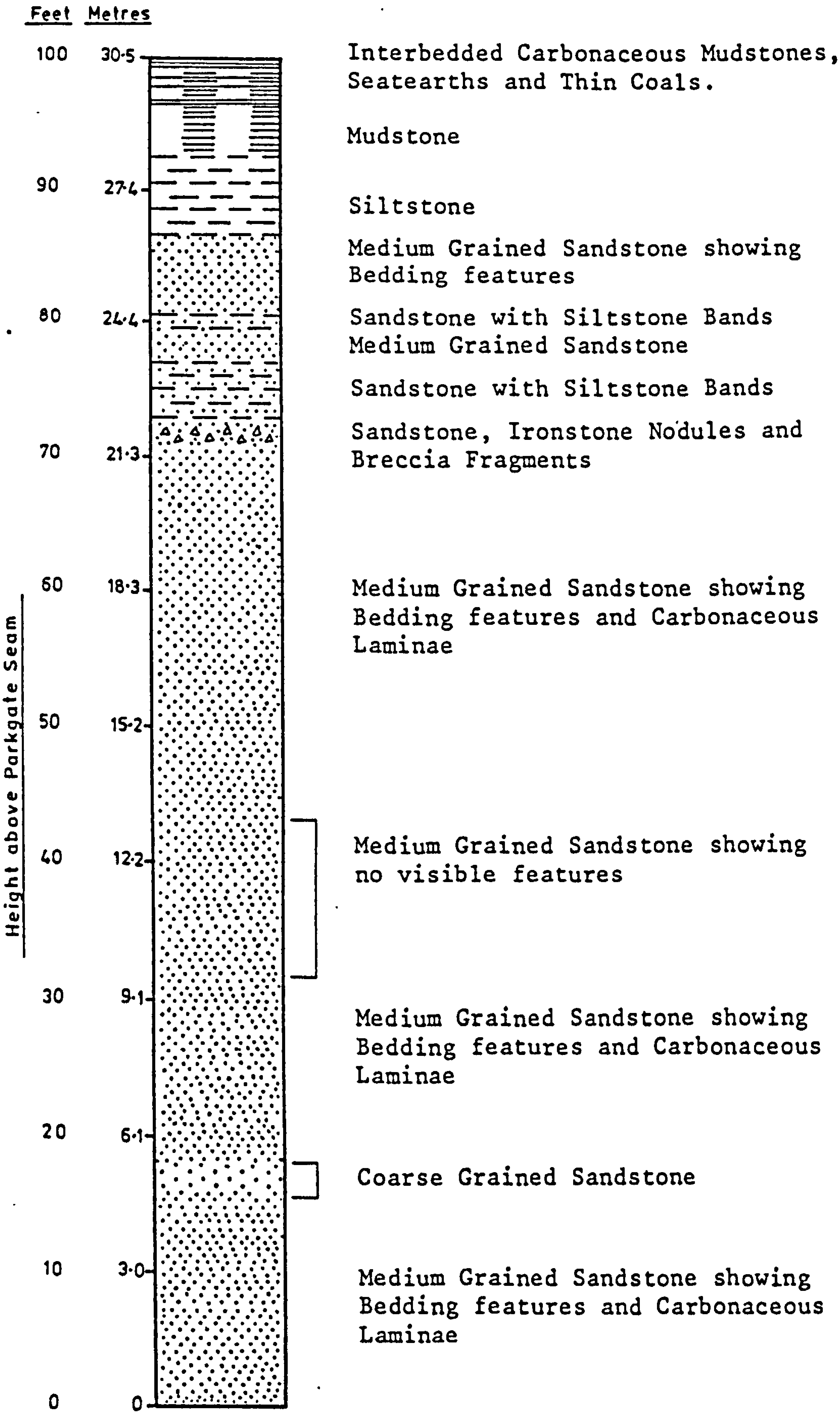


Figure 6.6 Geological Section of the Parkgate Rock, East Parkgate Area

(U.G.B.H. No. 24, Hickleton Colliery)

6.5 P30's Tailgate Extensometers

Extensometers were installed in P30's tailgate to monitor subsidence experienced by the Parkgate Rock during extraction. It was hoped to determine whether the overlying strata had been affected by extraction of the P29 panel and broken along the ribside area or was still bridging across P29's goaf.

6.5.1 Instrumentation

The P30 tailgate instrumentation consisted of two multi-wire, variable tension extensometers, sited directly over one another. The uphole was used to determine absolute movement and bed separation in the overlying strata, while the downhole acted primarily as a stable reference datum from which to calculate absolute movement in the uphole. This scheme was adopted after discussions with the survey staff revealed the impracticability of levelling to the site, each time a set of readings were required. This would be especially true once the face had passed and closure started to occur in the roadway.

Constant tension extensometers were considered impractical because of the site's central position in the roadway. Variable tension extensometers were therefore chosen and tensioned by means of a 6.8 kg weight before reading. Hedley (71) discusses the advantages and disadvantages of both types of extensometer, along with various corrections that can be applied.

6.5.1.1 The Uphole Extensometer

The installed extensometers were based on an original design by Whittaker and Hodgkinson (72), and consist of anchor points separated by a known distance onto which are attached fine multi-cored steel wires.

The anchor consists of a length of steel pipe, bevelled at each end, onto both sides of which is welded a length of corrugated steel. The corrugated sides provide extra grip against which grout can subsequently bond. A small hole is drilled at each end of the anchor and a steel wire attached to the lower end by means of a crimp, Figure 6.7. A piece of yellow tubing of predetermined length and 15 mm internal diameter (supplied by Celtite-Selfix to the NCB) is then threaded along the wire and attached to the anchor by means of one or two jubilee clips. The clips are subsequently bound with electrical insulating tape, in order to smooth protruding corners and help prevent snagging during installation. The next anchor is then threaded and attached in a similar manner. Further wires, tubing and anchors are then added until a desired length is reached. However, two points should be emphasised:

1) The top anchor has its uppermost end sealed in order to prevent grout from entering the extensometer.

2) Anchor spacing in a hole can either be purely arbitrary or based on some criteria such as the geological sequence.

A total of 20 anchors, at between 3 and 4 m spacings and 21 wires were used in the uphole extensometer, which was installed to a total height of 68 m above the Parkgate seam, Figure 6.10.

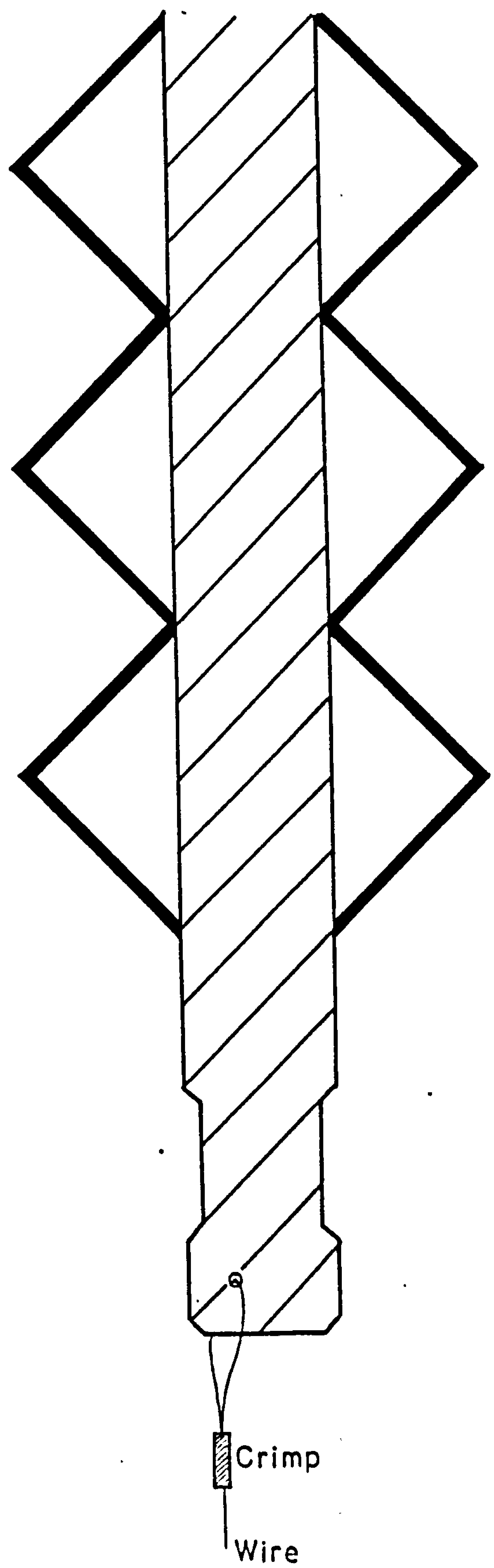


Figure 6.7 Extensometer Anchor showing attachment of Wire

6.5.1.2 The Downhole Extensometer

This was constructed in exactly the same manner as the uphole extensometer, except that the total length was only 15 m. Anchor separation distances of 3 m were used, resulting in a total of 5 anchors and 6 wires (2 wires were attached to the bottom most anchor).

6.5.2 Installation

Both the up and downhole extensometers were installed at the beginning of March 1980. Boreholes were drilled immediately prior to installation, with the face located 45 m behind the site.

6.5.2.1 Uphole Extensometer Installation

Once on site, the extensometer was unrolled along the roadway immediately prior to installation. A steel rod, previously welded to the topmost anchor, acted as an extension onto which could be slid a screw insertion rod. The extensometer was then pushed up the hole by simply adding further insertion rods. Exactly the same technique was used to install permeability monitoring equipment, Section 5.4.3 and Figure 5.9.

At 68 m, severe problems were encountered in supporting the extensometer and insertion rods, and it was decided to abandon the final 2 m of insertion. A rubber diaphragm made from a modified bicycle inner tube, attached to the topmost anchor and connected to the base of the hole by means of a narrow gauge nylon tube, was then inflated. This held the extensometer in place while the insertion rods were removed and a base plate fitted to the borehole standpipe. A grout pump was then attached and grout pumped up the hole until it emerged from a second narrow gauge nylon tube which also ran to the topmost anchor. This breather tube was perforated throughout its 1 m top section, so as to prevent blockage during insertion. When a steady stream of grout was seen emerging from the breather tube, it was sealed and the grout pump switched off.

Rubber washers were used to prevent leakage from between the extensometer tubes and baseplate. Similarly, although some leakage occurred from the surrounding roof strata, this was sealed using 'hard stop' cement.

6.5.2.2 Downhole Extensometer Installation

The downhole was filled with water remaining from drilling operations, a 6.8 kg weight was therefore attached to the bottom anchor and used to counter buoyancy effects on the extensometer as it was lowered down the hole. Grout was then pumped into the hole until full, when a baseplate was attached.

In both boreholes, a 2 m standpipe was inserted during the initial stages of drilling, for two main reasons:

1) To allow attachment of a baseplate and stop grout escaping from the base of the hole.

2) To protect strata surrounding the base of the hole from 'washing out' effects caused by the drill's flushing medium and also to provide a stable base from which to conduct drilling operations.

Grout was allowed to set for 24 hours before the initial set of readings were taken. In both holes, all the wires were tagged near their end with a copper crimp, on which existed an appropriate number of 'marks' to identify which anchor horizon it belonged to. However, particularly in the uphole, failure to achieve the desired horizon meant that all the wires had to be shortened and retagged, prior to the initial set of readings being taken. It was therefore essential to conduct this operation in a rigorous and systematic manner, to avoid mis-marking the wires and hence create spurious results.

6.5.3 Data Collection

The site was monitored on a regular basis by members of the colliery survey staff, although a member of the University was also usually present. It was soon found that a team of 4 were required to take readings in an accurate, rapid and organised manner.

In each case, it was attempted to keep the same team member booking and taking readings, in order to ensure continuity. Booking was undertaken by the University member, to compare current and previous readings and identify spurious results. The same tape was used for each set of readings and after discussion with the Survey Department, it was concluded that an accuracy of ±1 mm could be confidently expected.

Readings were started at the beginning of March with the face 45 m behind the site and finished in mid-May with the face 55 m infront of the site. Monitoring ceased when tailgate conditions finally became too dangerous to allow further access to the site. A total of 20 sets of readings were obtained for a face advance of 100 m.

In the uphole, readings were taken, after tensioning, between a 'marker' crimp (not the identifying crimp) and an arbitrary point on the baseplate adjacent to the wires. Location of the arbitrary point remained constant throughout the exercise, because it was initially marked with chalk and later with paint.

Originally, tensioning in the downhole was to be achieved using a spring balance. However, roadway conditions made this impractical, so a simple pulley system was devised. It consisted of two pieces of fine steel wire (extensometer wire), the first piece being attached to the roof, forming a loop. The second wire had two smaller loops, formed using crimps, at each end and this was passed through the loop of the first wire, Figure 6.8.

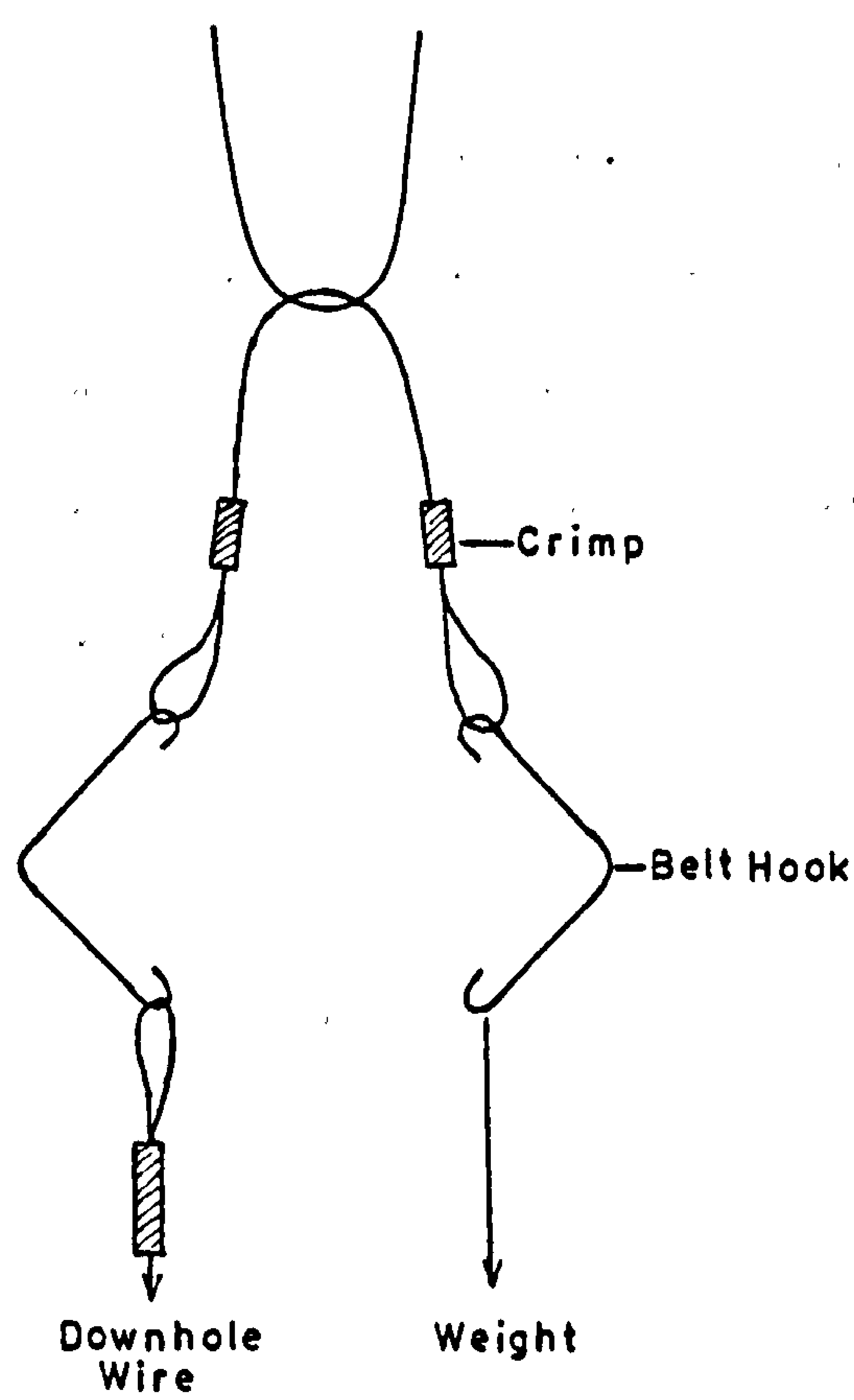


Figure 6.8 Pulley System for Tensioning Downhole Wires

A downhole wire and the weight were then attached, using 'belt hooks' and the system allowed to tension. Grease was periodically added to the point of contact between the two wires, to reduce friction. Readings were taken between an arbitrary point on the downhole baseplate and a 'marker' crimp (not the identifying crimp).

After completing a set of readings, the wires were carefully bound and tied 'out of the way' of roadway traffic. Downhole wires were fitted into a special steel lid, which bolted onto the baseplate and provided protection from passing traffic.

Readings were taken of vertical closure between the up and downhole baseplates, as well as roadway width between two arbitrary points on a nearby steel arch. Ideally, proper fixed measuring stations should have been installed in the roof and sides of the roadway, to measure convergence. However, difficulties experienced during drilling, involving the restricted passage of men and materials to the face and a general lack of time made this impractical. The standpipes must therefore be considered sufficiently stable to determine vertical closure, while the width measurements should be treated with caution.

6.5.4 Data Analysis

Data from P30's tailgate site can be divided into two categories:

1) Strata movement.

2) Roadway closure.

Strata movement can be sub-divided into: absolute movement, bed separation and vertical strain development. In addition, several.

roadway deformation surveys were also undertaken between the face line and site and these are discussed in Chapter 7. During the two and a half month investigation period, over 500 readings were collected and subsequently analysed.

6.5.4.1 Absolute Movement

The presence of a stable reference datum, in the form of an anchor 15 m below seam, allows absolute movement of the uphole anchors and hence strata, to be calculated, Whittaker and Hodgkinson (72).

Figure 6.9 illustrates this calculation, in which the units must be consistent.

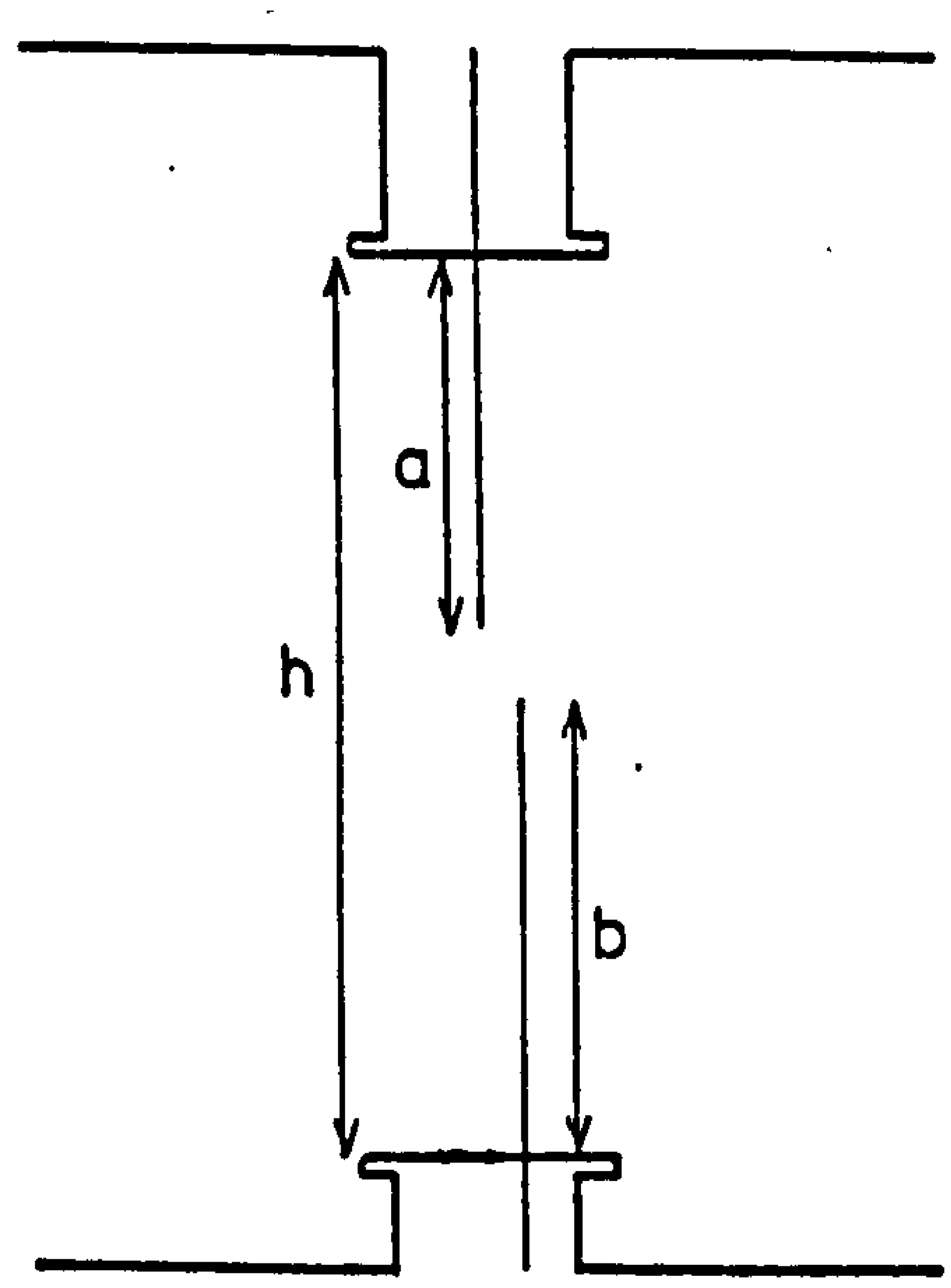


Figure 6.9 Calculation of Absolute Strata Movement

$$h_1 - (a_1 + b_1) = x_1$$
$$h_2 - (a_2 + b_2) = x_2$$

$$\text{therefore } x_1 - x_2 = y$$

where, a - Uphole wire reading

b - Downhole wire reading

h - Vertical distance between up and downhole base-plates.

x - Strata movement

y - Strata absolute movement.

6.5.4.2 Bed Separation and Vertical Strain Development

The amount of bed separation occurring between anchor/strata horizons can be calculated using the method described by Whittaker and Hodgkinson (72).

Vertical strain development between adjacent anchor horizons can be calculated by dividing the change in distance between adjacent anchors by the original anchor separation distance. Vertical strain development is therefore a function of bed separation.

6.5.5 Data Interpretation - P30's Tailgate

6.5.5.1 Absolute Movement

Absolute movement of the uphole anchor horizons, with reference to the basal downhole anchor, is shown in Figure 6.10. These values are tabulated in Table 6.1. Excellent agreement exists between the two wires attached to the topmost anchor, 68 m above seam, and emphasises the degree of confidence which can be placed in the recorded values.

During a total face advance of 100 m, strata overlying the site has undergone an increasing amount of settlement, with increased height above seam. At the top of the hole 0.56 m is recorded, compared with 0.30 m at the base, Figure 6.10. However, although differential settlement exists, respective movement has occurred almost simultaneously at all strata horizons, during face advance. Strata disruption and settlement starts to occur appreciably 20 m in front of the face, in the lower borehole sections (0 - 15 m). In the upper sections (15 - 68 m) this distance extends to nearer 35 m in front of the face. This is especially noticeable in the topmost borehole sections (59 - 68 m). An 'angle of draw' therefore appears to be developing in advance of the panel. At all horizons, the greatest rate of settlement is recorded between 10 m in front and 20 m behind the face, after which it slows appreciably.

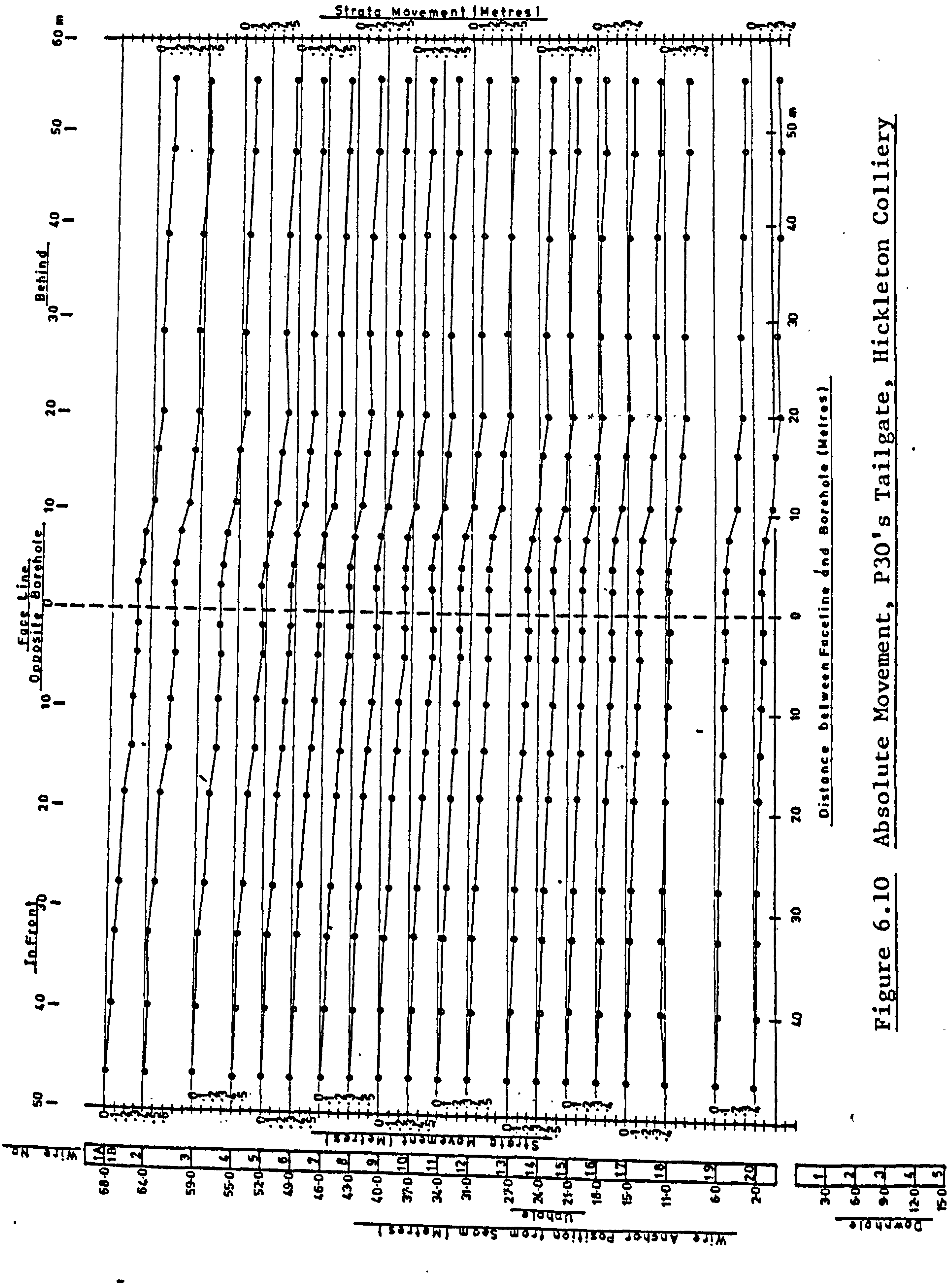


Figure 6.10 Absolute Movement, P30's Tailgate, Hickleton Colliery

Date	11/3	14/3	18/3	21/3	25/3	27/3	31/3	2/4	4/4	9/4	11/4	15/4	17/4	22/4	25/4	1/5	9/5	19/5	1980
Face position	+39	+32	+27	+18	+14	+9	+4	+1	-2	-4	-7	-10	-16	-20	-28	-38	-47	-55	(metres)
Anchor Horizon Number																			
1A	*0.056	0.075	0.127	0.165	0.226	0.246	0.282	0.287	0.281	0.332	0.351	0.441	0.467	0.511	0.500	0.524	0.565	0.561	
1B	0.042	0.060	0.111	0.151	0.224	0.242	0.307	0.302	0.292	0.311	0.349	0.430	0.465	0.515	0.501	0.524	0.568	0.558	
2	0.023	0.036	0.089	0.134	0.204	0.227	0.273	0.273	0.267	0.283	0.320	0.406	0.442	0.481	0.468	0.495	0.537	0.532	
3	0.020	0.038	0.101	0.140	0.200	0.218	0.248	0.245	0.241	0.262	0.299	0.385	0.411	0.471	0.456	0.476	0.509	0.513	
4	0.029	0.035	0.098	0.127	0.186	0.206	0.282	0.265	0.258	0.296	0.334	0.409	0.447	0.500	0.483	0.504	0.538	0.535	
5	0.032	0.047	0.101	0.135	0.189	0.212	0.254	0.266	0.258	0.280	0.320	0.400	0.443	0.482	0.473	0.494	0.535	0.534	
6	0.030	0.048	0.093	0.130	0.190	0.213	0.256	0.250	0.248	0.260	0.297	0.390	0.422	0.467	0.458	0.480	0.524	0.521	
7	0.040	0.057	0.096	0.142	0.183	0.205	0.264	0.265	0.252	0.265	0.301	0.382	0.424	0.465	0.455	0.473	0.520	0.516	
8	0.029	0.053	0.090	0.125	0.175	0.195	0.243	0.239	0.235	0.233	0.272	0.355	0.415	0.461	0.444	0.463	0.502	0.495	
9	0.018	0.044	0.076	0.118	0.167	0.186	0.230	0.224	0.215	0.223	0.255	0.338	0.370	0.427	0.405	0.430	0.470	0.471	
10	0.031	0.047	0.084	0.118	0.163	0.185	0.224	0.210	0.200	0.198	0.233	0.325	0.353	0.403	0.389	0.411	0.454	0.450	
11	0.028	0.050	0.078	0.107	0.161	0.175	0.210	0.195	0.197	0.200	0.236	0.330	0.360	0.408	0.396	0.418	0.456	0.458	
12	0.029	0.043	0.075	0.105	0.160	0.171	0.196	0.188	0.192	0.189	0.222	0.309	0.332	0.387	0.375	0.397	0.440	0.432	
13	0.037	0.056	0.082	0.114	0.151	0.166	0.193	0.189	0.180	0.186	0.218	0.296	0.330	0.384	0.367	0.392	0.433	0.426	
14	0.034	0.050	0.076	0.106	0.132	0.152	0.174	0.163	0.147	0.150	0.180	0.263	0.292	0.347	0.322	0.345	0.393	0.389	
15	0.032	0.047	0.073	0.091	0.124	0.135	0.153	0.142	0.146	0.149	0.180	0.254	0.287	0.343	0.329	0.351	0.396	0.387	
16	0.029	0.051	0.057	0.080	0.113	0.126	0.142	0.132	0.138	0.142	0.168	0.246	0.281	0.328	0.318	0.342	0.380	0.377	
17	0.024	0.043	0.053	0.066	0.098	0.109	0.124	0.124	0.124	0.121	0.148	0.233	0.265	0.310	0.295	0.320	0.356	0.356	
18	-0.045	-0.036	-0.033	-0.019	-0.003	0.008	0.018	0.022	0.022	0.019	0.052	0.125	0.160	0.203	0.197	0.220	0.261	0.254	
19	0.018	0.029	0.035	0.051	0.068	0.082	0.097	0.099	0.100	0.098	0.130	0.207	0.234	0.281	0.272	0.302	0.333	0.330	
20	0.010	0.019	0.021	0.032	0.050	0.061	0.074	0.075	0.075	0.073	0.109	0.178	0.213	0.255	0.251	0.271	0.311	0.304	

* All absolute movement values in metres

Table 6.1 Absolute Movement Values, P30's Tailgate, Hickleton Colliery

Differential strata movement between the vertical horizons can be explained by three interrelated factors.

1) P30's tailgate is sited immediately adjacent P29's goaf, Figure 6.2. If fracturing and caving of strata occurred during the working of P29, bed separations would result along the P29/P30 ribside. Therefore, when strata relaxation occurred along the ribside, cumulative closure of the old separations would also occur along with newer settlement associated with P30.

2) If it is assumed that no deflection of the uphole occurred during drilling, it should therefore be vertical. Working P30's panel will result in strata settlement into the goaf, with a resultant lateral movement or 'slumping' of the higher strata towards this area (normal subsidence profile formation theory). Top sections of the hole will be displaced further over the goaf area than the lower, since the amount of strata settlement increases away from the ribside into the extracted area. The top sections should therefore undergo greater recorded settlement than the lower. This argument would be supported further, if deflection of the hole towards the goaf occurred during drilling.

3) A 3 m packside coal pillar was left between the face end and tailgate, which necessitated the formation of 'bob' holes at regular intervals. At a depth of 860 m this pillar will eventually crush, although initially, it would provide additional support to strata surrounding the base of the hole. When final crushing did occur, access to the site would not be available to monitor the total amount of settlement attained.

6.5.5.2 Bed Separation

Table 6.2 lists relative amounts of bed separation occurring between adjacent anchor horizons. Although not graphically represented, the results show considerable fluctuation between separation and closure, not only between adjacent horizons but also with respect to face advance. No simple relationship appears to exist between the sign and magnitude of values and face advance.

However, the results do confirm the theory that strata above the P29/30 ribside was not intact, but had already been pre-fractured by the working of P29. Passage of P30's panel caused differential movement of these separations and this is shown by the results.

6.5.5.3 Vertical Strain Development

Vertical strain development between adjacent anchor horizons is tabulated in Table 6.3 and shown graphically in Figure 6.11.

The results show a predominance of compressive strain at most levels, although some tensile horizons do exist, anchors 3 - 4 and 18 - 19. Compressive strain indicates closure and tensile separation. It is therefore inferred that closure is dominant between most horizons. This can be explained by the combined effects of settlement associated with P30 and the closure of pre-existing separations from P29. Tensile horizons can be attributed to the further parting of existing separations and/or failure along planes of inherent strata weakness, associated with either geological features or micro-fracture networks induced by P29.

Anchor Horizon / Face position	+39	+32	+27	+18	+14	+9	+4	+1	-2	-4	-7	-10	-16	-20	-28	-38	-47	-55 (metres)
1A - 2	*-32.5	-6.0	+1.0	+7.0	+8.5	+2.5	+11.0	-5.0	-0.5	-24.5	+8.5	-4.0	+9.0	-5.0	-2.0	+2.5	+1.5	+1.5
1B - 2	-19.0	-4.5	+1.5	+5.0	-3.5	+5.0	-18.5	+5.0	+4.0	-3.0	-0.5	+5.0	0.0	-10.5	+1.5	+3.0	-1.5	+5.0
2 - 3	-4.0	+6.0	+9.5	-5.0	-10.5	-5.0	-16.0	-3.0	+2.0	+4.0	-0.5	0.0	-9.5	+20.5	-1.5	-6.0	-10.5	+9.5
3 - 4	+10.0	-13.0	+0.5	-10.5	-1.0	-2.5	+45.0	-13.5	-2.5	+16.0	+2.0	-11.0	+11.5	-11.5	+2.0	+1.5	+1.0	-7.0
4 - 5	+2.5	+9.0	-9.0	+5.5	-4.5	+2.0	-33.5	+29.0	-1.0	-16.5	+1.0	+5.0	+6.5	-9.5	+4.0	-1.0	+7.5	+2.0
5 - 6	-2.0	+3.0	-9.0	+3.0	+5.5	+0.5	+1.5	-19.0	+6.0	-9.5	-1.5	+11.5	-6.0	+1.0	-0.5	+1.0	+3.5	-2.0
6 - 7	+10.0	-0.5	-6.5	+8.5	-18.5	-1.5	+16.0	+8.0	-11.5	+1.0	-2.0	-10.0	+3.5	+0.5	-0.5	-3.5	+1.5	+0.5
7 - 8	-10.5	+6.5	-2.5	-10.5	+8.5	-1.0	-12.0	-5.0	+9.5	-15.0	+3.0	+2.0	+18.5	+5.0	-8.0	+1.0	-6.5	-4.5
8 - 9	-11.0	+2.0	-4.5	+6.5	-1.0	-1.5	-3.0	-2.5	-5.0	+10.0	-7.0	0.0	-28.0	+9.0	-4.5	+5.0	+3.5	+7.5
9 - 10	+13.0	-10.5	+6.0	+8.5	-4.0	+3.5	-5.5	-7.5	-2.0	-10.0	+3.5	+9.5	-4.0	-6.5	+7.5	-2.0	+2.0	-4.0
10 - 11	-3.0	+6.0	-9.0	-5.0	+9.0	-8.5	-4.5	-1.0	+13.5	+4.0	+1.5	+1.5	+0.5	-1.5	+2.5	0.0	-4.5	+5.5
11 - 12	+1.0	-8.0	+3.5	+2.0	+0.5	-3.0	-10.0	+7.0	+2.0	-6.0	-3.0	-7.5	-5.0	+6.0	-0.5	0.0	+4.0	-9.0
12 - 13	+8.0	+4.0	-5.5	+1.5	-17.5	+4.0	+2.0	+3.0	-13.5	+10.5	-1.0	-9.5	+11.0	-1.0	-4.5	+2.0	-1.0	0.0
13 - 14	-3.0	-2.5	-0.5	-2.5	-11.5	+5.0	-5.5	-6.5	-6.5	-4.5	-1.5	+5.5	-5.5	+1.0	-8.0	-1.0	+6.0	+3.0
14 - 15	-2.0	-1.0	-0.5	-11.0	+6.5	-9.5	-3.0	0.0	+20.0	+0.5	0.0	-8.5	+4.0	+1.0	+10.5	-1.0	-3.0	-4.5
15 - 16	-3.5	+7.0	-19.5	+4.5	-0.5	+2.5	-2.0	+0.5	+2.0	+1.0	-4.5	+3.5	+2.0	-9.0	+4.0	+2.0	-6.0	+5.5
16 - 17	-5.0	-2.5	+3.0	-9.5	-1.0	-2.0	-1.0	+10.5	-6.0	-7.0	+1.0	+7.0	-3.5	-2.5	-4.0	0.0	-1.5	+2.5
17 - 18	-9.5	-10.5	-6.5	+0.5	-14.5	-1.0	-5.0	+4.5	-0.5	0.0	+6.5	-12.5	+3.0	-1.0	+8.0	-1.5	+3.5	-6.5
18 - 19	+3.5	+2.5	+2.5	+2.5	0.0	+3.5	+4.5	-2.0	+1.0	+1.0	-2.0	+5.0	-7.5	+3.5	-2.0	+6.0	-9.0	+4.5
19 - 20	-8.0	-2.5	-3.5	-5.5	+1.5	-3.0	-1.5	-2.0	-0.5	0.0	+4.5	-8.5	+8.0	-4.5	+4.5	-7.5	+7.0	-5.0

* All separation values in millimetres (mm)   (-) = Closure (+) = Separation

Table 6.2 Bed Separation Values, P30's Tailgate, Hickleton Colliery

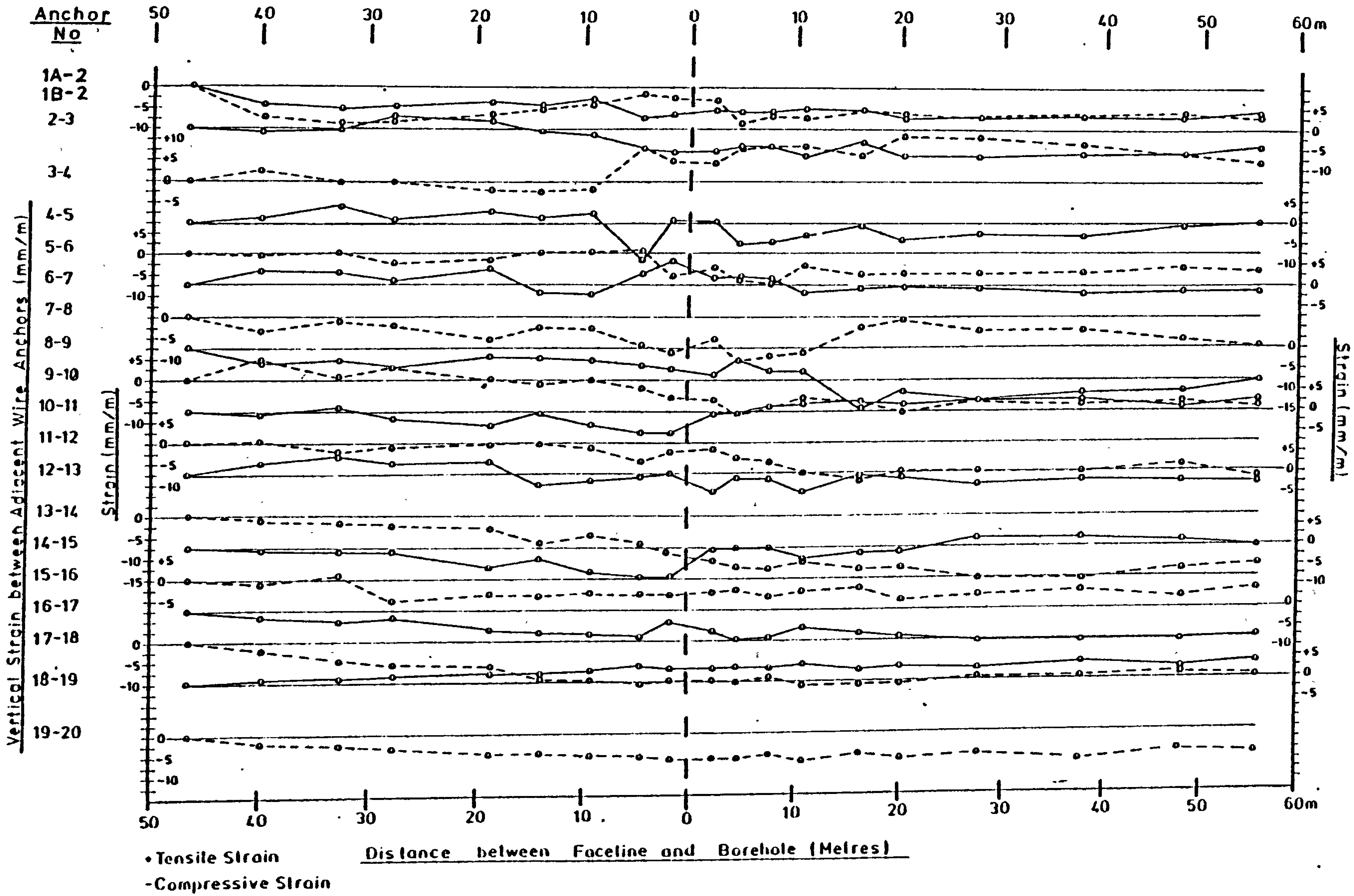


Figure 6.11 Vertical Strain, P30's Tailgate, Hickleton Colliery

Height Above Seam (metres)	Anchor Horizons / Face position	+39	+32	+27	+18	+14	+9	+4	+1	-2	-4	-7	-10	-16	-20	-28	-38	-47	-55 (metres)
68.4 - 64.0	1A - 2	*-7.6	-9.0	-8.8	-7.1	-5.1	-4.6	-2.0	-3.2	-3.3	-9.0	-7.0	-8.0	-5.8	-7.0	-7.5	-6.9	-6.5	-6.9
68.4 - 64.0	1B - 2	-4.4	-5.5	-5.1	-4.0	-4.8	-3.6	-8.0	-6.8	-5.8	-6.5	-6.7	-5.5	-5.5	-8.0	-7.6	-6.9	-7.3	-6.1
64.0 - 59.5	2 - 3	-0.9	-0.4	+2.5	+1.4	-0.9	-2.0	-5.5	-6.2	-5.7	-4.6	-4.7	-4.7	-6.8	-2.3	-2.6	-4.0	-6.3	-4.2
59.5 - 54.9	3 - 4	+2.2	-0.6	-0.5	-2.8	-3.1	-2.5	+7.3	+4.4	+3.8	+7.3	+7.8	+5.4	+7.9	+5.4	+5.8	+6.1	+6.3	+4.8
54.9 - 51.9	4 - 5	+0.8	+3.8	+0.8	+2.6	+1.1	+1.8	-9.2	+0.3	0.0	-5.4	-5.1	-3.4	-1.3	-4.4	-3.1	-3.4	-1.0	-0.3
51.9 - 48.8	5 - 6	-0.6	+0.3	-2.6	-1.6	+0.2	+0.3	+0.8	-5.4	-3.4	-6.5	-7.0	-3.3	-5.2	-4.9	-5.1	-4.8	-3.6	-4.3
48.8 - 45.8	6 - 7	+3.3	+3.1	+1.0	+3.8	-2.1	-2.6	+2.6	+5.3	+1.5	+1.8	+1.2	-2.1	-1.0	-0.8	-1.0	-2.1	-1.6	-1.5
45.8 - 42.7	7 - 8	-3.4	-1.3	-2.1	-5.6	-2.8	-3.1	-7.0	-8.7	-5.6	-10.5	-9.5	-8.8	-2.8	-1.0	-3.6	-3.3	-5.4	-6.9
42.7 - 39.7	8 - 9	-3.6	-2.9	-4.4	-2.3	-2.6	-3.1	-4.1	-4.9	-6.5	-3.3	-5.6	-5.6	-14.7	-11.1	-12.6	-11.0	-10.5	-8.0
39.7 - 36.6	9 - 10	+4.3	+0.8	+2.8	0.0	-1.3	-0.2	-2.0	-4.4	-5.1	-8.4	-7.2	-4.4	-5.7	-7.9	-5.4	-6.1	-5.4	-6.7
36.6 - 33.6	10 - 11	-1.0	+1.0	-2.0	-3.6	-0.6	-3.4	-4.9	-5.2	-0.8	+0.5	+1.0	+1.8	+2.0	+1.5	+2.3	+2.4	+0.8	+2.6
33.6 - 30.5	11 - 12	+0.3	-2.3	-1.1	-0.5	-0.3	-1.3	-4.6	-2.3	-1.6	-3.6	-4.6	-7.0	-8.7	-6.7	-6.9	-6.9	-5.6	-8.5
30.5 - 27.5	12 - 13	+2.6	+4.3	+2.5	+3.0	-2.8	-1.5	-0.8	+0.2	-4.3	-0.8	-1.1	-4.3	-0.6	-1.0	-2.5	-1.6	-2.0	-2.0
27.5 - 24.5	13 - 14	-1.0	-1.8	-2.0	-2.8	-6.2	-4.6	-6.4	-8.5	-10.6	-12.1	-12.6	-10.8	-12.6	-12.3	-14.9	-15.2	-13.3	-12.3
24.5 - 21.4	14 - 15	-0.7	-1.0	-1.1	-4.7	-2.6	-5.7	-6.7	-6.7	-0.2	0.0	0.0	-2.8	-1.5	-1.1	+2.3	+2.0	+1.0	-0.5
21.4 - 18.4	15 - 16	-1.1	+1.1	-5.1	-3.6	-3.8	-2.9	-3.6	-3.4	-2.8	-2.4	-3.9	-2.8	-2.1	-5.1	-3.8	-3.1	-5.1	-3.3
18.4 - 15.3	16 - 17	-1.6	-2.5	-1.5	-4.6	-4.9	-5.6	-5.9	-2.5	-4.4	-6.7	-6.4	-4.1	-5.3	-6.1	-7.4	-7.4	-7.9	-7.1
15.3 - 10.7	17 - 18	-2.1	-4.4	-5.8	-5.7	-8.8	-9.1	-10.1	-9.2	-9.3	-9.3	-7.9	-10.6	-9.9	-10.1	-8.4	-8.7	-8.0	-9.4
10.7 - 6.2	18 - 19	+0.8	+1.3	+1.9	+2.4	+2.4	+3.2	+4.2	+3.7	+3.9	+4.2	+3.7	+4.8	+3.2	+3.9	+3.5	+4.8	+2.8	+3.8
6.2 - 1.6	19 - 20	-1.7	-2.3	-3.1	-4.3	-3.9	-4.6	-4.9	-5.4	-5.5	-5.5	-4.5	-6.3	-4.6	-5.6	-4.6	-6.7	-4.7	-5.8

* All strain values in mm/m   - = Compression   + = Tension

Table 6.3 Vertical Strain Values, P30's Tailgate, Hickleton Colliery

Significant fluctuations in strain magnitude and sign, both along the borehole and with face advance, has made it impossible to draw meaningful vertical strain development distributions for strata surrounding the site. Also, since vertical strain development is a function of bed separation, graphical representation of the bed separation results, would produce a series of fluctuating lines similar to those seen in Figure 6.11.

6.5.5.4 Roadway Closure

Vertical closure between the up and downhole baseplates is shown in Figure 6.12. Until the face is about 30 m in front of the site, closure remains minimal and the roadway undisturbed. However, between 30 and 20 m in front of the face, the rate of closure begins to increase slowly and this increases rapidly after 20 m. The greatest amount of closure occurs once the face has passed the site. A closure of 0.26 m with the face parallel and a total of 0.64 m with the face 55 m in front, was recorded at the site.

These values only indicate the amount of strata closure experienced in the roadway and do not take into account floor heave. Visual inspection during the investigation revealed that once the face was 55 m passed site, about 1.0 m of floor heave had occurred, and that most of this was experienced after the site had been passed. However, some floor heave was also experienced in front of the face line, with the effects becoming noticeable 10 to 15 m in advance.

Figure 6.13 shows roadway width measurements, taken on a nearby arch. Stable conditions exist until the face is 30 m from site, after which closure begins to occur. Once passed 20 m, the

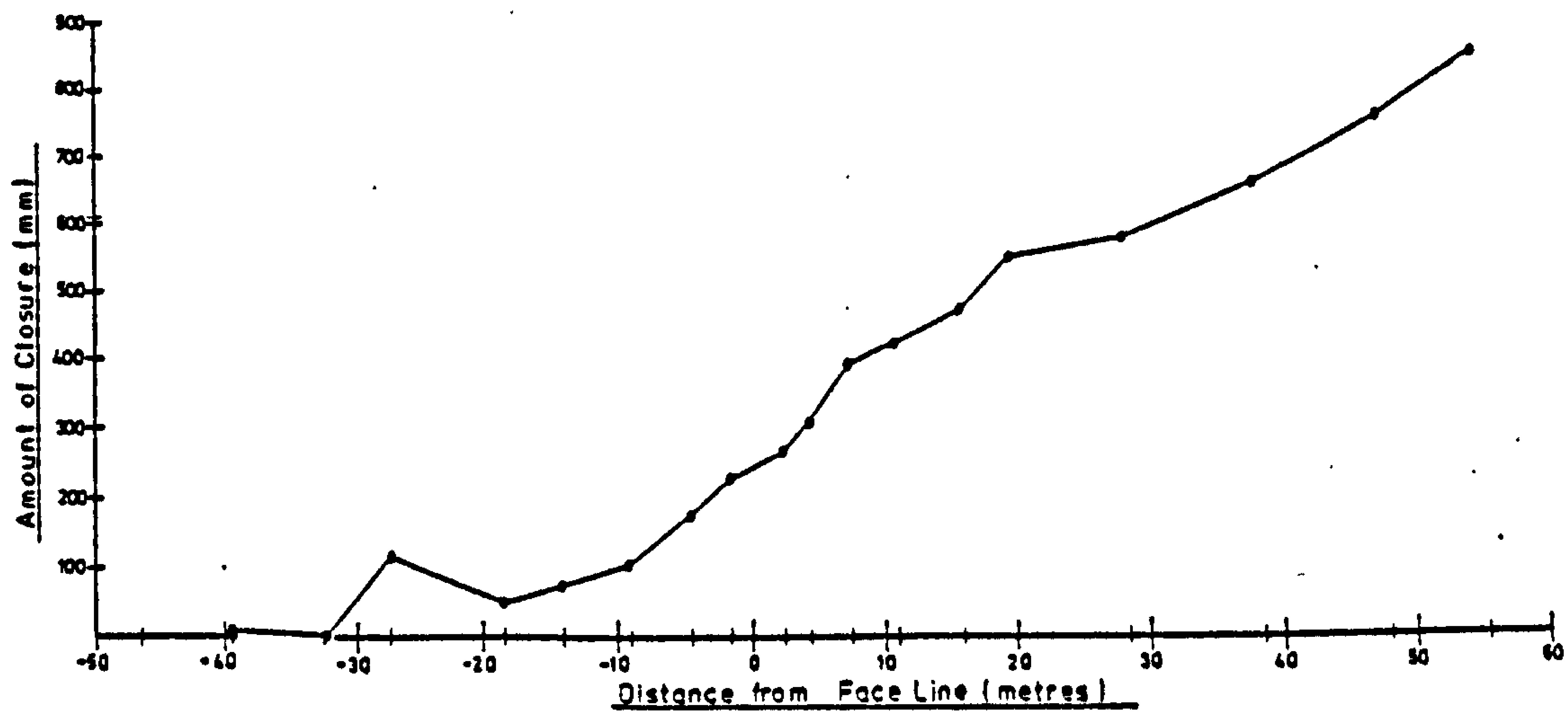


Figure 6.13 Horizontal Roadway Closure, P30's Tailgate

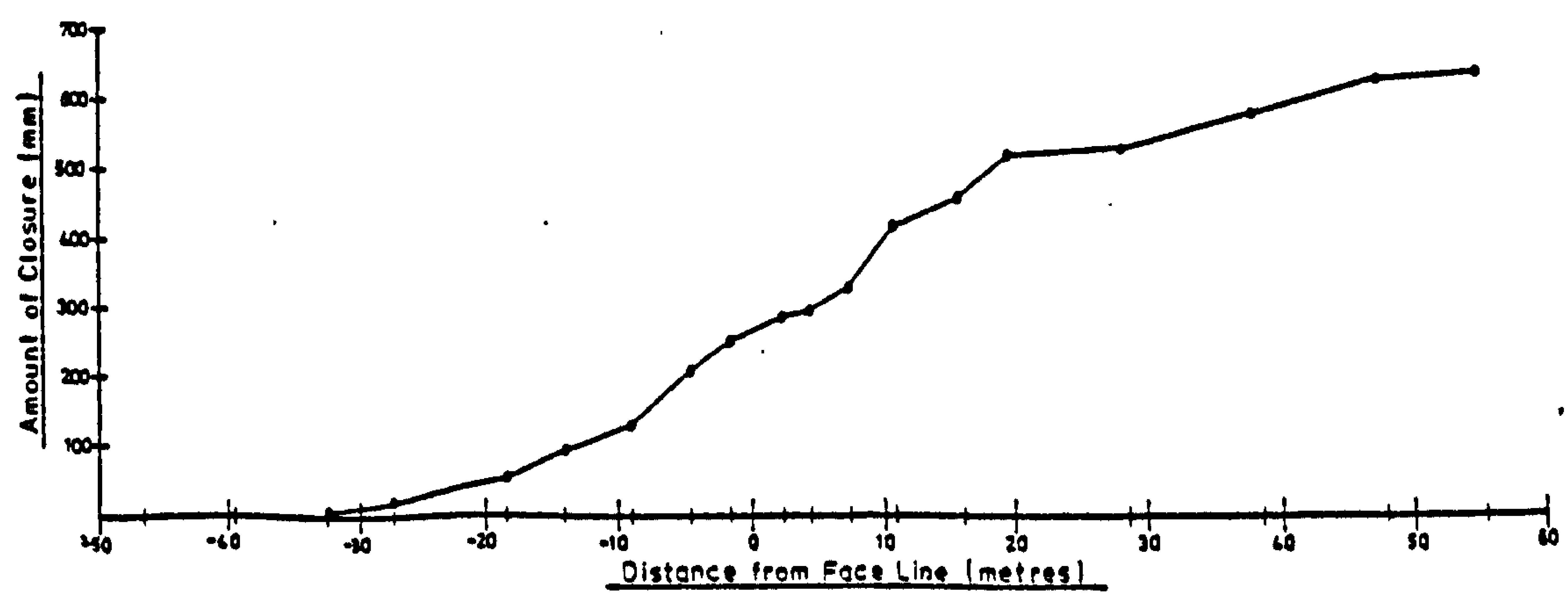


Figure 6.12 Vertical Roadway Closure, P30's Tailgate

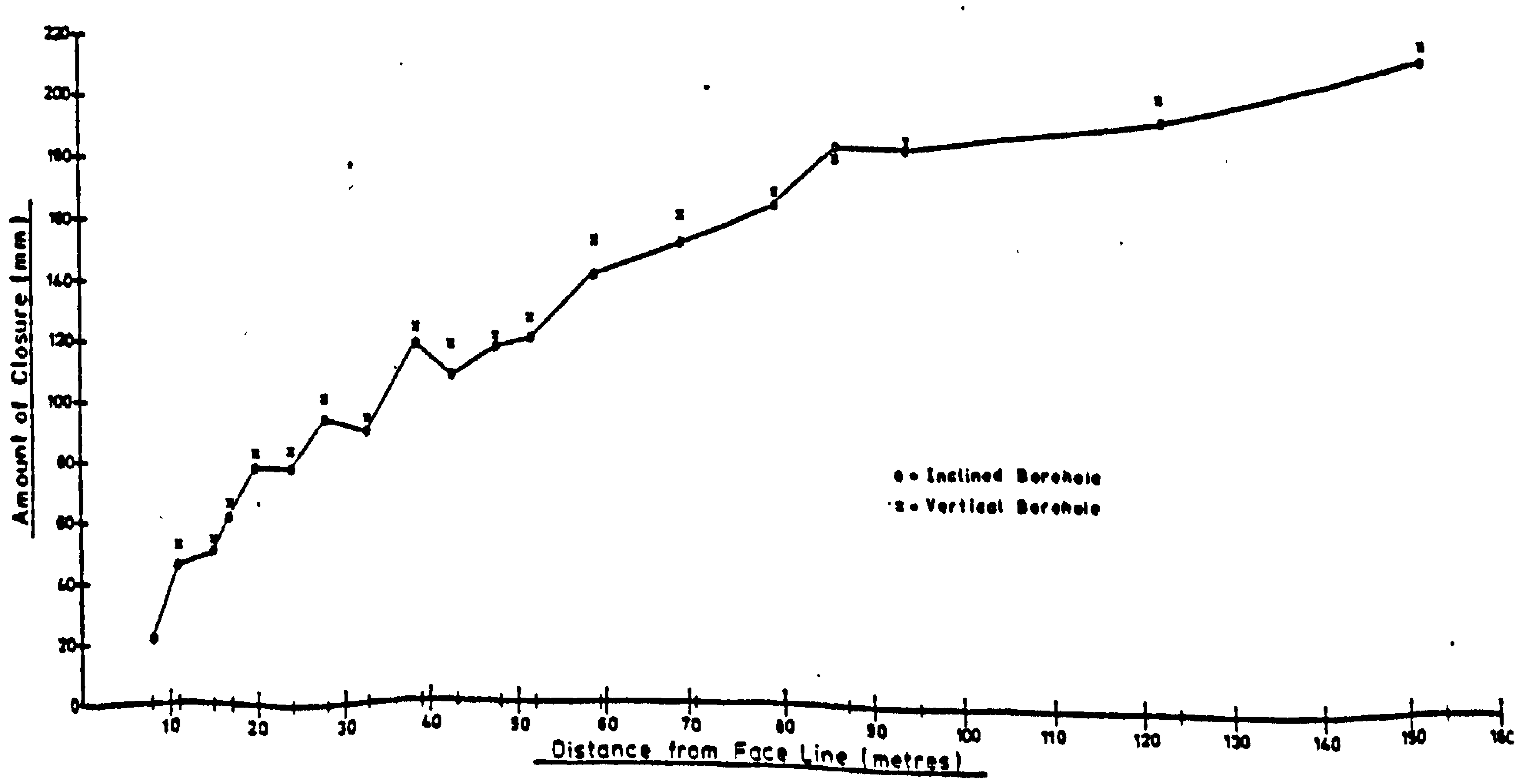


Figure 6.19 Vertical Roadway Closure, P30's Maingate

rate of closure begins to increase significantly, but as with vertical closure, the greatest amount again occurs once the face has passed the site. This suggests that once passed, strata settlement results in an increased vertical and lateral loading of the arches, with resultant bending and buckling. However, these results should be treated with caution, because of the non-stable nature of the measuring station.

6.6 P30's Maingate Extensometers

At a meeting on 5th September 1980 (73), it was concluded that results from P30's tailgate site were insufficient to alleviate fears that the Parkgate Rock might be cantilevered over the ribside after extraction of the panel. It was therefore decided to install a further three extensometers in P30's maingate, sited immediately behind the face line, in order to monitor strata behaviour as the face advanced away from the site.

6.6.1 Instrumentation

P30's maingate instrumentation consisted of three multi-wire, variable tension extensometers. Two upholes extending to a vertical height of 30 m, which corresponds to the projected thickness of Parkgate Rock over the panel. One uphole would be vertical and the other inclined at 75^{o} to the horizontal. Using this scheme, if cantilevering was occurring over the ribside, greater movement should be registered in the inclined rather than the vertical hole. However, it could also be argued, that since the inclined hole is sited over the extracted area, it should undergo greater movement anyway due to strata settlement into the goaf. Alternatively, if movement in both holes is small and identical, then the strata may well be bridging across the panel.

A 15 m downhole extensometer was also installed to provide a stable reference datum from which to calculate absolute strata movement in the upholes. In each extensometer, anchors were installed at horizons which corresponded with those in the tailgate, to allow a direct comparison of results. A total of 9 anchors and 10 wires (two were attached to the top anchor) were installed in each of the upholes and 5 anchors, 6 wires in the downhole, Figures 6.15 and 6.16.

The design and construction of each extensometer was identical to those used in the tailgate and already described in Section 6.5.1.

6.6.2 Installation

Drilling and instrumentation of the boreholes occurred during a holiday period, thus allowing unrestricted use of and access to, the face end region. Installation of all three holes was undertaken using methods already employed at the tailgate site and described in Section 6.5.2.

The two uphole extensometers were installed adjacent to one another, Plate 7 and the downhole instead of being located directly beneath, was moved 10 m outbye. This was necessitated by their initial close proximity to the gate end. If the downhole had been

Plate 7

P30's Maingate Uphole
Extensometers

Plate 8

Data Collection
P30's Maingate Downhole
Extensometer

sited beneath the upholes, observations might have interfered with drivage operations and the instrumentation irrevocably damaged by the 'mucking out' bucket. A new system of measuring vertical distances between the up and downhole baseplates was therefore employed.

6.6.3 Data Collection

A four man team was again found to be necessary for collecting a set of readings. Uphole wires were tensioned using a 6.8 kg weight, while for downhole wires a wire pulley system, Plate 8, was again used, Section 6.5.3.

After discussion with the Colliery Surveyor, it was decided to use a survey level, taking front and back sights on a tape measure at each site, to measure vertical distances between the up and downhole baseplates.

6.6.3.1 Measurement of Vertical Closure

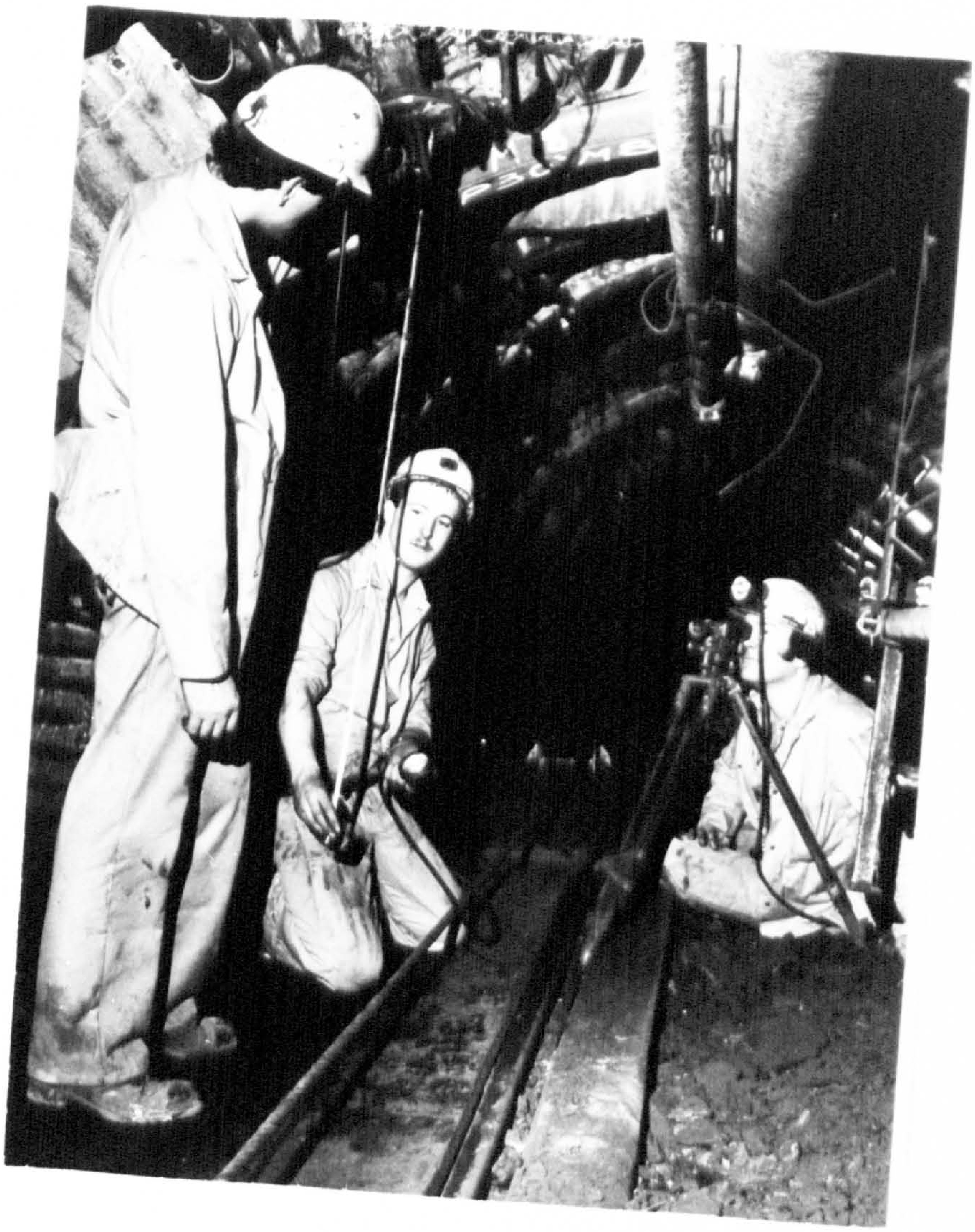
Figure 6.14 shows a schematic diagram of the proposed measuring system. A survey level is set up mid-way between the extensometer sites and levelled into position. A tape is then held to the reference point on one of the uphole baseplates, a 'cap lamp' shone along it and a reading taken, Plate 9. The same procedure is then repeated for the downhole, Plate 10 and the two readings added. A second set of readings are then taken and compared with the first. If both results agree and compare favourably with previous values, it is booked, otherwise the whole operation is repeated. The same procedure is then repeated for the second uphole.

Plate 9

Vertical Closure Measurement
- Uphole, P30's Maingate

Plate 10

Vertical Closure Measurement
- Downhole, P30's Maingate



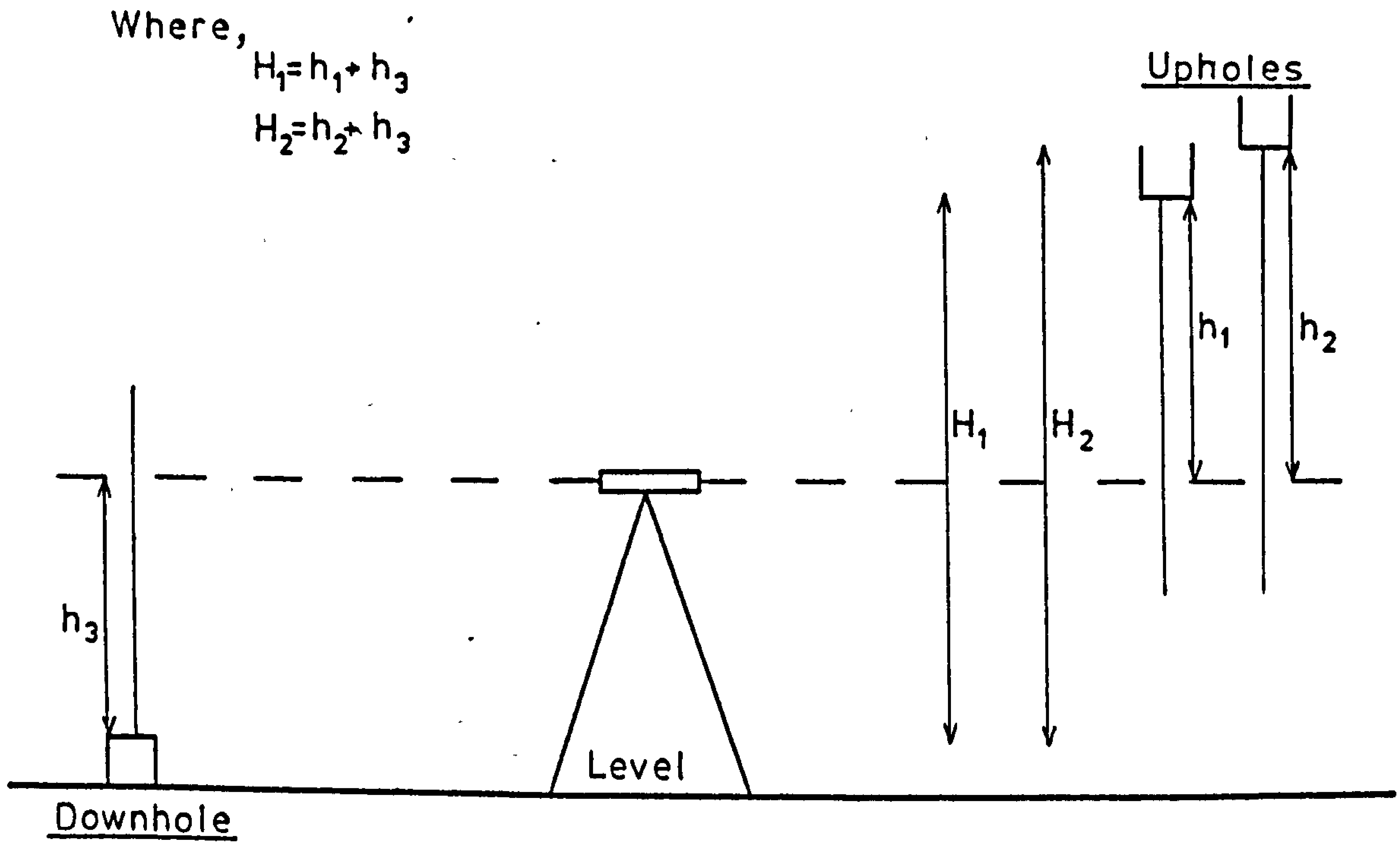


Figure 6.14 System for Measuring Vertical Distance between the Up and Downhole Baseplates, P30's Maingate

For each set of results, the level was set up and operated by the same team member, in order to ensure continuity. It is concluded by both survey staff and team members that the results obtained were accurate to ±1 mm.

6.6.4 Data Analysis

Data from P30's maingate site can again be divided into two categories:

1) Strata movement.

2) Roadway closure.

Strata movement can again be sub-divided into absolute movement, bed separation and vertical strain development. Calculation of all three parameters was undertaken using methods employed for the tailgate results and described in Section 6.5.4.

During the four month investigation period, over 400 readings were collected and subsequently analysed. However, all values analysed for the inclined extensometer were corrected by the sine of 75^{o} (0.966), to give a calculated vertical component rather than the measured inclined component. Finally, no roadway deformation surveys were undertaken between the face line and site.

6.6.5 Data Interpretation - P30's Maingate

6.6.5.1 Absolute Movement

Figures 6.15 (Vertical) and 6.16 (Inclined) show absolute movement of the uphole anchor horizons, with reference to the basal downhole anchor. These values are tabulated in Tables 6.4 and 6.5. In each case, a good agreement exists between the two wires attached to the topmost anchor.

Both extensometers show simultaneous strata settlement occurring at monitored horizons, as the face advances away from the site. Final strata settlement values are given in Table 6.6. These reveal that the top 10 m section of strata in the vertical hole has settled 3 cm less than the same section in the inclined, 0.15 m compared with 0.18 m. However, in the bottom 20 m strata section, a slightly greater amount of settlement has occurred in the vertical hole rather than the inclined, 0.16 m compared with 0.15 m. In both holes, generalised settlement values of between 0.15 m and 0.18 m exist for all horizons.

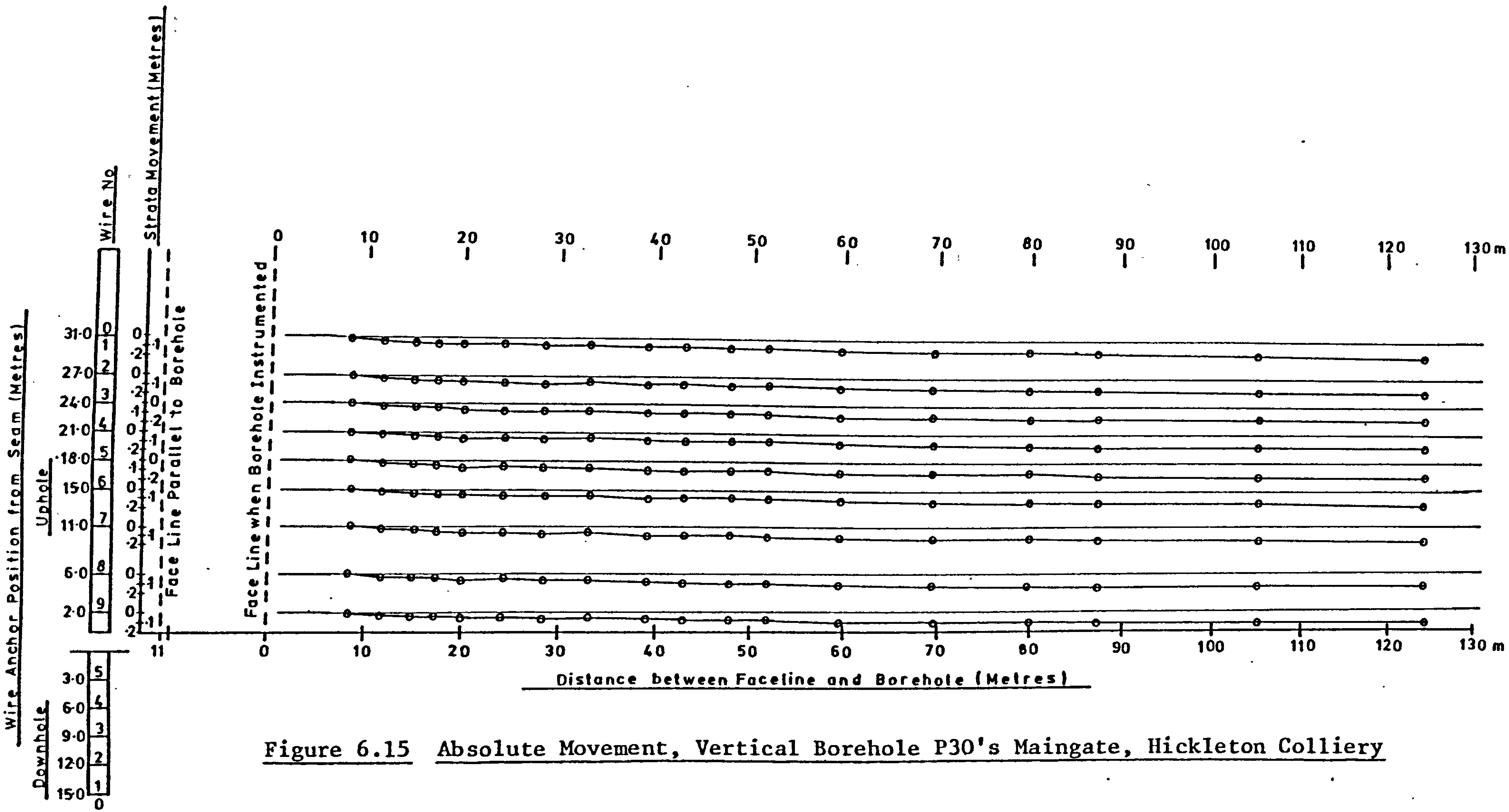


Figure 6.15 Absolute Movement, Vertical Borehole P30's Maingate, Hickleton Colliery

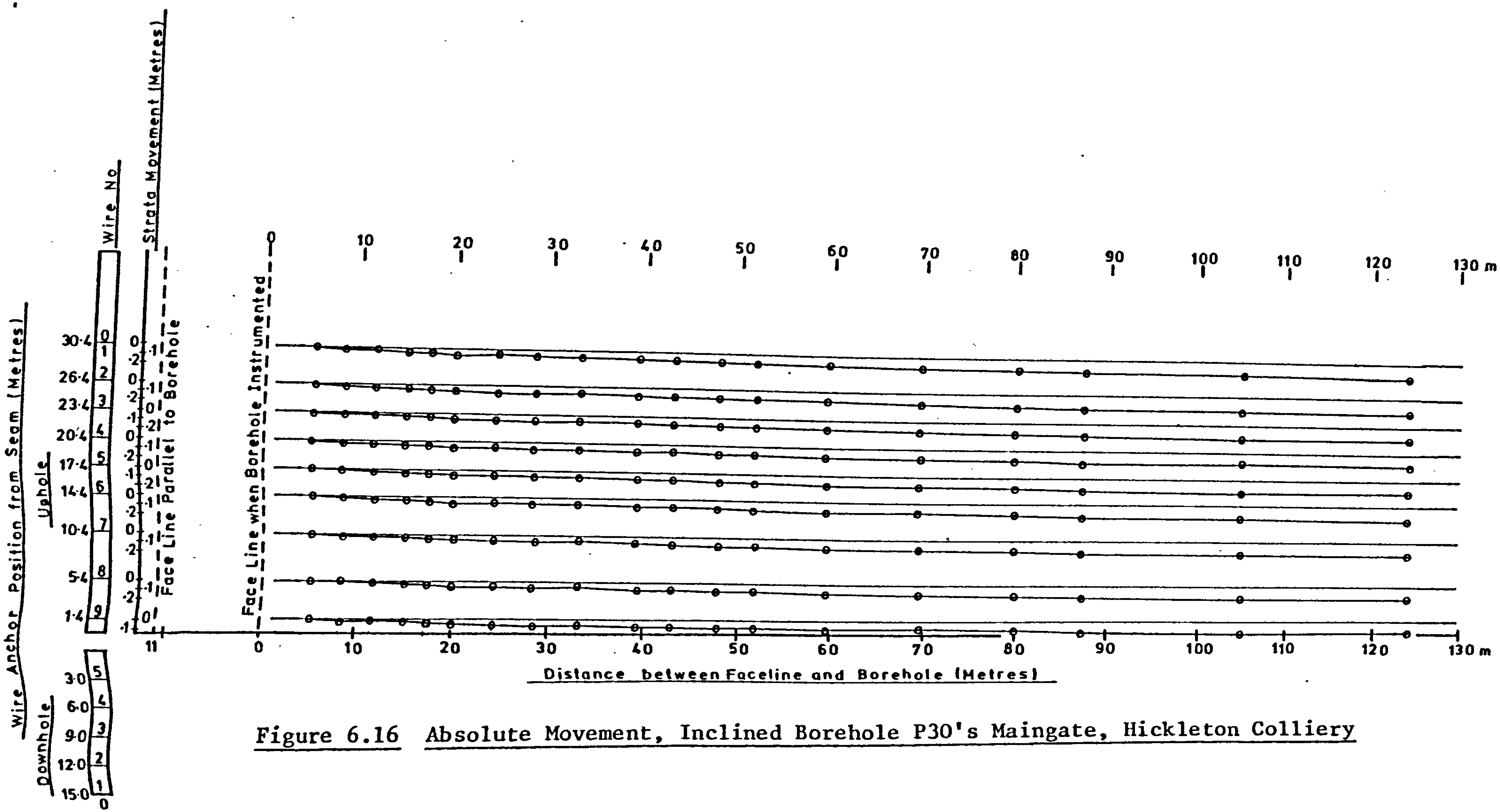


Figure 6.16 Absolute Movement, Inclined Borehole P30's Maingate, Hickleton Colliery

Anchor height above seam (m)	Wire number	19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81
Face position		-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2 (metres)
31.0	0	* -	0.035	0.043	0.048	0.066	0.054	0.072	0.062	0.082	0.090	0.087	0.093	0.110	0.112	0.115	0.124	0.131	0.137	0.153
31.0	1	-	0.039	0.046	0.051	0.061	0.066	0.072	0.056	0.080	0.089	0.082	0.083	0.109	0.111	0.107	0.121	0.126	0.137	0.152
27.0	2	-	0.032	0.043	0.046	0.064	0.060	0.068	0.056	0.083	0.091	0.087	0.095	0.110	0.112	0.116	0.125	0.131	0.140	0.151
24.0	3	-	0.034	0.043	0.048	0.065	0.061	0.072	0.061	0.084	0.095	0.092	0.097	0.112	0.114	0.118	0.128	0.133	0.143	0.157
21.0	4	-	0.028	0.043	0.048	0.064	0.062	0.071	0.062	0.086	0.096	0.092	0.093	0.114	0.116	0.119	0.128	0.133	0.142	0.157
18.0	5	-	0.036	0.046	0.047	0.068	0.065	0.074	0.066	0.092	0.100	0.098	0.104	0.120	0.124	0.123	0.137	0.144	0.156	0.174
15.0	6	-	0.034	0.042	0.047	0.063	0.059	0.070	0.063	0.087	0.095	0.094	0.100	0.117	0.119	0.118	0.133	0.141	0.152	0.172
11.0	7	-	0.032	0.040	0.047	0.065	0.059	0.071	0.063	0.087	0.097	0.094	0.098	0.116	0.118	0.121	0.130	0.140	0.152	0.172
6.0	8	-	0.030	0.039	0.045	0.061	0.056	0.069	0.061	0.082	0.091	0.090	0.097	0.115	0.114	0.114	0.127	0.135	0.146	0.165
2.0	9	-	0.029	0.039	0.044	0.060	0.056	0.068	0.059	0.079	0.089	0.088	0.095	0.108	0.111	0.116	0.126	0.132	0.144	0.161
Face RM (m)		510.7	514.0	517.3	519.7	522.3	526.6	530.7	535.4	541.5	545.3	550.2	554.9	561.9	572.0	582.3	589.7	607.4	626.6	656.6

Vertical hole RM 491.5

Face when hole instrumented RM 502.4

* All absolute movement values in metres.

Table 6.4 Absolute Movement Values, Vertical Borehole P30's Maingate, Hickleton Colliery

Anchor height above seam (m)	Wire Number	19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81
Face position		-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2 (metres)
30.4	0	*0.028	0.034	0.053	0.057	0.071	0.069	0.084	0.077	0.100	0.100	0.107	0.113	0.120	0.126	0.135	0.154	0.149	0.154	0.176
30.4	1	0.031	0.036	0.056	0.060	0.077	0.071	0.088	0.079	0.093	0.096	0.111	0.102	0.120	0.129	0.136	0.157	0.152	0.156	0.180
26.4	2	0.023	0.032	0.046	0.052	0.066	0.068	0.074	0.065	0.089	0.091	0.101	0.095	0.113	0.117	0.128	0.147	0.143	0.149	0.168
23.4	3	0.029	0.035	0.050	0.047	0.066	0.066	0.072	0.066	0.085	0.092	0.101	0.101	0.112	0.120	0.125	0.148	0.145	0.148	0.169
20.4	4	0.027	0.036	0.049	0.053	0.068	0.057	0.073	0.068	0.089	0.091	0.099	0.099	0.114	0.119	0.125	0.144	0.145	0.148	N/A
17.4	5	0.024	0.028	0.041	0.051	0.062	0.059	0.067	0.065	0.086	0.089	0.099	0.098	0.107	0.117	0.125	0.146	0.147	0.151	0.171
14.4	6	0.024	0.028	0.041	0.048	0.064	0.057	0.068	0.064	0.085	0.086	0.097	0.097	0.110	0.115	0.126	0.145	0.146	0.150	0.170
10.4	7	0.026	0.029	0.041	0.050	0.065	0.060	0.070	0.064	0.086	0.087	0.100	0.098	0.112	0.116	0.128	0.145	0.143	0.152	0.174
5.4	8	0.023	0.024	0.037	0.043	0.060	0.055	0.062	0.057	0.079	0.078	0.090	0.090	0.100	0.103	0.118	0.133	0.136	0.139	0.161
1.4	9	0.021	0.019	0.031	0.039	0.054	0.048	0.056	0.050	0.071	0.071	0.082	0.082	0.093	0.097	0.109	0.125	0.124	0.130	0.149
Face RM		510.7	514.0	517.3	519.7	522.3	526.6	530.7	535.4	541.5	545.3	550.2	554.2	561.9	572.0	582.3	589.7	607.4	626.6	656.6

Inclined hole RM - 491.5

Face when hole instrumented RM - 502.4

* All absolute movement values in metres.

Table 6.5 Absolute Movement Values, Inclined Borehole P30's Maingate, Hickleton Colliery

Wire No	Height (Metres)	Vertical Borehole	Height (Metres)	Inclined Borehole
0	31.0	0.153	30.4	0.176
1	31.0	0.152	30.4	0.180
2	27.0	0.151	26.4	0.168
3	24.0	0.157	23.4	0.169
4	21.0	0.157	20.4	N/A
5	18.0	0.174	17.4	0.171
6	15.0	0.172	14.4	0.170
7	11.0	0.172	10.4	0.174
8	6.0	0.165	5.4	0.161
9	2.0	0.161	1.4	0.149

Table 6.6 Final Strata Settlement Values, P30's Maingate, Hickleton Colliery

Examination of Figures 6.15 and 6.16 reveals that the maximum settlement rate occurs between 10 m and 30 m passed the site. Between 30 m and 60 m the rate slows slightly and after 60 m it slows considerably. In general, greater strata settlement seen in the inclined hole can be attributed either to cantilevering effects or settlement over the goaf area, as would be expected. The observed settlement values of between 0.15 m and 0.18 m do however suggest, that the Parkgate Rock is not bridging across the panel.

6.6.5.2 Bed Separation

Tables 6.7 and 6.8 list relative amounts of bed separation occurring between adjacent strata horizons, in both boreholes. Although the results are not graphically represented, both holes show considerable fluctuations between closure and separation, not only within the hole but also with respect to face advance. Comparison with the tailgate values again reveals no simple relationship between the nature and magnitude of the values and face advance.

However, the results do suggest that strata is undergoing differential movement, possibly associated with pre-existing separations or fissures, although since no adjacent goaf areas exist, these cannot be explained by the proximity of historical workings. Bed separations experienced at the maingate site are in general much smaller in magnitude than equivalent tailgate values.

6.6.5.3 Vertical Strain Development

Vertical strain development between adjacent anchor horizons for both boreholes is shown in Figures 6.17 and 6.18 and tabulated in Tables 6.9 and 6.10.

Date		19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81	
Face position		-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2	(metres)
Anchor Horizon	0 - 2	*-1.5	-2.0	+3.5	-2.0	-0.5	+8.5	-10.0	-1.5	+6.5	-0.5	-0.5	+1.5	-1.5	0.0	+1.0	+0.5	-1.5	+2.5	-4.5	
	1 - 2	-6.0	-1.0	+3.5	-1.5	+7.5	-8.5	+2.5	+4.0	0.0	+1.0	+3.5	+6.5	-11.0	+1.0	+7.5	-4.0	0.0	-3.0	-2.5	
	2 - 3	+1.5	+0.5	-1.5	+1.5	+0.5	0.0	+2.0	+1.0	-3.5	+3.0	+0.5	-2.5	+0.5	0.0	-1.5	+2.0	-1.0	+1.0	+3.5	
	3 - 4	+0.5	-1.0	+0.5	0.0	-1.0	+1.5	-1.0	+2.0	+0.5	-1.0	-1.0	-3.5	+5.0	0.0	+0.5	-2.5	+0.5	0.0	0.0	
	4 - 5	+3.0	0.0	0.0	-4.5	+5.5	-1.0	-0.5	+1.5	+2.0	-1.5	+1.5	+4.5	-4.0	+2.0	-5.0	+6.0	+1.5	+3.0	0.0	
	5 - 6	-6.0	+4.5	-3.0	+4.5	-5.5	0.0	+1.5	+0.5	-1.0	-0.5	+1.5	0.0	+0.5	-2.5	+0.5	+0.5	+1.0	-0.5	+2.0	
	6 - 7	+1.5	-3.5	0.0	+2.0	+2.0	-2.0	+1.5	0.0	-1.5	+2.0	-2.5	-2.0	+1.5	+0.5	+4.0	-6.0	+2.5	0.0	+0.5	
	7 - 8	-3.0	+1.0	+1.0	-0.5	-2.0	+0.5	+0.5	+0.5	-3.0	-1.5	+3.0	+2.5	-0.5	-3.0	-3.0	+4.0	-2.0	-1.0	-1.0	
	8 - 9	+0.5	-1.0	+1.0	-1.5	-0.5	+1.0	0.0	-1.5	+4.0	-4.0	0.0	0.0	-5.0	+4.5	+2.5	-1.0	-2.0	+1.5	-2.5	

*All separation values in millimetres

(-) = Closure (+) = Separation

Table 6.7 Bed Separation Values, Vertical Borehole P30's Maingate, Hickleton Colliery

	Date	19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81
	Face position	-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2 (metres)
Anchor Horizon	0 - 2	*-5.0	+3.0	-4.0	+2.5	+0.5	+3.5	-9.5	-1.5	-3.0	+4.0	+4.5	-12.0	+11.0	-3.0	+3.5	-1.0	+2.0	+0.5	-2.5
	1 - 2	-8.0	+4.5	-6.5	+3.0	-3.5	+7.5	-10.5	0.0	+3.5	+2.0	-1.5	+2.0	+1.0	-6.0	+4.5	-3.5	+2.5	+2.0	-3.5
	2 - 3	+6.0	-3.5	+1.5	-9.0	+5.0	-2.5	+0.5	+2.5	-1.0	+2.5	-2.5	+7.0	-7.5	+3.5	-5.5	+4.0	+1.0	-3.0	+1.5
	3 - 4	-2.0	+3.0	-2.0	+7.0	-4.5	-10.5	+10.5	+1.0	+1.5	-5.0	-1.0	+0.5	+3.5	-3.0	+1.5	-4.0	+4.0	0.0	N/A
	4 - 5	-2.5	-5.5	0.0	+4.0	-2.5	+8.5	-8.5	+3.5	0.0	+1.0	+2.5	-1.5	-6.0	+5.5	+1.0	+2.0	+0.5	+1.0	N/A
	5 - 6	-0.5	+0.5	-1.0	-0.5	+3.0	-3.5	+3.0	-3.0	+1.0	-2.0	+1.0	+0.5	+4.0	-4.5	+3.5	-3.0	0.0	0.0	+0.5
	6 - 7	+2.0	-0.5	-1.5	+2.0	-1.0	+2.0	-1.0	-2.0	+0.5	+1.0	+1.0	-1.5	+1.5	-1.0	+0.5	-2.0	-2.5	+5.0	+2.0
	7 - 8	-2.5	-2.5	+2.5	-4.0	+2.0	-1.0	-2.5	+0.5	+1.0	-2.5	-1.5	+1.5	-2.5	-2.0	+3.0	-0.5	-5.0	-7.0	0.0
	8 - 9	-2.0	-2.5	-2.0	+3.0	-2.5	-0.5	0.0	0.0	-1.0	+1.0	-1.0	+0.5	-1.0	+1.5	-2.5	+0.5	-3.0	+3.0	-4.0

All values corrected for inclination of 75° i.e. (cos15° = 0.966) * All separation values in millimetres (mm)

(-) = Closure (+) = Separation

Table 6.8 Bed Separation Values, Inclined Borehole P30's Maingate, Hickleton Colliery

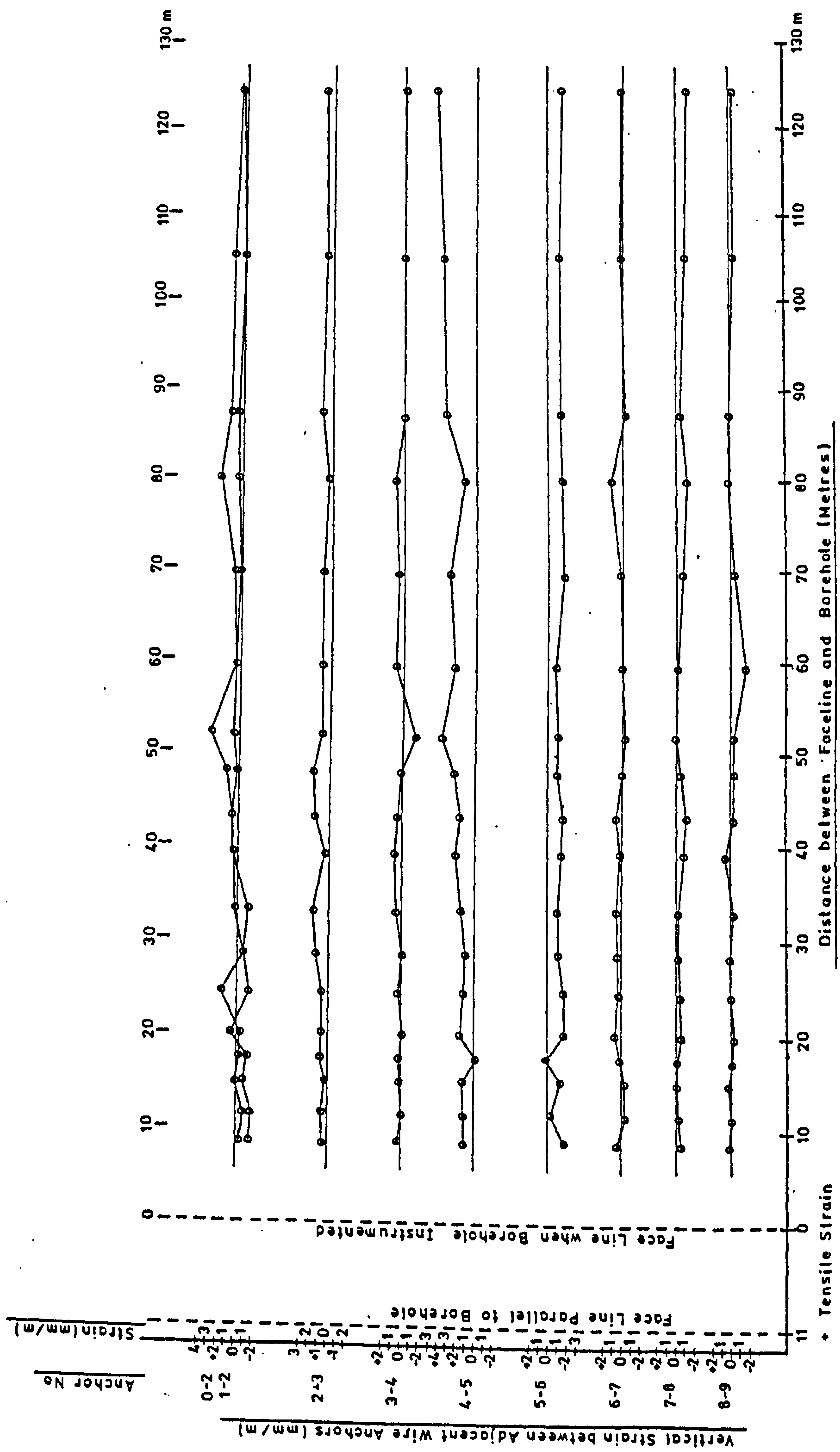


Figure 6.17 Vertical Strain, Vertical Borehole P30's Maingate, Hickleton Colliery

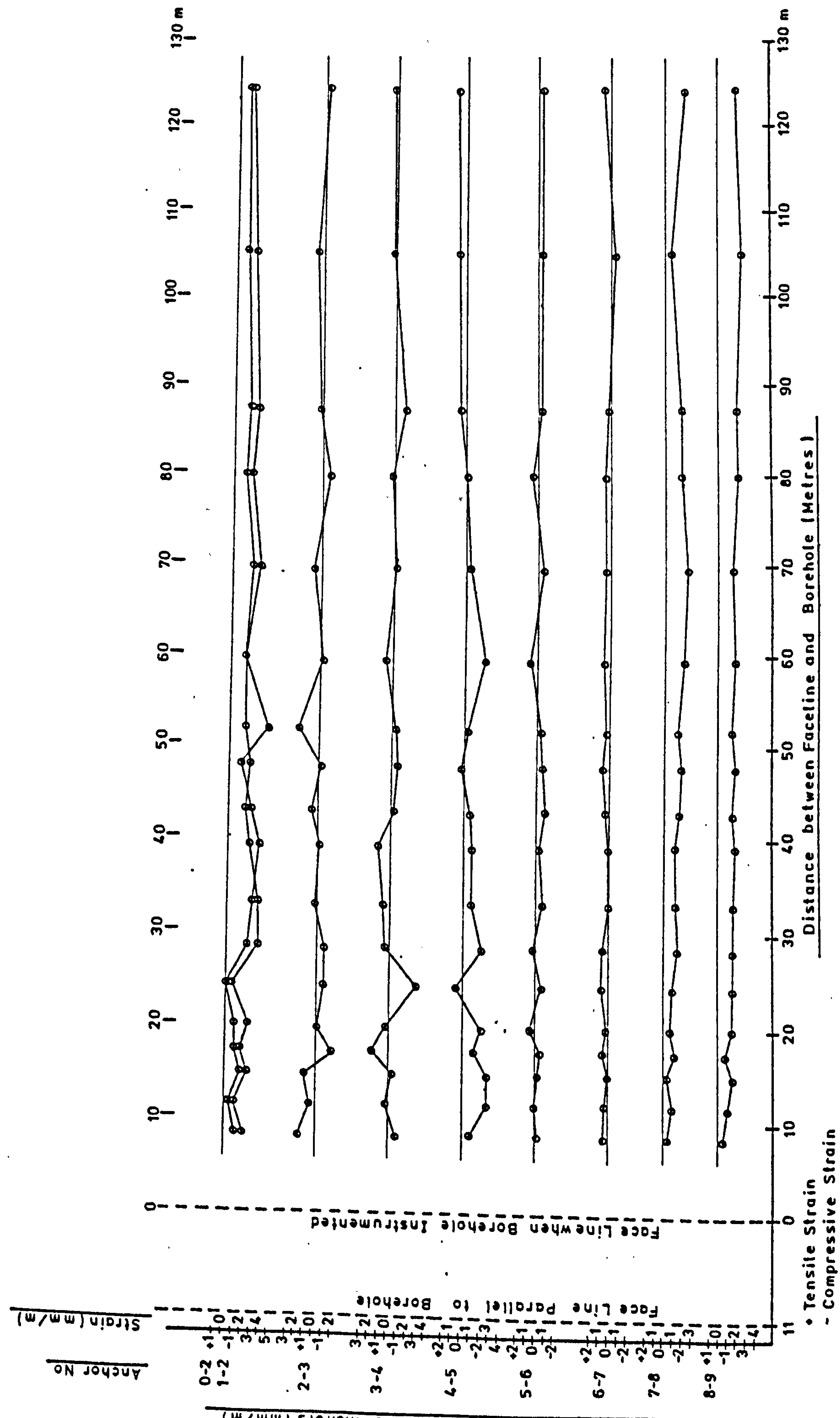


Figure 6.18 Vertical Strain, Inclined Borehole P30's Maingate, Hickleton Colliery

Date		19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81
Face position		-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2 (metres)
Anchor Horizon	0 - 2	*-0.4	-0.9	0.0	-0.5	-0.6	+1.5	-1.0	-1.4	+0.2	+0.1	0.0	+0.4	0.0	0.0	+0.2	+0.4	0.0	+0.6	-0.5
	1 - 2	-1.5	-1.7	-0.9	-1.2	+0.6	-1.5	-0.9	+0.1	+0.1	+0.4	+1.2	+2.9	+0.1	+0.4	+2.2	+1.2	+1.2	+0.5	-0.1
	2 - 3	+0.5	+0.7	+0.2	+0.7	+0.5	+0.5	+1.2	+1.5	+0.3	+1.3	+1.5	+0.7	+0.8	+0.8	+0.3	+1.0	+0.7	+1.0	+2.2
	3 - 4	+0.2	-0.2	0.0	0.0	-0.3	+0.2	-0.2	+0.5	+0.7	+0.3	0.0	-1.2	+0.5	+0.5	+0.7	-0.2	0.0	0.0	0.0
	4 - 5	+1.0	+1.0	+1.0	-0.5	+1.3	+1.0	+0.8	+1.3	+2.0	+1.5	+2.0	+3.5	+2.2	+2.8	+1.2	+3.2	+3.7	+4.7	+4.7
	5 - 6	-2.0	-0.5	-1.5	0.0	-1.8	-1.8	-1.3	-1.2	-1.5	-1.7	-1.2	-1.2	-1.0	-1.8	-1.7	-1.5	-1.2	-1.3	-0.7
	6 - 7	+0.4	-0.5	-0.5	0.0	+0.5	0.0	+0.4	+0.4	0.0	+0.5	-0.1	-0.6	-0.2	-0.1	+0.9	-0.6	0.0	0.0	+0.1
	7 - 8	-0.6	-0.4	-0.2	-0.3	-0.7	-0.6	-0.5	-0.4	-1.0	-1.3	-0.7	-0.2	-0.3	-0.9	-1.5	-0.7	-1.1	-1.3	-1.5
	8 - 9	+0.1	-0.1	+0.1	-0.2	-0.4	-0.1	-0.1	-0.5	+0.5	-0.5	-0.5	-0.5	-1.7	-0.6	0.0	-0.2	-0.7	-0.4	-1.0

* All strain values in mm/m

\- = Compression
\+ = Tension

Table 6.9 Vertical Strain Values, Vertical Borehole P30's Maingate, Hickleton Colliery

	Date	19/9	23/9	25/9	29/9	1/10	3/10	7/10	9/10	14/10	16/10	21/10	23/10	29/10	5/11	12/11	19/11	3/12	17/12	21/1/81
	Face position	-8.3	-11.6	-14.9	-17.3	-19.9	-24.2	-28.3	-33.0	-39.1	-42.9	-47.8	-51.8	-59.5	-69.6	-79.9	-87.3	-105.0	-124.2	-154.2 (metres)
Anchor Horizon	0 - 2	*-1.2	-0.5	-1.7	-1.1	-1.0	-0.1	-2.5	-2.9	-3.6	-2.6	-1.5	-4.5	-1.7	-2.5	-1.6	-1.9	-1.4	-1.2	-1.9
	1 - 2	-2.0	-0.9	-2.5	-1.7	-2.6	-0.7	-3.4	-3.4	-2.5	-2.0	-2.4	-1.9	-1.6	-3.1	-2.0	-2.9	-2.2	-1.7	-2.9
	2 - 3	+2.0	+0.8	+1.3	-1.7	0.0	-0.8	-0.7	+0.2	-0.2	+0.7	-0.2	+2.2	-0.3	+0.8	-1.0	+0.3	+0.7	-0.3	+0.2
	3 - 4	-0.7	+0.3	-0.3	+2.0	+0.5	-3.0	+0.5	+0.8	+1.3	-0.3	-0.7	-0.5	+0.7	-0.3	+0.2	-1.2	+0.2	+0.2	N/A
	4 - 5	-0.8	-2.7	-2.7	-1.3	-2.2	+0.7	-2.2	-1.0	-1.0	-0.7	+0.2	-0.3	-2.3	-0.5	-0.2	+0.5	+0.7	+1.0	N/A
	5 - 6	-0.2	0.0	-0.3	-0.5	+0.5	-0.7	+0.3	-0.7	-0.3	-1.0	-0.7	-0.5	+0.8	-0.7	+0.5	-0.5	-0.5	-0.5	-0.3
	6 - 7	+0.5	+0.4	0.0	+0.5	+0.2	+0.7	+0.5	0.0	+0.1	+0.4	+0.6	+0.2	+0.6	+0.4	+0.5	0.0	-0.6	+0.6	+1.1
	7 - 8	-0.5	-1.0	-0.5	-1.3	-0.9	-1.1	-1.6	-1.5	-1.3	-1.8	-2.1	-1.8	-2.3	-2.7	-2.1	-2.2	-1.2	-2.6	-2.6
	8 - 9	-0.4	-1.1	-1.6	-0.9	-1.5	-1.6	-1.6	-1.6	-1.9	-1.6	-1.9	-1.7	-2.0	-1.6	-2.2	-2.1	-2.9	-2.1	-3.1

All values have been corrected for an inclination of 75° ($\cos 15^{\circ} = 0.966$)   * All strain values in mm/m.

\- = Compression

\+ = Tension

Table 6.10 Vertical Strain Values, Inclined Borehole P30's Maingate, Hickleton Colliery

Values fluctuating in both magnitude and sign are seen at all levels within both holes. Compressive strain indicates closure and tensile separation. In the first 18 m section of each hole, compressive strains predominate, while in the top 12 m section tensile strains predominate. This is particularly noticeable in the vertical hole. However, exceptional horizons do exist (inclined anchors 6 - 7). Similarly, strains encountered in the inclined hole are greater in magnitude than those in the vertical.

Examination of Figures 6.17 and 6.18 reveals that as the face advances away from the site, the magnitude of the fluctuations decreases and by 120 m passed, near stable conditions exist. As with the tailgate site, fluctuating values have again made it impossible to draw meaningful vertical strain development distributions for strata surrounding the site. Similarly, since vertical strain is a function of bed separation, graphical representation of the bed separation results would produce a series of fluctuating lines similar to those seen in Figures 6.17 and 6.18.

The general stabilisation of strain values with increased face distance from site can almost certainly be associated with goaf consolidation effects. Therefore, since initial fracturing cannot be associated with historical workings, it must have occurred in the immediate face vicinity. Unfortunately, installation of the instrumentation missed this crucial phase and the results only show differential movement associated with consolidation of the broken strata into the goaf.

6.6.5.4 Roadway Closure

Figure 6.19 shows vertical closure between the up and downhole baseplates for both the vertical and inclined boreholes. In each case, the closure rate increases rapidly until the face is 40 m from site (0.11 m of closure). A reduction in the closure rate then occurs until 90 m (0.07 m of closure), after which a further reduction occurs until 150 m (0.03 m of closure). A total closure of 0.21 m was therefore experienced for a face advance of 150 m passed the site. No width measurements were taken during the investigation, due to the presence of face end machinery in the initial stages and the conveyor during the latter.

6.7 Comparison of the Main and Tailgate Results

Anchor horizons in the maingate boreholes were calculated to coincide with those of the tailgate, in order to allow a direct comparison of the results. Equivalent anchor horizons are Nos. 0 - 9 in the maingate and Nos. 12 - 20 in the tailgate. Monitored face advance in the tailgate varied from 45 m behind to 55 m in front of the site. A comparison of results can therefore only be made in the compatible face advance range of 10 m to 55 m in front of the site.

Table 6.11 shows cumulative amounts of strata movement in each of the three upholes, for a face advance of 10 m to 55 m in front of the site. Each value at a similar horizon appears remarkably consistent, although the tailgate values are persistently some 1.5 times greater than equivalent maingate values. The difference

Anchor Number		Height of Anchor Horizon above Seam (metres)			Total Amount of Strata Settlement (metres)*		
Tailgate	Maingate	Tailgate	Maingate	Maingate	Tailgate	Maingate	Maingate
			Vertical Hole	Inclined Hole		Vertical Hole	Inclined Hole
12	0	30.5	31.0	30.4	0.123	0.075	0.086
12	1	30.5	31.0	30.4	0.123	0.070	0.084
13	2	27.5	27.0	26.4	0.130	0.078	0.081
14	3	24.5	24.0	23.4	0.126	0.069	0.077
15	4	21.4	21.0	20.4	0.133	0.086	0.078
16	5	18.4	18.0	17.4	0.131	0.084	0.079
17	6	15.3	15.0	14.4	0.123	0.083	0.082
18	7	10.7	11.0	10.4	0.129	0.084	0.083
19	8	6.2	6.0	5.4	0.123	0.085	0.076
20	9	1.6	2.0	1.4	0.126	0.079	0.074

* All values for range -10 to - 55 m behind the face line.

Table 6.11 Total Strata Movement (Absolute) in P30's Main and Tailgate Boreholes for Face Range -10 to -55 m between the maingate inclined and vertical hole results remains small. Slightly greater movement was recorded in the maingate inclined hole rather than the vertical, particularly in the top section, since this is located further over the goaf and therefore subject to greater subsidence.

A comparison of bed separation and vertical strain values is considerably more difficult due to the marked fluctuations encountered at all levels. However, tailgate values were found to be consistently greater in magnitude than those experienced in the maingate.

A comparison of vertical roadway closure results, over the range 10 m to 55 m in front of the site, shows 0.23 m of closure in the tailgate compared with only 0.12 m in the maingate. Movement in the tailgate is therefore almost twice that experienced in the maingate. Similarly, although problems with floor heave were encountered in the tailgate, only minor problems were experienced in the maingate. Finally, it is suggested that greater settlement recorded in the tailgate can be associated with complete extraction of the P29 and P30 panels. In the maingate, however, the site is located adjacent to a ribside pillar and therefore only subject to partial subsidence.

6.8 Discussion of Results

In the authors opinion, although the results are discussed in Sections 6.5, 6.6 and 6.7, there are certain points which do not fall into any of these sections, but which are considered sufficiently important to warrant further discussion.

An examination of the tailgate results reveals that a value of 20 m in advance of the face is significant, since in Figure 6.10 the lower strata horizons begin to show increased settlement rates at this point. Similarly, in the roadway, the vertical closure rate also begins to increase. Additional evidence collected during the investigation, revealed that conditions in the tailgate started to deteriorate some 10 - 15 m in advance of the face, with the onset of roadway disruption and floor heave.

The total amount of roadway closure monitored in the tailgate was 0.64 m, while 6 m above (anchor 19) only 0.30 m of settlement was experienced. Anchor 20, 1.6 m above, may or may not be situated in the borehole standpipe. It is thought that differences between these values could be explained by two interrelated factors. Strata in the roadway roof and associated with the yield zone, allowed 0.64 m of closure. However, above say 4 - 6 m, time dependent properties of the strata begin to occur. Originally, working P29 would have caused relatively little disruption in the strata of lower sequences above the ribside, therefore making it more competent. However, the packside coal pillar left during extraction of P30, will still provide support for strata in the roadway vicinity, but not necessarily in the immediate roof. A significant portion of the recorded closure may therefore be due to reactivation of the yield zone around the tailgate by extraction of the P30 panel.

Examination of the maingate results, reveals that a greater amount of settlement is occurring within the inclined hole rather than the vertical. This can be attributed to either cantilevering effects or greater settlement over the goaf area. However, particularly in the lower strata regions of both holes, significant support will be provided by the ribside pillar, roadway supports and gateside pack system. Not only will these factors control the amount of settlement recorded in the lower strata section, but also indirectly, the amount recorded in the upper section. Additional evidence suggested that weight was occurring in the maingate some 200 - 300 m behind the face and that the settlement rate therefore contains a time dependent factor. It would have been interesting to monitor the site at 200 m, 250 m and 300 m intervals behind the face and determine whether these effects were occurring throughout the roadway or as just localised manifestations.

A comparison of vertical strain development values in the maingate boreholes reveals more tensile strain being experienced in the vertical hole and more compressive in the inclined. If this is related to the fact that more settlement is recorded in the inclined hole rather than the vertical, then this could be associated with a 'slumping' of the strata towards the goaf area, in a type of 'cantilevering' action. However, it should be emphasised that the cantilever is not composed of massive sandstone beams resting on a thin bed of caved material, but rather a mass of fractured strata which has 'slumped' towards the goaf. The amount of recorded movement in the two holes, rules out strata bridging across the panel. Similarly, the amount of recorded movement and bed separation suggests that the strata is breaking at all levels and therefore moving towards the goaf area. More tensile strain should therefore be developed in the vertical hole and more compressive in the inclined, and this is shown by the results.

Fluctuating values of bed separation and vertical strain seen as the face advances away from the maingate site, are probably due to fracturing which occurs in the face vicinity. Evidence from the tailgate site suggests that fracturing may be starting to occur up to 20 m in advance of the face. Installation of the maingate site 10 m behind the face will therefore have missed this crucial stage and the results only show differential movement associated with consolidation of the broken strata over the goaf. In theory, these effects should decrease with increased face distance from the site and this is confirmed by the results.

6.9 Proposed Method of Working P28's Panel

Interpretation of data collected from the P30 sites suggests that fracturing of the Parkgate Rock is occurring along the rib-side zones and 'normal' caving is occurring over the goaf. It is therefore concluded that extraction of P28's panel can be successfully achieved provided special attention is given to face design. Whittaker (74) lists six points which need specific consideration:

1) Correct location and support of the face heading.

2) Avoidance of coal ribs/fenders which could create spasmodic caving.

3) Choice of an effective face support system which has a proven history of successful operation in the Parkgate or similar seams.

4) Effective support of the face-end areas and gates within 30 m of the face line.

5) The face should be cut end-to-end.

6) The face should be monitored using specialist strata control techniques to establish the general strata behaviour.

Work undertaken on P30's panel has high-lighted three potential schemes of 'real-time' monitoring instrumentation which could be used during the extraction of P28 to determine insitu stresses and strata behaviour. The three methods which could be used either together or separately to form the investigation are:

1) Extensometer methods.

2) Permeability methods.

3) 'Insitu seam' stress determination methods.

The extensometer methods are fully described in this chapter, while the permeability methods have already been discussed in Chapter 5. The final method, 'insitu seam' stress determination is described more fully in Chapter 7. However, if trials prove successful, it is probable that the third method will be adopted as a single investigative scheme. However, until it has been fully tested, it would be wise to supplement it with a secondary scheme of either extensometer or permeability methods, both of which have their relative merits.

6.10 Conclusions

At Hickleton Colliery, extensometers were installed on the P30 panel to determine caving characteristics of the Parkgate Rock and assess its relationship to the formation of weight bump conditions. Five extensometers were successfully installed at two sites in the main and tailgates. The tailgate site involved the successful installation and subsequent monitoring of a 68 m uphole extensometer containing 20 monitoring horizons.

No bumps of significance were recorded throughout the period of investigation or during the panels lifetime. Contrary to prevailing theories, the results indicate that the Parkgate Rock is fracturing both over the goaf and along the ribside. Similarly, the results suggest that the strata cannot bridge one panel let alone two and it is concluded that the effects of bumps experienced on the 324 and 325 faces was magnified by the type of face support system then in operation (props and bars). It is significant to note that since the introduction of modern powered face support systems, no bumps of even vaguely similar magnitude have been recorded. However, this should not be interpreted as being the answer to the problem, since the actual bump mechanism is still far from being fully understood.

Finally, it is concluded that the instrumentation installed on P30's panel was successful in monitoring strata conditions above the face line. A variety of schemes were used, which included several novel yet successful innovations on current techniques. The results indicate that the strata above P30's panel was fracturing and undergoing settlement, and that the proposed P28 panel could be worked successfully, provided special attention was paid to face design and a 'real-time' monitoring scheme installed to assess insitu strata stresses.

CHAPTER 7

GEOLOGICAL AND GEOTECHNICAL ASPECTS OF STRATA CONTROL INVESTIGATIONS

CHAPTER 7

GEOLOGICAL AND GEOTECHNICAL ASPECTS OF STRATA CONTROL INVESTIGATIONS

7.1 Introduction

When conducting any type of strata control investigation, whether into caving characteristics, roadway deformation or permeability changes surrounding an extraction panel, it is important to remember that the geological environment will always exert a significant influence on the final result. Floor heave problems in roadways can always be associated with the presence of seat earths near the seam base. The presence of structural discontinuities can effect the stability of strata surrounding gateroads. Similarly, the occurrence of water on a face is dependent upon the presence of aquicludes and aquitards within the intervening strata between the extraction and aquifer horizons.

It is therefore intended to use this chapter to illustrate various geological and geotechnical techniques, which can be applied during the course of an investigation. However, rather than just present these techniques, it is proposed to use work undertaken at Hickleton Colliery, NCB Doncaster Area, to illustrate some of the methods available. It is also intended to discuss a proposed method of in-seam stress determination for coal seams, which in the authors opinion could have important applications in areas with potential bump conditions.

7.2 Geologist vs. Engineer - The Conflict

When mining problems of any type are encountered, there is a tendency to either ignore or only give low priority to the geological considerations. Historically, this is probably due to two inter-related viewpoints. Few mining engineers appear to understand the concepts of geology and essentially work in a practical, real-time environment. Geologists on the other hand, tend from their training to consider problems partly from a practical viewpoint, but more significantly from a theoretical or academic stand. A rapid concise answer to an engineers question will therefore tend to become drawn out, either intentionally or otherwise into a discussion on the 'pro's and con's' or relative merits of a particular course of action. It is therefore easy to understand why mining engineers would prefer to employ 'one-handed geologists'. Geological information should be presented in a concise, definite and positive manner, with a bias towards practical implications.

An example of this communication problem can be illustrated by a theoretical classification of the Coal Measures cyclotherm:

Coal	Siltstone
Seatearth	Sandy Siltstone
Carbonaceous Mudstone	Silty Sandstone
Mudstone	Fine Sandstone
Silty Mudstone	Medium Sandstone
Muddy Siltstone	Coarse Sandstone

Many of the cyclotherm sequences cannot be readily identified except by using thin section microscope techniques. A mining engineer will usually only want to know whether a particular strata type is sandstone, mudstone, seatearth or coal and is probably more interested in its properties rather than its name. The theoretical finesse therefore becomes merely irrelevant and superfluous data, impractical in the day to day operations of coal winning.

In addition, the interpretation of terms between engineers and geologists also exhibits divergence. At Hickleton, the Parkgate Rock was initially described as a strong massive sandstone and this was interpreted by engineers as meaning homogeneous and relatively isotropic at all levels. In geological terms, near homogeneous and isotropic sandstone sequences 30 m thick in the Coal Measures, are almost unknown. A study of Coal Measures lithology will reveal variations both horizontally and vertically across an area. Similarly, sedimentary features are always present in the form of bedding planes and structural features in the form of joint networks and compaction faults. All Coal Measures sequences therefore exhibit both lithological and structural anisotropy. Subsequent examination of Parkgate Rock core, revealed approximately 30 m of interbedded sandstones, showing minor lithological variations containing isolated ironstone nodules and exhibiting well developed bedding features on both a micro and macro scale. Evidence of a natural joint network was also found. The interpretation of a massive sandstone sequence, therefore appears to be one thing to a mining engineer and another to a geologist.

7.3 Geological Investigations

A number of geological techniques are available for investigating mining and related problems. Initially, the first stage in an investigation is to either determine the overlying geology or reappraise the existing information. Reference will therefore be made to work carried out at Hickleton, in order to illustrate the practical applications of these techniques and highlight the often difficult task of making specific interpretations from often scarce and scant information.

7.3.1 Reappraisal of Existing Geological Data

When the Hickleton investigation commenced, geological information concerning the roof strata over the P30 and P28 panels was scarce. Figure 7.1 shows the full extent of the available geological borehole information. In essence, it consists of only one borehole, UGBH No. 1, which is situated near the southern end of the P21 panel in the East Parkgate workings. Other information shown consists of a shaft section for the nearby Brodsworth Colliery and UGBH No. 26 which is situated in adjacent Goldthorpe workings. In addition, the Hickleton shaft sections were also available, but considered to be of secondary importance, since the shafts are located some 2 - 3 km to the west of the area under consideration. Finally, many records exist for geological sections taken across roadway headings and although extremely detailed, only consider strata to a maximum height of about 3 m above seam. The accurate interpolation of a strata section above any of the East Parkgate panels is therefore difficult and should be treated with caution.

7.3.1.1 East Parkgate Geological Environment

Workings in the East Parkgate are situated on the eastern side of a large washout structure which bisects the Parkgate seam into the East and West areas. It has an effective width in the region of

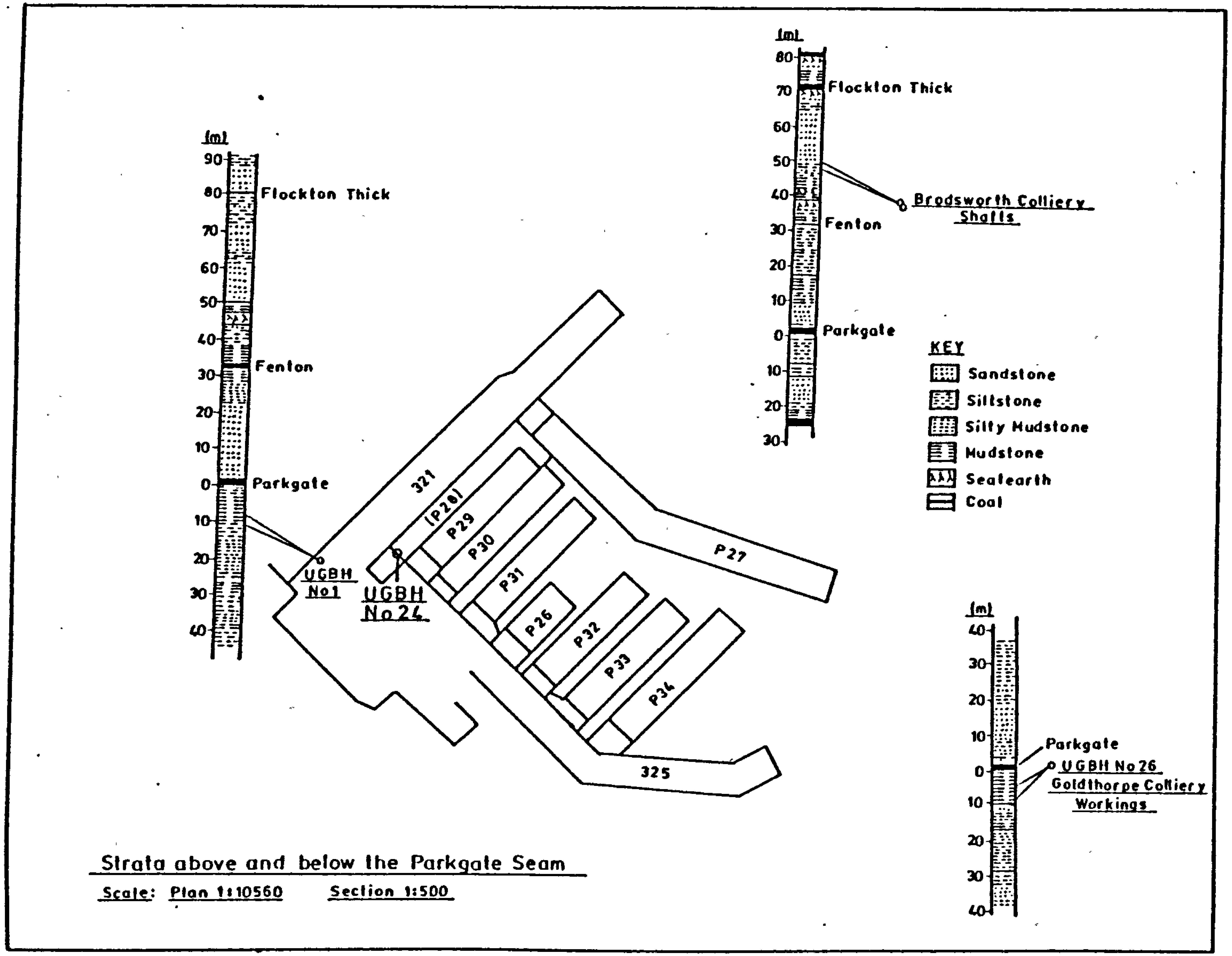


Figure 7.1 Geological Information available at the start of the Investigation, East Parkgate Area

1 km, although the actual cut-off point is unknown away from the main transport routes. Structurally, it is thought that the thick sandstone/siltstone sequences which comprise it, thin gradually with increased distance away from it. No large or medium sized faults cross the area (75).

Evidence that the Parkgate Rock thins away from the washout is provided by Figure 7.1, although the distance of the Brodsworth and UGBH No. 26 sections from the area should instill care into this type of interpretation. The thickness of Parkgate Rock derived from Figure 7.1 and other sources is:

U.G.B.H. No. 1		23 m
Brodsworth Shafts		10 m
U.G.B.H. No. 26	≃	12 m
Hickleton Shafts	≃	23 m

Based on this information, the thickness of Parkgate Rock normally assumed to overlie the East Parkgate area is 30 m (100 ft).

Geologically, the seam is immediately overlain by a shaley sandstone varying in thickness from 0.3 m to 2.5 m, which contains numerous coal streaks. The band of shaley sandstone is clearly visible in Plate 11. This in turn is overlain by a thick, 15 - 25 m, sandstone sequence with a variable arenaceous lithology (Parkgate Rock). In its topmost section, the Parkgate Rock begins to grade rapidly into a series of siltstones and mudstones, which eventually form a seatearth underlying the Fenton seam. Above the Fenton seam lies a series of interbedded mudstones, siltstones and sandstones which eventually culminate in a seatearth underlying the Flockton Thick seam, Figure 7.1.

Plate 11

P30's Maingate Heading showing
the Parkgate Rock and basal shaley
sandstone sequence

Examination of the existing geological data does not therefore reveal any new information.

7.3.2 Examination of Drill Cuttings

One of the simplest and cheapest methods of obtaining information in an area void of geological data is to collect and examine drill cuttings from methane drainage or instrumentation boreholes. In the initial stages of the investigation, drilling of the P30 tailgate boreholes had been completed before the lack of detailed geological information in the area, had been fully appreciated.

The second stage of the investigation, drilling the inclined permeability boreholes, should have provided an excellent opportunity for obtaining additional geological data on the strata overlying P30's panel. However, from the beginning it was found to be prohibitive in both cost and time to continually have on site either a member of the NCB Geological Staff or the University. It was therefore arranged for the drillers to collect a small sample of cuttings each time a new drill rod was added. Unfortunately for a variety of reasons no identifiable samples were obtained. Similarly, no samples were obtained from either of the two maingate boreholes.

Finally, it is thought that in addition to the collection of drill cuttings, it would also have been useful to log drill penetration rates for subsequent analysis and comparison with geological sections.

7.3.3 The Cored Borehole

The most accurate method of determining the geological sequence of an area is to drill several cored boreholes and examine the resultant cores.

At a meeting on 5th September 1980 (76), it was decided to drill a cored borehole over the P30 and P28 panels and determine the precise nature of the Parkgate Rock. A seam depth of 900 m prohibited a surface borehole and it was therefore decided to use an underground site with a cored uphole. The borehole would be used to derive information from two sources:

1) On-site logging would provide detailed information on lithological sequences comprising the Parkgate Rock and strata above the Fenton seam. The existence of planes of natural weakness and potential separation could also be determined.

2) After completion of the logging, the core would be transported to Nottingham University for subsequent geotechnical logging and examination.

Drilling operations by the Area Drillers were started at the beginning of October and completed by the beginning of November. The final borehole length was 72 m (237 ft) with a core diameter of 72 mm.

7.3.3.1 Borehole Site Location

To achieve maximum benefit from data yielded by the cored borehole, it was important to decide upon a site which fulfilled certain criteria:

1) The borehole should be sited in stable ground to provide easy drilling conditions for good core recovery.

2) The site should be sufficiently close to the P28 and P30 panels to allow reasonable geological interpretation.

3) The site must be readily accessible for both men and materials as well as close to the main services of water, power and compressed air.

Consultation with the Colliery Manager, Surveyor and Mine Geologist located such a site in P25's maingate scour, Figure 7.1. The core was logged on site by the Mine Geologist and the author before transportation to the University for geotechnical examination.

7.3.4 The Geological Log

Essentially, geological logging involves the examination of core lithology and the interpretation of actual and recorded 'pull-out' lengths. Cyclotherm units and the presence of structural features such as bedding planes, natural breaks, joints and fault gouges are also recorded.

The Hickleton UGBH No. 24 core can be divided into two main sections:

1) The Parkgate Rock

2) The Fenton Seam to Top of the Hole.

7.3.4.1 The Parkgate Rock

Figure 7.2 shows a geological section for the Parkgate Rock. Examination of the core revealed 27 m (88 ft) of massive medium grained sandstone containing numerous sedimentary bedding features

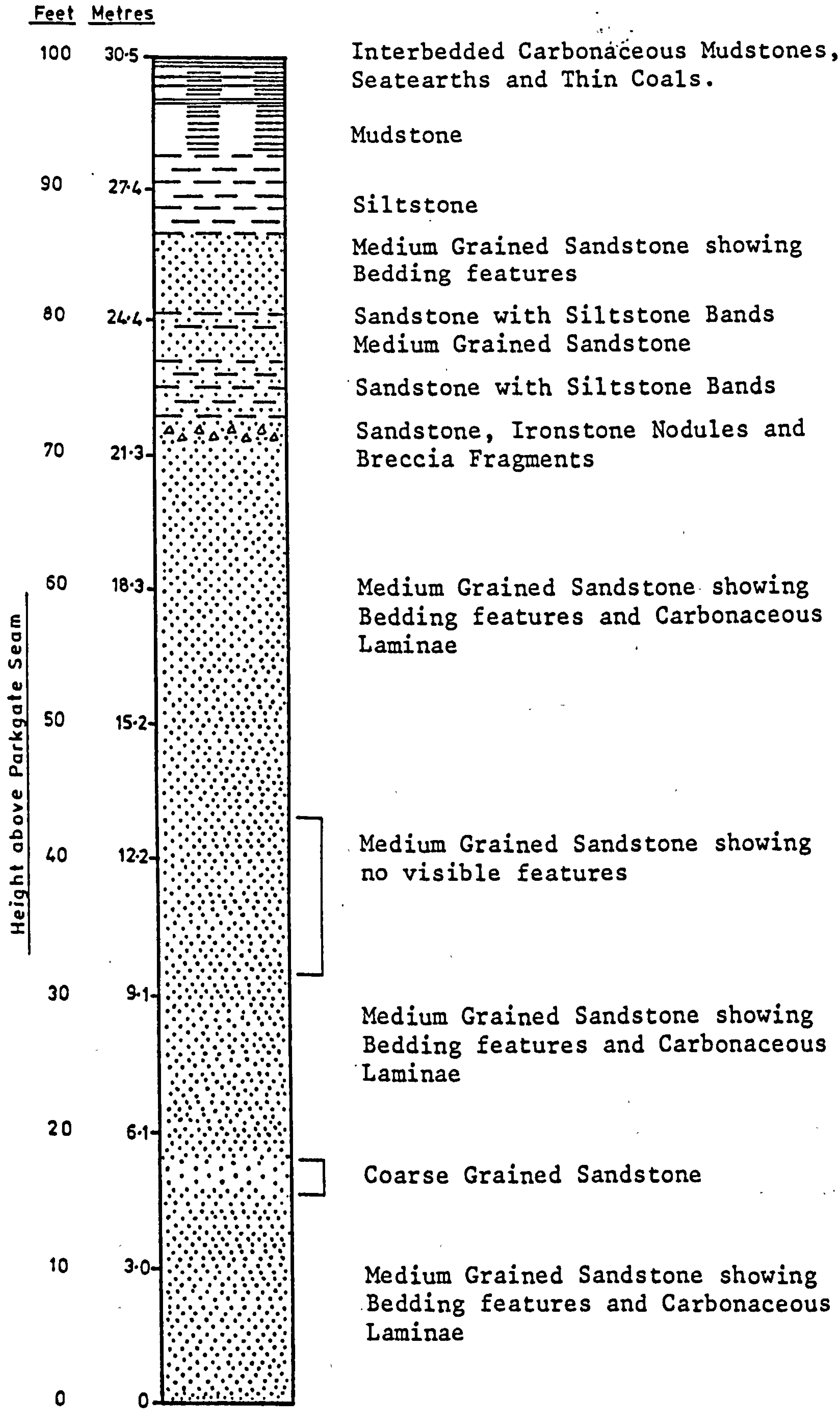


Figure 7.2 Geological Section of the Parkgate Rock, East Parkgate Area

(U.G.B.H. No. 24, Hickleton Colliery)

at most horizons. Only one band of truely massive sandstone was found (no discernable features on the core surface) between 9.7 m and 13.1 m. Several thin bands of coarse grained sandstone were also found, notably a 0.6 m band between 4.9 and 5.5 m. However, these rapidly graded back into a medium grained sandstone. Ironstone nodules were found at several horizons and an ironstone breccia, associated with a marked change in bedding, was encountered at 21.6 m.

Above 21.6 m, the strata became increasingly changeable with the appearance of siltstone laminae. An increasing proportion of siltstone was found until 27.0 m, when it became the major matrix constituent. At 27 m, the sequence graded rapidly upwards through siltstone and mudstone until a thin coal (0.23 m) was encountered at 29.4 m and identified as the Fenton Seam.

Finally, it is interesting to note that a dark material containing coal fragments was recorded by the drillers near the top of the P30 maingate boreholes. This suggests that the Fenton seam is situated over the P30 and P28 panels in its predicted location.

7.3.4.2 Fenton Seam to Top of the Hole

Figure 7.3 shows a complete geological section for UGBH No. 24 in 25's maingate scour. A thin coal and associated carbonaceous mudstones found in the vicinity of 30 m represents the Fenton seam, which is uneconomical throughout the area. Above the Fenton seam, the strata consists predominantly of mudstones with some ironstone nodules and an increasing proportion of siltstone, which becomes the dominant matrix component at 37 m. The sequence then rapidly grades through a series of mudstones and dirts to a further series of thin

Figure 7.3 Geological Section of U.G.B.H. No. 24, East Parkgate Area (Hickleton Colliery)

coals between 38 m and 40 m. These cannot be readily identified with any particular seam or seam leaf. Further mudstone sequences are predominant until 43 m, when a rapid transition to siltstone and sandstone occurs. However, a rapid return to mudstone also occurs with a further thin coal (0.10 - 0.15 m) at 44.8 m. This is immediately overlain by 5 m of sandstone containing well developed micaceous and carbonaceous bedding laminae, which grades into a 2 m siltstone band at 50 m. Further sandstone and siltstone horizons then occur until 58 m afterwhich siltstone becomes predominant until 69 m. Above 69 m a transition to mudstone occurs which rapidly grades into seatearth materials at 72 m, the top of the hole. These uppermost seatearths represent basal sequences of the Flockton Thick seam.

Finally, every sequence encountered above the Fenton seam was found to contain numerous well-developed sedimentary features in the form of microscale cross and current bedding (several centimetres). Some macro horizons were also found when traversing up the hole and these represent significant changes in the local depositional environment. Whisps of carbonaceous and micaceous material occurred along many of these bedding planes and when split revealed fossiliferous plant material.

7.4 Geotechnical Investigations

Core from UGBH No. 24 provided a unique opportunity to assess geotechnical properties of the Parkgate Rock and strata overlying the Fenton seam. Five geotechnical parameters were examined:

1) Natural Separation Planes
2) Point-Load Index
3) Tensile Strength
4) Uniaxial Compressive Strength
5) Shear Strength

Derivation of the theory and practice of each technique has already been adequately described by other authors, notably Jaeger and Cook (77), Vutukuri et al (78) and is therefore considered beyond the scope of this work. However, relevant authors will be referenced where appropriate.

7.4.1 Natural Separation Planes

All natural separation planes occurring within the retrieved core were recorded. Induced breaks caused by the removal of core from the barrel or by emplacement in the core boxes were ignored.

Over 350 separations were recorded along the core length of 72 m. However, in the friable units of mudstone, coal and seat-earth fragmentation was often too great to allow meaningful interpretation. Separations were classified into one of five types:

1) Horizontal - H
2) Sub-horizontal - SH
3) Inclined - IN
4) Irregular - IR
5) Wavey - W.

In general, the majority of separations were found to occur along pre-existing bedding planes and to be associated with either carbonaceous or micaceous laminae.

The borehole was divided into ten sections based on lithological divisions, Figure 7.3. For each section, the nature of the separations was established and three core classification indices derived, Franklin et al (79) and Deere (80):

1) Fracture Spacing Index (I_F)
2) Joint Frequency
3) Rock Quality Index (RQD).

Table 7.1 lists the ten borehole sections along with their respective intervals, lithological units and core quality indices.

7.4.1.1 Rock Quality Designation (RQD)

This classification index was developed by Deere (80) and is commonly used in North America and other English speaking countries. The RQD of a sequence is calculated by summing the total length of core pieces of length 0.1 m or greater and expressing the result as a percentage of the total unit length.

Divisions for the RQD classification are given in Table 7.2

RQD%	Rock Quality
0 - 25	Very poor
25 - 50	Poor
50 - 75	Fair
75 - 90	Good
90 - 100	Excellent

Table 7.2 Rock Quality Designation (after Deere (80))

Lithological Sequence	Height Range above Seam (m)	Sequence Thickness (m)	Natural Separation Planes						RQD Classification	Average Joint Frequency (I)	Average Fracture Spacing Index (I_F)
			H	SH	IN	IR	W	Total			
Sandstone (Parkgate Rock)	0.0 - 27.0	27.0	14	6	6	9	5	40	99% Excellent	1.48	0.67
Siltstone	27.0 - 28.4	1.4	3	0	1	4	0	8	99% Excellent	5.44	0.18
Mudstone	28.4 - 36.0	7.6	43	13	5	31	4	96	35% Poor	12.26	0.08
Siltstone	36.0 - 37.9	1.9	1	7	3	6	0	17	53% Fair	10.43	0.09
Mudstone	37.9 - 43.9	6.0	56	1	1	8	2	68	66% Fair	11.26	0.09
Sandstone	43.9 - 50.0	6.1	9	2	5	6	1	23	91% Excellent	3.80	0.26
Siltstone	50.0 - 52.4	2.4	1	0	7	1	1	10	84% Good	5.15	0.19
Sandstone	52.4 - 57.0	4.6	7	0	1	4	0	12	99% Excellent	2.38	0.42
Siltstone	57.0 - 70.0	13.0	19	3	13	27	7	69	93% Excellent	5.27	0.19
Mudstone	70.0 - 72.3	2.3	0	1	0	6	0	7	53% Fair	3.10	0.32
		72.3						350			

Table 7.1 Lithological Units and Rock Quality Indices, U.G.B.H. No. 24, East Parkgate Area

A comparison of calculated RQD values for the UGBH No. 24 sections reveals that the sandstone and siltstone sequences exhibit a good to excellent rock quality index. However, mudstone and coal sequences in the region of the Fenton seam only exhibit a poor to very poor quality index and this is associated with their friable nature.

7.4.2 Point-Load Index

Techniques, applications and limitations of point-load testing have already been adequately described and discussed by Bieniawski (81), Hassani (82) and Elkington (83).

A portable testing machine was used which has already been described in detail by Elkington (83). Testing was undertaken at frequent intervals along the core, with at least 10 cm separating respective test sites. The method of diametral testing was used and the point-load index calculated using Equation 7.1, Franklin et al (79),

$$I_s = P/D^2 \qquad \text{.......... } 7.1$$

where I_s - Point-Load Index

P - Applied Point Load

D - Core diameter (original distance between the points).

Since 72 mm core was tested, and specimen diameter is recorded as significantly influencing the final index value, Broch and Franklin (84) and Hassani et al (85), it was decided to use the size correction method proposed by Hassani et al (85) to derive equivalent index values for 50 mm core diameters and therefore standardise the results.

Table 7.3 lists geotechnical data for the ten borehole sections and in each case the Is_{50} values appear to fall within a relatively narrow band for each separate unit. Similarly, core sections with well developed bedding features show a marked decrease in index values.

7.4.3 Indirect Tensile Strength

Indirect tensile strength values for the core were obtained using the Brazilian Disc method. Values given in Table 7.3 are derived using Equation 7.2, Vutukuri et al (78),

$$Ts = \frac{2P}{\pi LD} \qquad \text{..........} \quad 7.2$$

where Ts - Tensile strength

P - Load

D - Diameter of Disc

L - Disc Thickness.

Data obtained from the separate lithological units again falls into relatively narrow bands. Difficulty was also experienced during testing, with weaker sequences which exhibited well developed bedding, since the specimens tended to spall around the edges rather than form a clean break. As a result, only a limited number of results were obtained from the weaker horizons. Similarly, the degree of bedding plane development exhibited by a specimen, effected the final strength value.

Lithological Sequence	Height Range above Seam (m)	Sequence Thickness	Point-Load Index Is_{50} MN/m^2 Range	Tensile Strength MN/m^2 Range	Uniaxial Compressive Strength MN/m^2 Range	Shear Strength MN/m^2	Internal Angle of Friction
Sandstone (Parkgate Rock)	0.0-27.0	27.0	2.1-3.5 Bedding 0.7-1.4	5.0-7.5	60 - 70	17.0	41^o
Siltstone	27.0-28.4	1.4	2.3-3.0	5.1-6.5	54 - 60	N/A	N/A
Mudstone	28.4-36.0	7.6	0.6-0.8	1.8-2.8	20 - 40	7.0	31^o
Siltstone	36.0-37.9	1.9	2.0-2.5	5.0-6.0	50 - 60	13.0	37^o
Mudstone	37.9-43.9	6.0	0.5-0.7	1.5-2.0	40 - 50	8.0	33^o
Sandstone	43.9-50.o	6.1	2.0-3.0	3.5-6.0	75 - 85	16.0	36^o
Siltstone	50.0-52.4	2.4	0.6-0.7	1.5-3.0	40 - 50	13.0	35^o
Sandstone	52.4-57.0	4.6	2.0-3.0 Bedding 0.7-1.5	2.5-4.0 Bedding 1.0-2.0	75 - 85	16.0	36^o
Siltstone	57.0-70.0	13.0	2.0-3.0 Bedding 0.7-1.6	3.0-7.0 Bedding 1.0-2.0	65 - 80	16.0	36^o
Mudstone	70.0-72.3	2.3	N/A	N/A	N/A	N/A	N/A

Young's Modulus - Parkgate Rock - 5700 - 6900 MN/m^2

Table 7.3 Geotechnical Data for U.G.B.H. No. 24, East Parkgate Area

7.4.4 Uniaxial Compressive Strength

Uniaxial compressive strength was determined using cylindrical specimens with a length-diameter ratio of 2:1 and calculated using Equation 7.3, Vutukuri et al (78),

$$\sigma_c = \frac{F}{A} \qquad \text{..........} \quad 7.3$$

where σ_c - Compressive Strength of Specimen

F - Applied Load

A - Cross-sectional Area of Specimen

Whenever possible, a full range of samples were obtained from a particular section, in order to ensure representative values. The results are given in Table 7.3 and show that values for each horizon fall within relatively narrow bands. Bedding features within the specimens were not found to significantly effect the final results, as they had done with the Point-Load index and Tensile strength values.

Young's modulus (Table 7.3) was calculated for representative samples of the Parkgate Rock and determined by means of the 50% value of uniaxial compressive strength, using strain measurements taken over the entire specimen length.

7.4.5 Shear Strength

The original core diameter of 72 mm was reduced to 38 mm to permit triaxial testing. This was achieved by re-coring selected specimens. Representative samples from the various lithological divisions were obtained and tested using a length-diameter ratio of

2:1. Confining pressures of up to 25 MN/m^2 were used on the majority of rock types. Results were analysed using the Mohr Criterion of Failure and Mohr envelopes constructed, Vutukuri et al (78). Figures 7.4 - 7.6 show Mohr diagrams for 3 of the lithological units and the results are summarised in Table 7.3.

7.4.6 Discussion of Geotechnical Results

Table 7.1 lists core quality indices and Table 7.3 geotechnical data for UGBH No. 24.

The Parkgate Rock (0 - 27 m) and strata above the Fenton seam (27 - 72 m) show no unexpected variation in strength characteristics. The presence of well developed bedding planes significantly affects the point-load index and tensile strength values of certain horizons. In Table 7.3, a point-load value of between 2.1 - 3.5 MN/m^2 can be confidently expected for samples of the Parkgate Rock showing poorly developed bedding features. However, when these features become well developed, a decrease in the point-load index occurs to between 0.7 - 1.4 MN/m^2, a reduction of over 50%. Similar variations in the magnitude of tensile strength also occur at several strata horizons. A reduction in rock strength associated with well developed bedding planes is less apparent in the compressive and shear strength values. No significant change in rock strength was detected across the band of massive sandstone between 9.7 m and 13.1 m.

Where no results exist for a given lithological horizon, the strata was either too weak and broken or insufficient in quantity to allow testing. Similarly, although over 300 tests were carried out on the core, certain limitations must be considered when

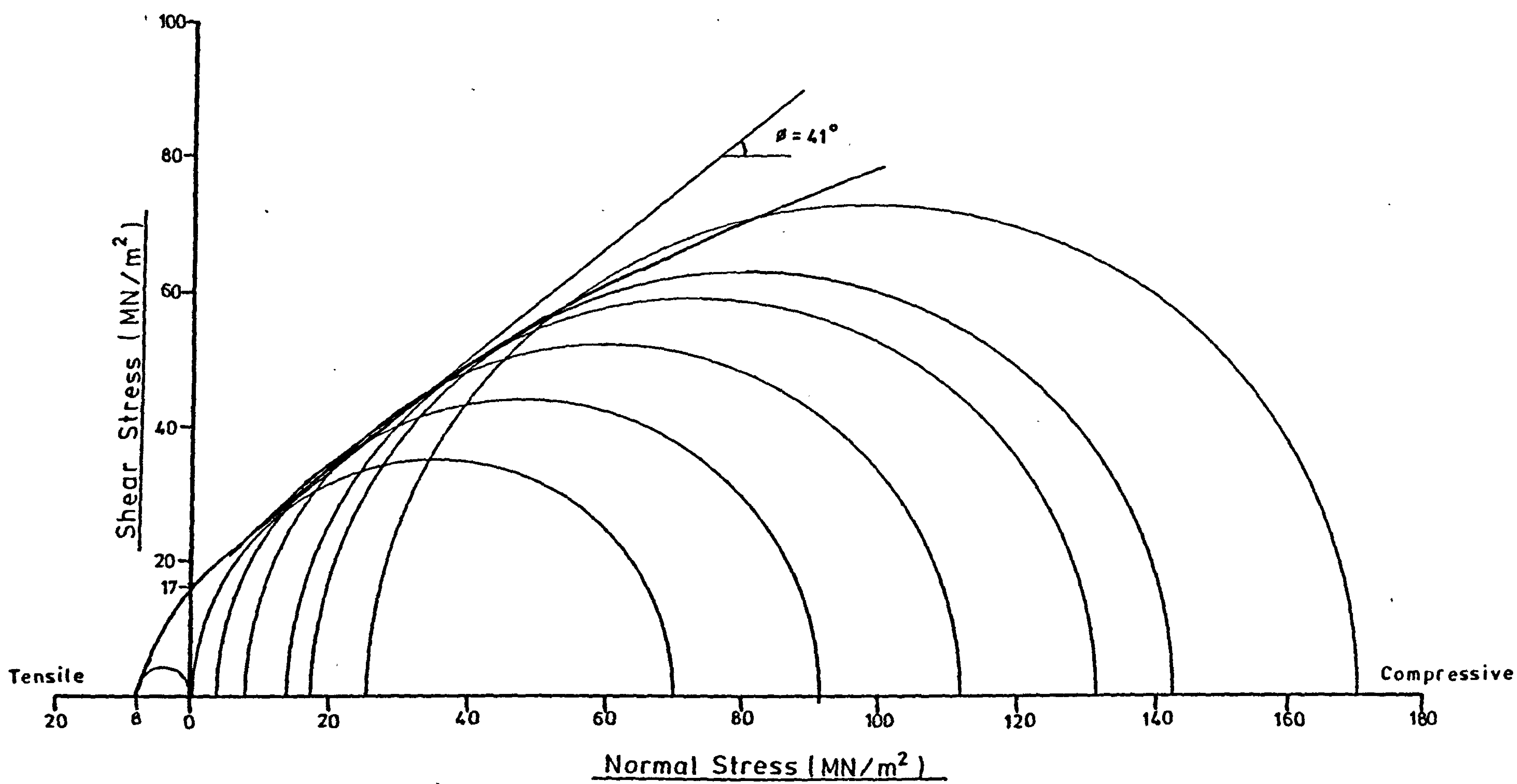


Figure 7.4 Mohr Diagram - Parkgate Rock (0.0 - 27.0 m), East Parkgate Area

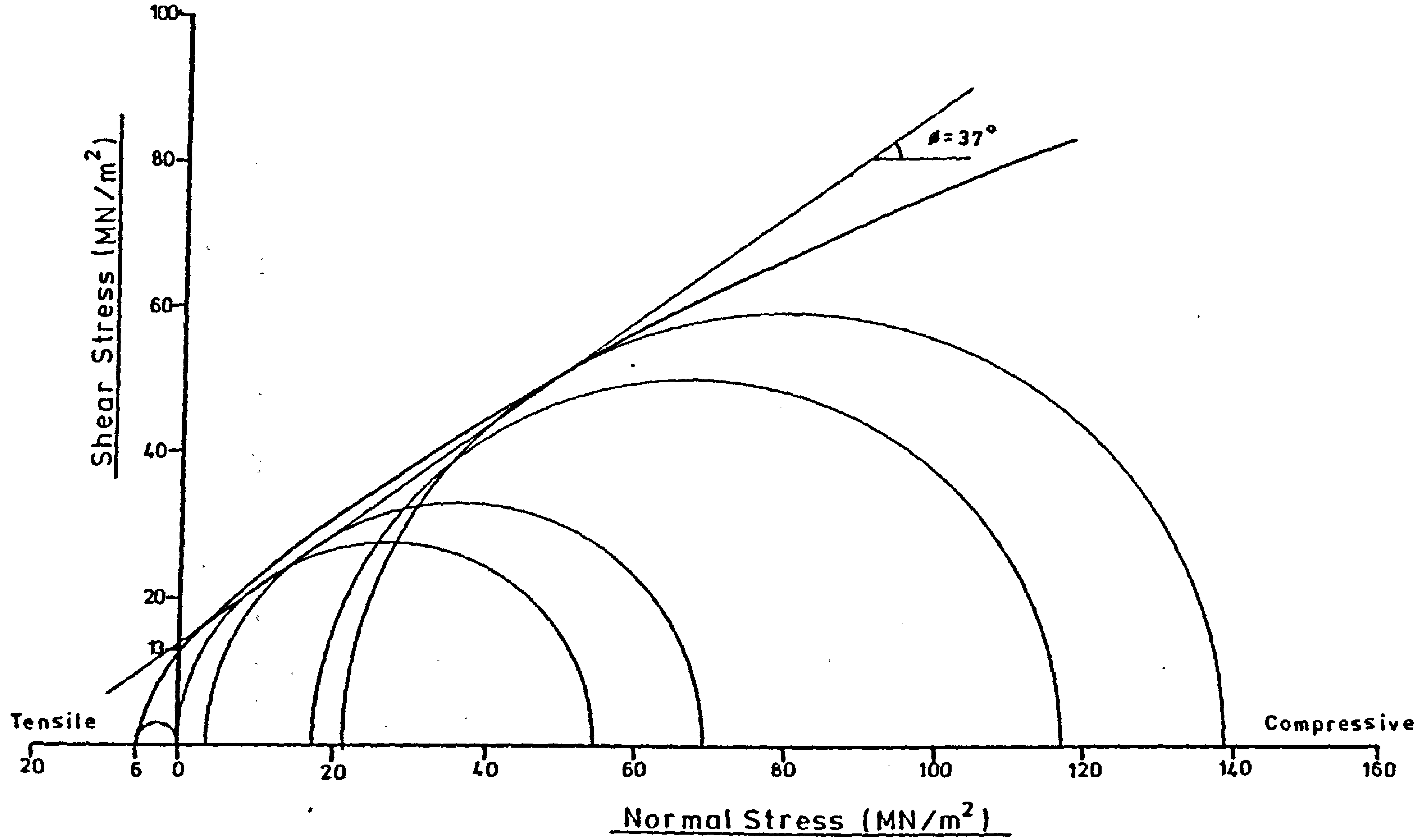


Figure 7.5 Mohr Diagram - Siltstone Horizon (36.0 - 37.9 m), East Parkgate Area

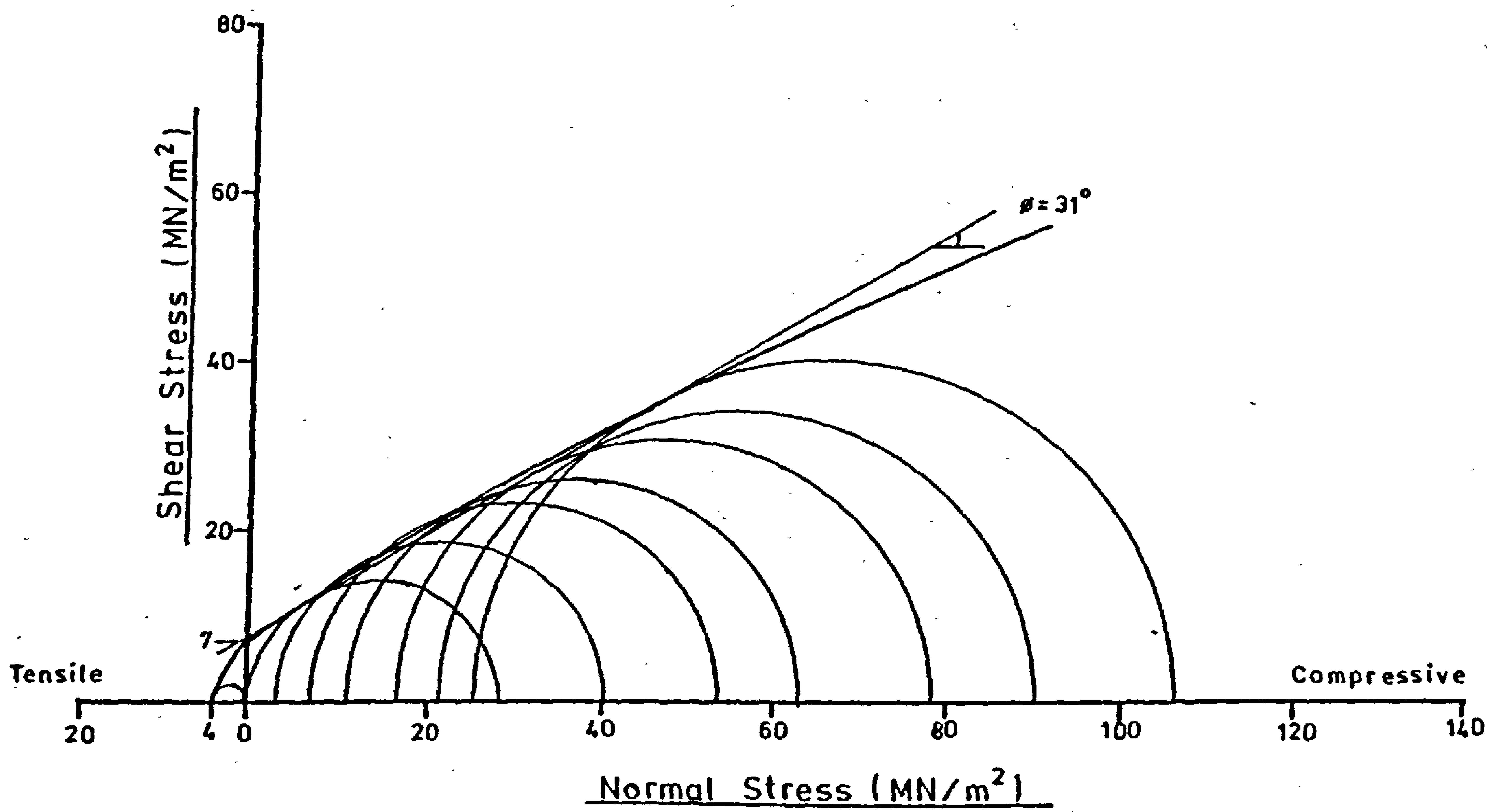


Figure 7.6 Mohr Diagram - Mudstone Horizon (28.4 - 30.6 m), East Parkgate Area

assessing data tabulated in Table 7.3. A borehole core only allows one specimen from one horizon to be tested in any given manner. Therefore when assessing any given rock strength parameter, it is essential to remember that between 50 and 100 samples are required from each rock type or strata horizon, in order to achieve representative results and smooth out statistically random values.

A comparison of the UGBH No. 24 results with general strength characteristics for Coal Measures strata, Tables 7.4 and 7.5, reveals that the Parkgate Rock only has parameters typical of a medium strength Coal Measures sandstone. It does not exhibit characteristics typical of a strong, massive sandstone sequence, as was initially proposed in Chapter 6. The medium strength characteristics are due to the presence of numerous sedimentary bedding features at all levels. Similarly, Coal Measures strata can also exhibit anisotropic properties in the form of lithological variations (grain size and orientation, packing density and matrix constituents) as well as structural variations associated with sedimentation processes.

Hassani et al (86) states that anisotropy effects can reduce the uniaxial compressive strength of siltstones by up to 32% and 43% respectively, for dry and saturated samples. Similar values for sandstone are 23% and 20%. Uniaxial compressive strength values given in Table 7.3 may well accommodate the percentage change in strength proposed by Hassani et al (86) for dry specimens exhibiting X anisotropic affects. Anisotropy can also effect the tensile and shear strength of strata in both the dry and saturated state.

Rock Type	No. of Tests	Uniaxial comp. Strength		Tensile Strength		Young's Modulus	
		Dry MN/m^2 $\sigma_c \pm$ S.Dev.	Saturated MN/m^2 $\sigma_c \pm$ S. Dev.	Dry MN/m^2 $\sigma_t \pm$ S. Dev.	Saturated MN/m^2 $\sigma_t \pm$ S. Dev.	Dry MN/m^2 $E \pm$ S. Dev.	Saturated MN/m^2 $E \pm$ S. Dev.
Mudstone	90	45.0 ± 5.0	5.0 ± 2.0	4.5 ± 0.5	1.5 ± 0.2	6075 ± 3000	N.A.
Seatearth	40	29.0 ± 8.0	1.25 ± 0.1	3.5 ± 1.3	<0.5	4440 ± 600	52.5 ± 13.5
Lam. siltstone	190	64.0 ± 13.0	45.0 ± 9.0	8.0 ± 1.0	4.0 ± 0.9	17000 ± 2700	3300 ± 400
Mass. siltstone	60	77.0 ± 17.0	65.0 ± 12.0	7.5 ± 1.2	6.5 ± 1.0	8600 ± 3400	6000 ± 1500
Fine gr. sandstone	250	85.5 ± 9.0	56.0 ± 3.0	6.0 ± 1.0	5.0 ± 0.3	8000 ± 400	4000 ± 900
Med. gr. sandstone	160	68.0 ± 14.0	30.0 ± 15.0	5.0 ± 2.5	4.0 ± 2.0	6200 ± 1400	3600 ± 500
Coarse gr. sandstone	120	34.0 ± 3.5	23.0 ± 4.0	1.5 ± 0.1	0.65 ± 0.04	5700 ± 1500	3100 ± 650

Table 7.4 General Strength Characteristics of Coal Measures Rocks from Opencast Sites
(after Hassani et al (86))

Rock Type	Test Condition	Uni. Comp. Strength σ_c		Tan β σ	Angle of internal friction ϕ		Cohesive Strength So	
		meas MN/m^2	calc MN/m^2		meas°	calc°	meas MN/m^2	calc MN/m^2
Fine gr. sandst.	Dry	85.0	95.0	4.74	55	40	23.0	22.0
Fine gr. sandst.	Sat.	56.0	83.0	3.70	40	35	17.0	21.5
Med. gr. sandst.	Dry	68.0	80.5	4.40	50	39	15.0	19.0
Med. gr. sandst.	Sat.	50.0	65.0	3.60	45	34	13.5	17.0
Coarse gr. sandst.	Dry	34.0	43.0	2.67	35	35	14.0	14.5
Lam. siltst.	Dry	64.0	94.5	3.88	50	36	20.0	24.0
Lam. siltst.	Sat.	45.0	53.3	2.89	39	29	16.5	16.0
Mass siltst.	Dry	74.0	73.5	3.41	50	33	16.5	21.5
Strong mudst.	Dry	48.0	89.6	4.4	50	39	14.0	21.0
Weak mudst.	Dry	38.0	49.1	3.14	40	31	7.0	14.0
Seatearth	Dry	29.0	42.8	2.67	30	27	8.0	13.0

Table 7.5 Triaxial Strength Data for Coal Measures Rocks
(after Hassani et al (86))

7.5 Geotechnical Aspects of Roadway Deformation Surveys

One of the simplest strata control investigation methods available is the roadway deformation survey. This is conducted along a gateroad either infront of or behind the faceline, depending on the type of extraction method used. However, in order to achieve maximum interpretation of the results, it is essential to consider geological and geotechnical aspects of the surrounding strata.

7.5.1 Roadway Deformation Survey - P30's Tailgate

A roadway deformation survey was conducted along P30's tailgate from Site No. 1 (Permeability test, Chapter 5) to just behind the faceline. Readings were taken at 5th arch intervals using a telescopic measuring rod. Initially chalk and later paint was used to mark the relevant arches and measuring points, although this was only achieved on one side, due to the conveyor position within the roadway. Fixed datum measuring stations were not installed. Horizontal measurements were taken from arch side to side, keeping the rod in a horizontal position. Vertical measurements were taken from the arch to floor, with any loose material directly beneath being scraped away. Measurements from the arch crown could not be taken due to the conveyor position.

Figure 7.7 shows total vertical closure and Figure 7.8 total horizontal closure recorded during the survey period. Readings were taken over a 90 m section of roadway for a total face advance of 25 m. However, only four sets of readings were obtained during this period. About 0.35 m of vertical and 0.60 m of horizontal

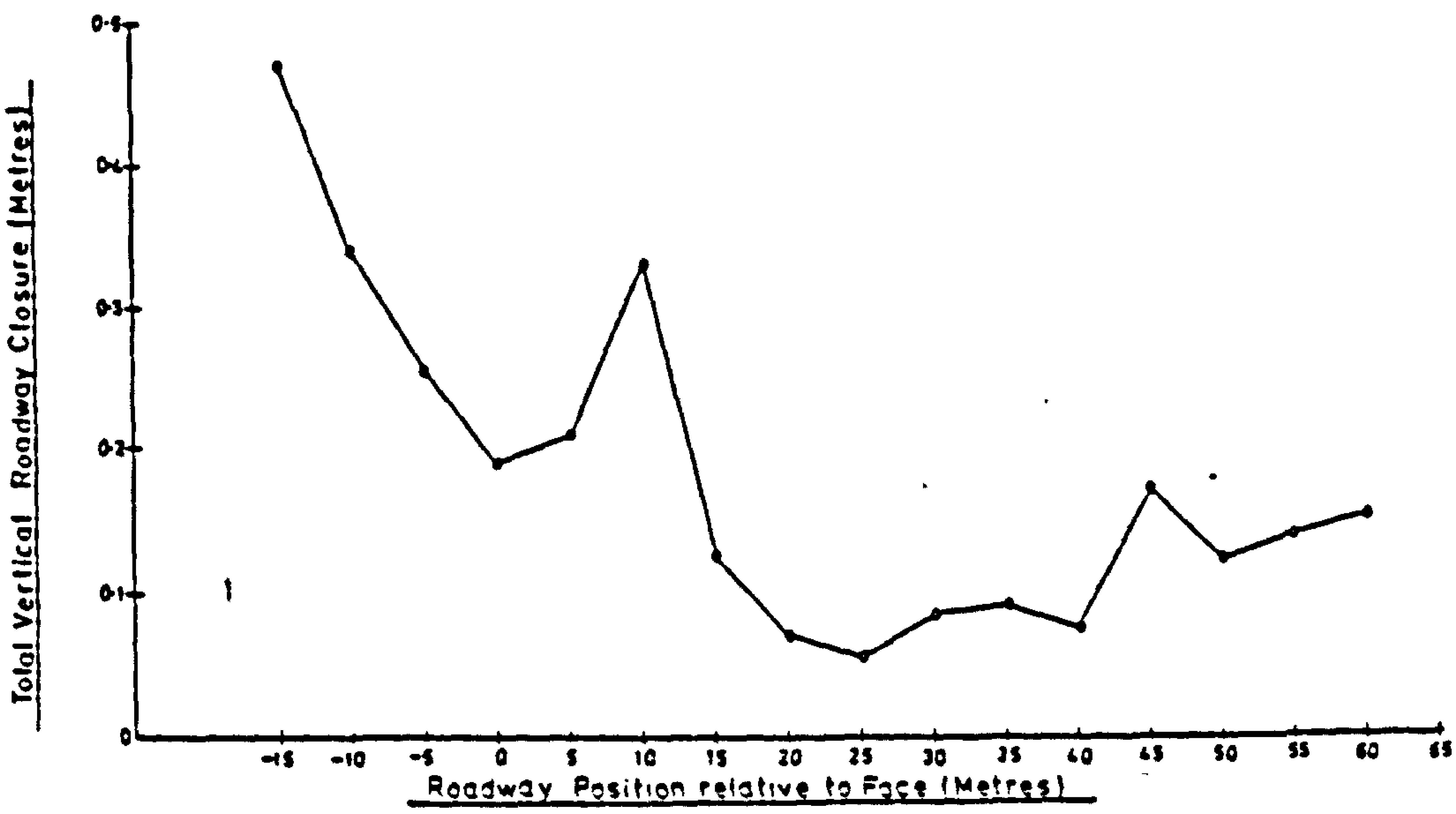


Figure 7.7 Total Vertical Closure, P30's Tailgate (Roadway Deformation Survey)

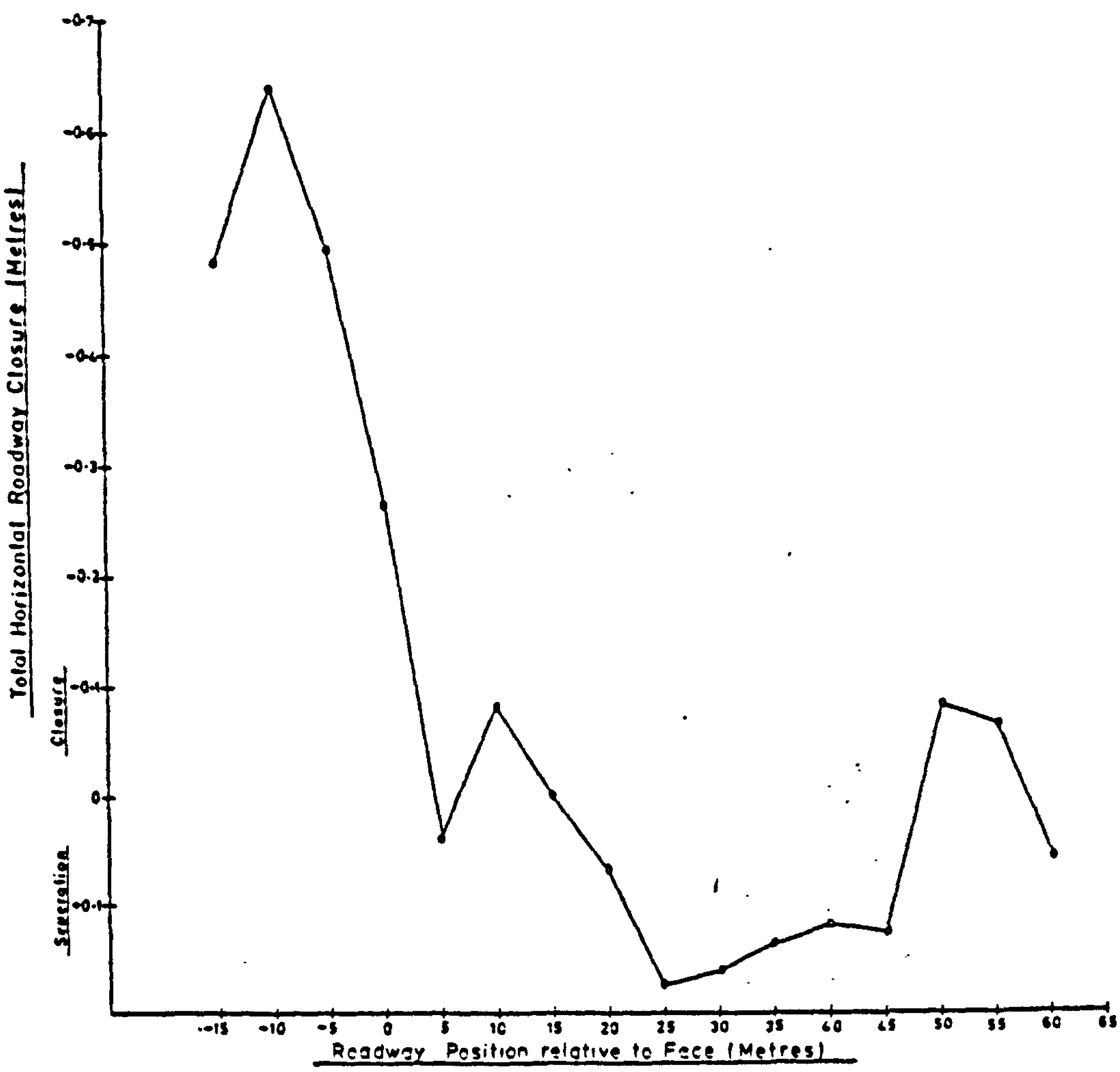


Figure 7.8 Total Horizontal Closure, P30's Tailgate (Roadway Deformation Survey)

closure was recorded in the roadway and most of this was found to occur in the immediate face vicinity.

In addition to examining total convergence results, the rate of vertical closure experienced by each station between adjacent readings was also calculated, Figure 7.9, Jukes (87). Although some anomalous points do exist, an expected trend does appear, where the greatest rate of convergence is found immediately in front of and behind the faceline.

7.5.2 P30's Tailgate - Geotechnical Considerations

Ideally, data obtained from a roadway deformation survey should be compared with detailed geological and geotechnical assessments of the surrounding strata. If a roadway cuts through inclined geological sequences, a variety of rock types will be supported. Deformation associated with each rock type can therefore be monitored and compared. However, in the East Parkgate area, the seam is relatively flat and the workings therefore only intersect a narrow band of Parkgate Rock.

Historically, P30's tailgate has already been used once as P29's maingate and strata surrounding the roadway will already have been subject to de-stressing and yield zone formation. Additional problems which would be encountered during a survey would be the conveyor position and the quantity of dust and crystallised salts deposited on the roadway sides. Ideally, a survey should be conducted during driveage operations and not at some later period during its lifetime.

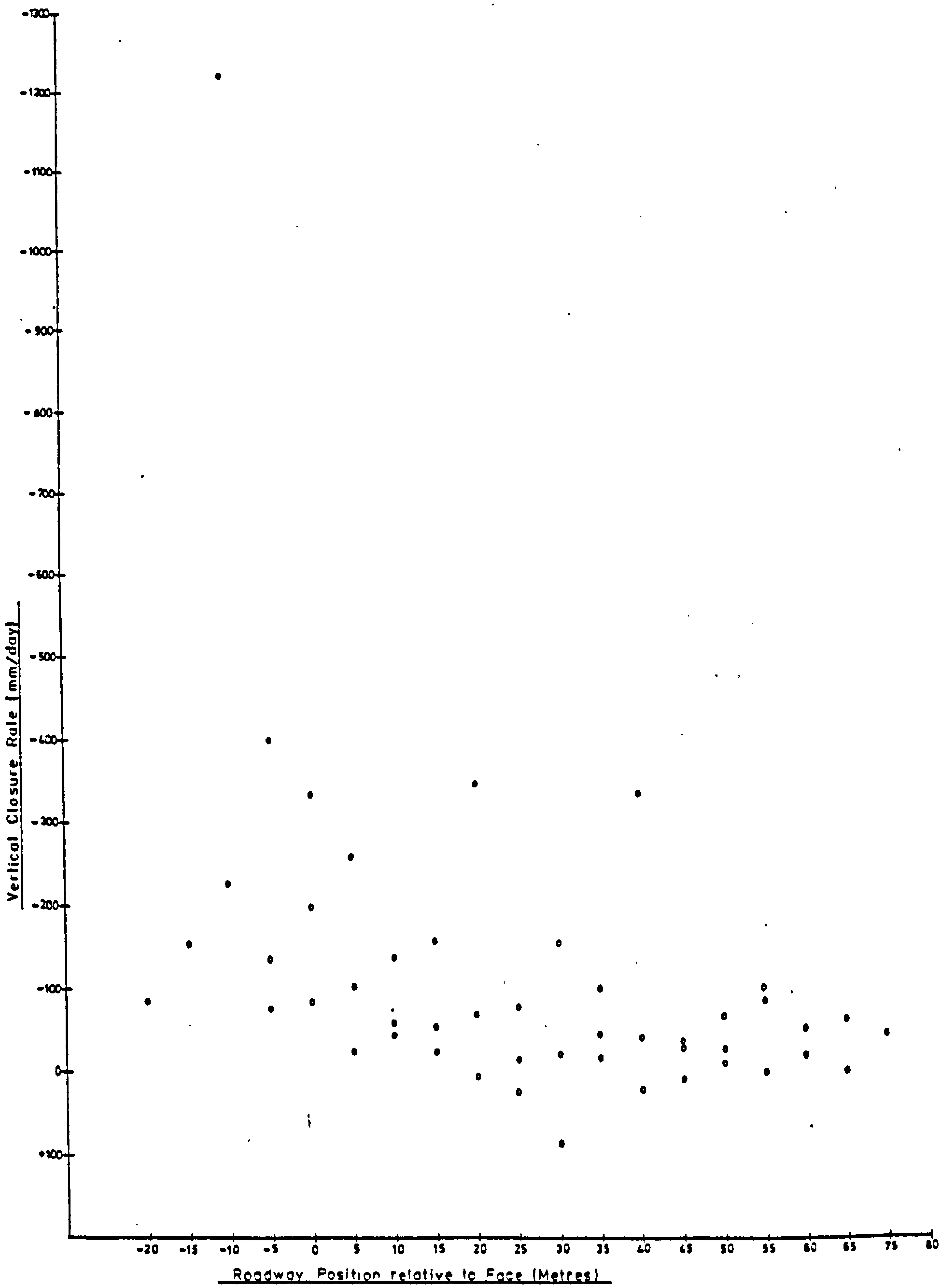


Figure 7.9 Rate of Vertical Closure, P30's Tailgate
(Roadway Deformation Survey)

No faulting was encountered during the driving of P30's tailgate, (75). If support loading had occurred away from the face vicinity, it might indicate movement along faults or significant planes of weakness, such as joint networks, Singh et al (88). However, away from the faceline, the roadway throughout its length was in excellent condition, which indicated that no significant discontinuity systems existed within the surrounding strata. Similarly, mapping natural discontinuity systems in roadways, especially if they are formed by drill and blast methods, is not always easy because of the difficulty in distinguishing between natural and induced networks.

Finally, Figures 7.7 and 7.9 show that vertical strata disruption in the roadway is beginning to occur between 10 m - 15 m in front of the faceline. This agrees with values derived from alternative sources, Chapter 6, Sections 6.5.5.4 and 6.8.

7.6 Stress Determination in Potential Rockburst Areas

At the beginning of the Seventies, the West German mining industry recognised that an increased risk of rockbursts (weight bumps) was likely to occur with increased working depths and irregular extraction due to selective mining. This led in 1974 to the formation of a special Rockburst Prevention Development Committee and the establishing of a Rockburst Prevention Centre. In 1978, the first symposium on rockburst prevention was given by the Rockburst Prevention Development Committee in Essen, (63).

In Britain, it is currently recognised that future workings in many NCB mines will be going deeper into seams overlain by thick sandstone sequences and therefore subject to potential bump conditions.

Experiences at the Barnborough and Hickleton collieries (Chapter 6) in the form of weight bumps has brought about renewed interest and concern. It has therefore been proposed that all workings with historical, current and future workings in areas of potential bump conditions should be examined and compared with continental conditions.

During investigations on the P30 panel at Hickleton Colliery, a copy of the North Rhine Westphalian Rockburst Guidelines (89) was obtained, which describes a practical method of assessing potential bump conditions in a working area. The method involves in-seam drilling techniques and was closely examined with a view to undertaking field trials in the Parkgate seam. However, the resulting change in policy concerning the East Parkgate area, indefinitely postponed this proposition.

7.6.1 In-seam Drilling Operations

Full details regarding potential weight bump conditions and assessment techniques can be found in the North Rhine Westphalia Rockburst Guidelines (89). However, rather than just refer to this reference, it is proposed to highlight several features of the technique which are not readily discernable from the guideline text.

Test holes are usually between 42 mm and 50 mm in diameter, sited in the middle of the seam and be sufficient in number to cover the entire area of potential hazard. The hole is horizontal and at right angles to the face, with a length calculated using Equation 7.4,

$$L = 3M + A \qquad \text{..........} \quad 7.4$$

where L - Length of Hole (m)

M - Total Seam Thickness (m)

A - Face or Drivage Advance until next test boring (m).

For static faces or drivages, a hole length in excess of 4 M is normally used.

Non-coring bits are used, with the cuttings being transported from the hole by means of the drill rods. Air flushing can be used for de-stressing or relief drilling, but is not normally used for test drilling. Cuttings are collected in a graduated bucket, where the volume is estimated to within 0.5 ℓ. A short length of stand-pipe can be inserted into the neck of the hole, but this is usually only done for special conditions, such as steep seams and does not effect the volume of cuttings retrieved, Braüner (90)

7.6.2 Evaluation of Drilling Results

The existance of weight bump conditions is confirmed if a test or subsequent borehole yields any of the following results from a 4 M length under static, or 3M + A length under dynamic conditions, North Rhine Westphalia Rockburst Guidelines (89):

1) If the quantity of cuttings yielded per metre of hole drilled exceeds 6 ℓ for 42 mm holes, 7 ℓ for 46 mm holes and 8 ℓ for 50 mm holes.

2) The drill rods undergo sticking due to the strata pressure.

3) Loud bumps or noises are heard, which are caused by the release of energy resulting from the drilling operations.

Gottig (91) gives results obtained from de-stressing rather than test boreholes, and these provide useful guidelines on what could be expected, if field trials were undertaken in this country:

'A total of 27 de-stressing holes were drilled in roadways, with an average length of 19.4 m and a range between 13 and 43 m. The volume of cuttings derived per metre of hole averaged 2.0 m^3, although figures for individual holes varied from 0.34 to 22.86 m^3. Average drilling time per hole was 185 minutes, with a maximum time of 800 minutes.

Along the faceline, 21 de-stressing holes between 17 and 34 m long were drilled, which had an average length of 26.5 m. The volume of cuttingsderived per hole averaged 8.8 m^3 and ranged from 0.6 m^3 to 20.8 m^3. Similarly, if this is related to 1 metre lengths of a drilled hole, an average volume of 0.3 m^3 is obtained, which is effectively 21 times the hole volume. Drilling time per hole averaged 365 minutes.'

Finally, it is considered useful to try and determine the success rate of the technique for identifying potential bump conditions. Values given by Hess (92), suggest a low effectiveness when used indiscriminately for the detection of high stress concentrations:

'A survey of test boreholes carried out by Bergbau A.G. Westphalia in June 1977 and March 1978 revealed that potential risk areas can be too large. Of the 5645 and 7961 boreholes drilled, only 34 and 94 holes respectively yielded positive results on potential bump conditions. This is equivalent to 0.6 and 1.2% of the total holes drilled.'

Use of the technique and evaluation of the results therefore requires caution. Experience is essential for its use and so too is the prior identification of areas with a high probability of bump conditions.

7.6.3 Alternative Detection and De-stressing Techniques

Hess (92) mentions that work is currently progressing on three alternative methods for identifying potential rockburst conditions:

1) The detection of seismic activity and the propagation of seismic waves both in the face vicinity and in solid coal.

2) The development of multiple pressure probes and extensometers, to detect fluctuating stress patterns with respect to the location of activity.

3) Micro-acoustic activity in the high frequency range, involving the use of geophone detection equipment for the monitoring of close proximity fracture noises and the establishment of a continuous monitoring system.

In addition to the detection of high stress concentrations, in-seam drilling techniques can also be used for the alleviation and elimination of such phenomena. However, two further de-stressing techniques are also available.

1) De-stress Shot-firing - where test holes which yield critical amounts of cuttings are charged with explosives and fired. This causes stress relief within the coal seam and surrounding strata. It also has the added advantage of being less time and labour intensive than the drilling technique and more easily integrated into the operational working cycle.

2) De-stress Water Infusion - where water is added to the strata under high pressure, in order to achieve stress relief. It also has the added advantage of controlling dust formation.

7.7 Conclusion

The successful completion of any strata control investigation requires a comprehensive understanding of geological and geotechnical properties of the surrounding strata. This can best be achieved by means of a cored borehole, which will not only provide detailed lithological data but also samples for geotechnical assessment. Similarly, the effect of structural discontinuities and anisotropy within the strata should not be underestimated, when considering its behavioural characteristics under mining induced stresses.

Where a cored hole is available, preliminary geotechnical parameters can be derived on site, using simple mechanical logging techniques, such as the point-load index. However, this method is subject to many limitations and can never replace comprehensive laboratory based investigations. Nonetheless, the method is available and can or should be used, provided the limitations are recognised.

Finally, in-seam stress determination using drilling techniques has for many years been used successfully in West German mining conditions. Its potential application to British conditions has clearly not yet been recognised. A series of field trials should therefore be initiated as soon as possible, in order to assess the technique and establish its respectability as a method for monitoring high in-seam stress concentrations.

CHAPTER 8

SUBSIDENCE EFFECTS IN THE UNDERSEA COALFIELD WORKINGS OF NORTH-EAST ENGLAND

CHAPTER 8

SUBSIDENCE EFFECTS IN THE UNDERSEA COALFIELD WORKINGS OF NORTH-EAST ENGLAND

8.1 Introduction

Historical depletion of inland reserves within the North-East Coalfield has necessitated the working of vast reserves which extend eastwards beneath the North Sea. Nine coastal collieries currently produce about 60% of the area output from these reserves. However, as the workings continue to extend eastwards beneath the sea, the problems and costs associated with coal production increase,Watson (93).

A major problem associated with undersea or for that matter mining under any large surface water body or sub-surface aquifer, is the prevention of water entering the workings, Sheard and Hurst (94) and Plumptre (95). In the North-East Coalfield there has always been a general and healthy respect against the tapping and subsequent breakthrough of not only the sea, but also an 'underground sea' contained within the overlying Permian strata, Saul (96).

As a result of this and experiences in other parts of the country, individual collieries and areas have always followed codes of practice, which it was hoped would prevent a major water occurrence within the workings. However, during the mid-1960's, it was decided that the various codes of practice should be rationalised and in 1968 the National Coal Board issued an instruction PI/1968/8 - 'Working Under the Sea' (2).

8.2 Research Objectives and Procedure

The NCB 1968 guidelines (2) state that for extraction by longwall methods:

'the cover between the top of the worked seam and seabed shall not be less than 105 m, of which 60 m must be Carboniferous strata. The tensile strain at seabed, shall not exceed 10 mm/m, for both first and successive seam working.'
Similar criteria are also laid down for extraction by room and pillar methods (2).

However, certain limitations of the 1968 guidelines, with respect to the maximum induced tensile strain of 10 mm/m at seabed, have recently been depicted in several areas of undersea working. Economic areas of coal in both existing and potentially viable seams have been effectively sterilised unless an alternative extraction method is used. Consequently, discussions have arisen concerning the feasibility of relaxing the original 10 mm/m tensile strain imposed on undersea workings. No change is proposed for the minimum allowable thickness of cover between the extracted seam and seabed, as set out in the guidelines (2).

Work by the National Coal Board has concluded that the occurrence of water on longwall faces cannot be readily attributed to normal mining parameters such as induced tensile strain at either seabed or the base of Permian, the width-depth ratio of the extraction or any other such factor. Garritty (97) has drawn together much of this data and analysed it using standard statistical techniques. A similar conclusion was reached which neither confirmed or rejected the guideline criteria that more than 10 mm/m of tensile strain will result in water, while less than 10 mm/m will prevent water from occurring within the workings. Similarly, extensive work by Garritty (97) into the effect of strata lithology on water occurrences, has concluded that a high risk exists in any area where the overlying strata contains less than 35% of sandstone. This is a surprising conclusion, since sandstone sequences are normally assumed to act as potential aquifer horizons. To explain this phenomena, Garritty (97) suggests that when less sandstone is present, the overlying strata becomes less competent with the result that the induced fracture zone is propagated to a greater height. However, in the authors opinion, Chapters 6 and 7, the presence of sandstone horizons does not necessarily reinforce the overlying strata against the effects of mining induced stresses. Similarly, an increased proportion of mudstones and siltstones, both of which can exhibit excellent flow inhibitor properties (Chapter 1, Section 1.4 and Chapter 4, Section 4.6), should reduce the risk probability of water occurring within the workings.

Since normal statistical techniques appear to produce either misleading or inconclusive results, it was proposed that an investigation should be initiated to evaluate existing information. The occurrence of water with respect to the geological and hydro-geological environment as well as the tensile strain predicted at two significant horizons, namely the base of Permian and seabed, should be re-examined. Once the controlling factor(s) have been established, further proposals can then be submitted for a revision of the existing guidelines and the elimination of recently depicted anomalous areas.

Visits were subsequently made to four collieries with extensive undersea workings in the Durham coastal coalfield, namely: Blackhall, Horden, Westoe and Dawdon. Relevant information was then collected concerning mine lay-out, geology and the nature of water occurrences experienced. Choice of the four collieries was based on the criteria, that the first three had all experienced major water occurrences, while the fourth had remained 'dry'. Predicted tensile strain at the base of Permian and seabed was calculated using methods given in the Subsidence Engineers Handbook (3) and Chapter 3.

8.3 Geology and Hydrogeology of the North-East Coalfield

8.3.1 Geology

The Coal Measures of the North-East Coalfield constitute an approximate triangle with an area of 2000 km^2, Figure 8.1. The strata has a total thickness of some 725 m, which can be divided into two main areas:

1) an exposed western area
2) a concealed eastern area.

In the eastern area, the Coal Measures are concealed beneath a transgressive cover of Permian strata, the dominant member of which

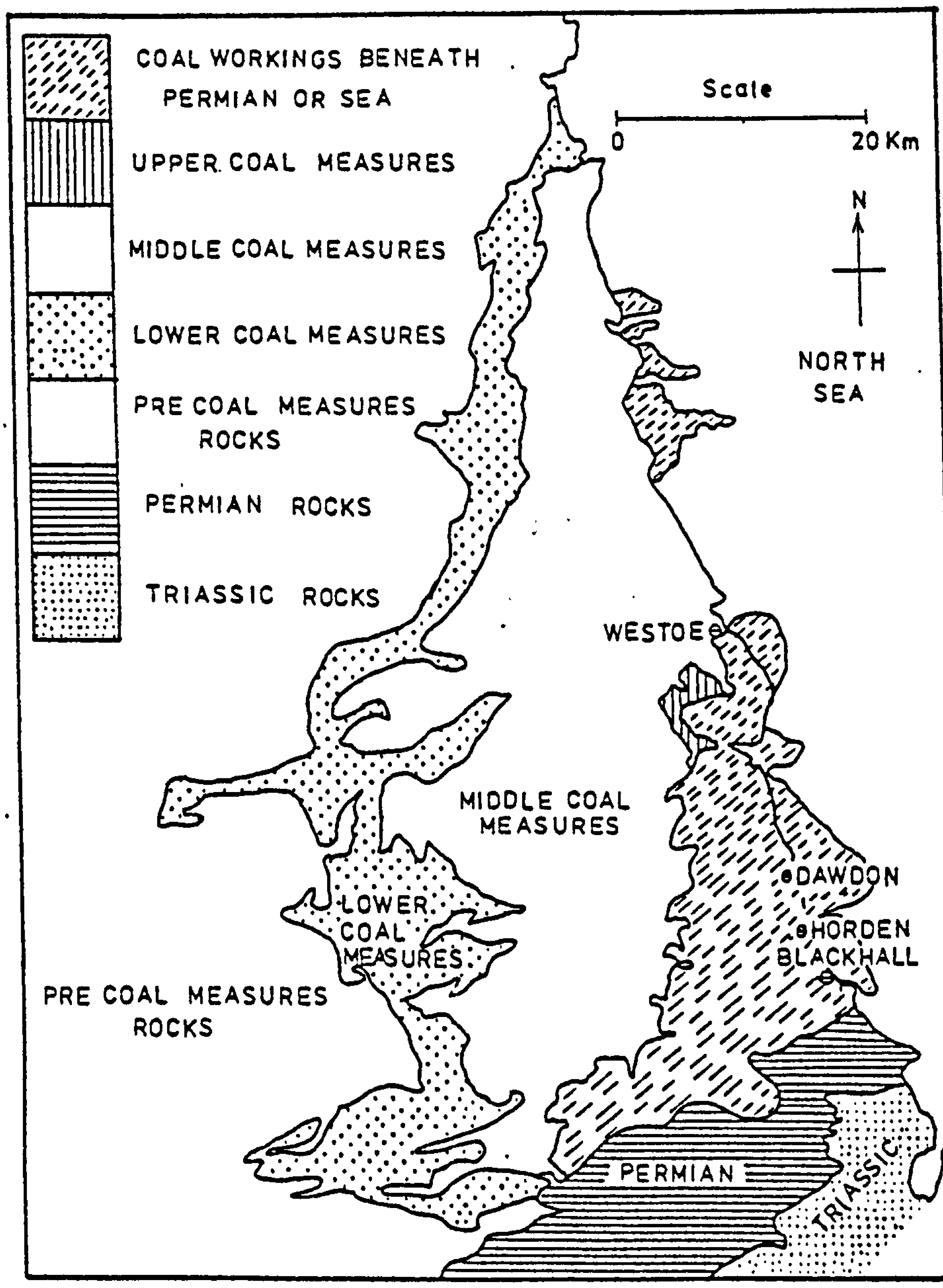


Figure 8.1 Geology of the North-East Coalfield

is Magnesium Limestone. The Permian outcrops along the Durham coast from the River Tyne to West Hartlepool, where it is faulted against the Trias. The main Permian outcrop does not extend north of the River Tyne into Northumberland, Trueman et al (16).

8.3.1.1 Structural Considerations

Structurally, the coalfield forms a classical irregular basin with a North-West to South-East trend. The strata dips gently eastwards at between 5 and 10^{o} beneath the North Sea, where it has been proved to extend over 10 km by offshore boreholes. By whom? In the Durham Coalfield, the principal structural feature is a pitching syncline with a South-East trending axis.

The coalfield is traversed by two main fault systems:

1) East-North-East to West-South-West

2) North-West to South-East.

Several minor fault systems are also found, which trend either North to South or East to West.

Two systems of intrusive igneous dykes are also found traversing the coalfield. The primary series, trend East-North-East to West-South-West and are thought to have been intruded during the Armorican Orogeny, prior to the erosion of the Upper Coal Measures and subsequent to the deposition of the Permian, Smith et al (98). A secondary series also exist, which trend East-South-East to West-North-West and these are thought to be Tertiary in age, Trueman et al (16). Mineral veins of barytes, galena and witherite are also found associated with the Coal Measures strata.

A detailed lithological and structural description of the Coal Measures and Permian strata existing across the area is considered beyond the scope of this work and has already been adequately described by Smith et al (98) and Trueman et al (16).

8.3.1.2 Basal Permian Sequences

At the junction of the Coal Measures and the Permian Limestone lies a band of sands, sandstones and breccias known as the Basal Permian or Yellow sands. Their importance in relation to the occurrence of water within colliery workings has been postulated by several workers, notably Saul (96).

The sands are aeolian in origin and were deposited across the eroded surface of the Carboniferous land mass. The composition and thickness varies markedly across the Durham area and can be attributed to their accumulation in hollows on the land mass surface or as dune formations, Smith et al (98). Similar variations in the degree of compaction are due to changes in environmental chemistry at the time of deposition. The well cemented formations contain a high proportion of calcite in the matrix material.

8.3.2 Hydrogeology

The natural groundwater regime of the North-East Coalfield is extremely complex and certainly far from being comprehensively understood. However, it is proposed to briefly outline some of the assumptions normally made concerning the three main strata formations. Similarly, two structural systems are mentioned which can also allow the passage of water to mine workings.

8.3.2.1 Permian Limestone Formations

The Permian Limestone formations outcrop at seabed and are therefore assumed to be saturated throughout their entire thickness. They are thought to possess fissure permeability resulting from well developed solution channels and cavities, because of the characteristics exhibited by limestone formations elsewhere in the country. However, this assumption may be erroneous, since local variations in lithology can produce impermeable and/or semi-permeable horizons, which restrict both the vertical and horizontal flow of water across the area.

8.3.2.2 Basal Permian Sands

The Basal Permian sands exhibit a well developed intergranular permeability throughout the Durham area. Variations in thickness and lithology probably affect localised flow regimes, particularly in areas with a high degree of consolidation. During some shaft-sinking operations, the formation has been encountered as 'quicksand', which has led to additional technical difficulties. In general, they provide an excellent aquifer medium for both the vertical and horizontal flow of water across the area.

8.3.2.3 The Coal Measures

Coal Measures strata exhibiting typical cyclothermal variations is normally assumed to be impermeable, unless well developed sandstone sequences or structural discontinuity networks are present. The mudstone and siltstone horizons can make excellent aquitards and aquicludes, which restrict both the vertical and horizontal flow of water. Sandstone horizons can act as potential aquifers, providing they exhibit sufficient intergranular permeability, although the yield is likely to be restricted. Similarly, local variations in strata lithology, Chapter 4, can be expected to affect the flow either from or across a given sequence.

If a major aquifer, such as the Permian, is displaced against Coal Measures sandstones, recharge can result in the formation of secondary aquifer systems. Little recharge is thought to occur to the concealed Coal Measures from their land outcrop, although localised recharge could occur when they outcrop against the Permian or seabed.

8.3.2.4 Fault Zones

Where fault zones penetrate Carboniferous and Permian strata, water can occur at a mining horizon from two sources:

1) directly along the fault plane

2) the displacement of existing aquifers against potential aquifer horizons. Recharge occurs across the fault plane over geological time, which forms a secondary aquifer system in close proximity to the potential mining horizon.

8.3.2.5 Dyke Systems

Dyke systems have been encountered in the workings of several Durham coastal collieries, notably Westoe. Although igneous in nature, they are found to contain cavities which are interconnected by a well developed fissure network. They can make excellent aquifers when in hydraulic continuity with the overlying Permian.

The effect of dykes in the Westoe workings is described in Section 8.6.

8.4 Blackhall Colliery

Blackhall Colliery was the most southerly of the Durham coastal collieries, Figure 8.1, but due to cumulative water problems in the workings, ceased production early in 1981. Pumping is still continuing from the mine, to prevent flooding of the adjacent Horden workings.

8.4.1 Mine Development

Extraction was restricted to single seam working in the Low Main (J) seam, although some working of the Hutton (L) seam had also occurred in the shaft vicinity. Development progressed steadily eastwards until at the time of closure, the workings extended some 8 km from shaft bottom.

Figures 8.2 and 8.3 show the workings under investigation, but only include those dating back to the late 1960's. A large regional fault bisects the workings and conveniently divides them into the northern and southern areas.

In the southern area, the workings comprise a J10 series which run parallel to the major fault. A J20 series was then worked perpendicular to the fault and in all but one case, terminated beneath it with considerable water problems. As a result, the J50 and J60 series faces were reorientated to run parallel to the fault.

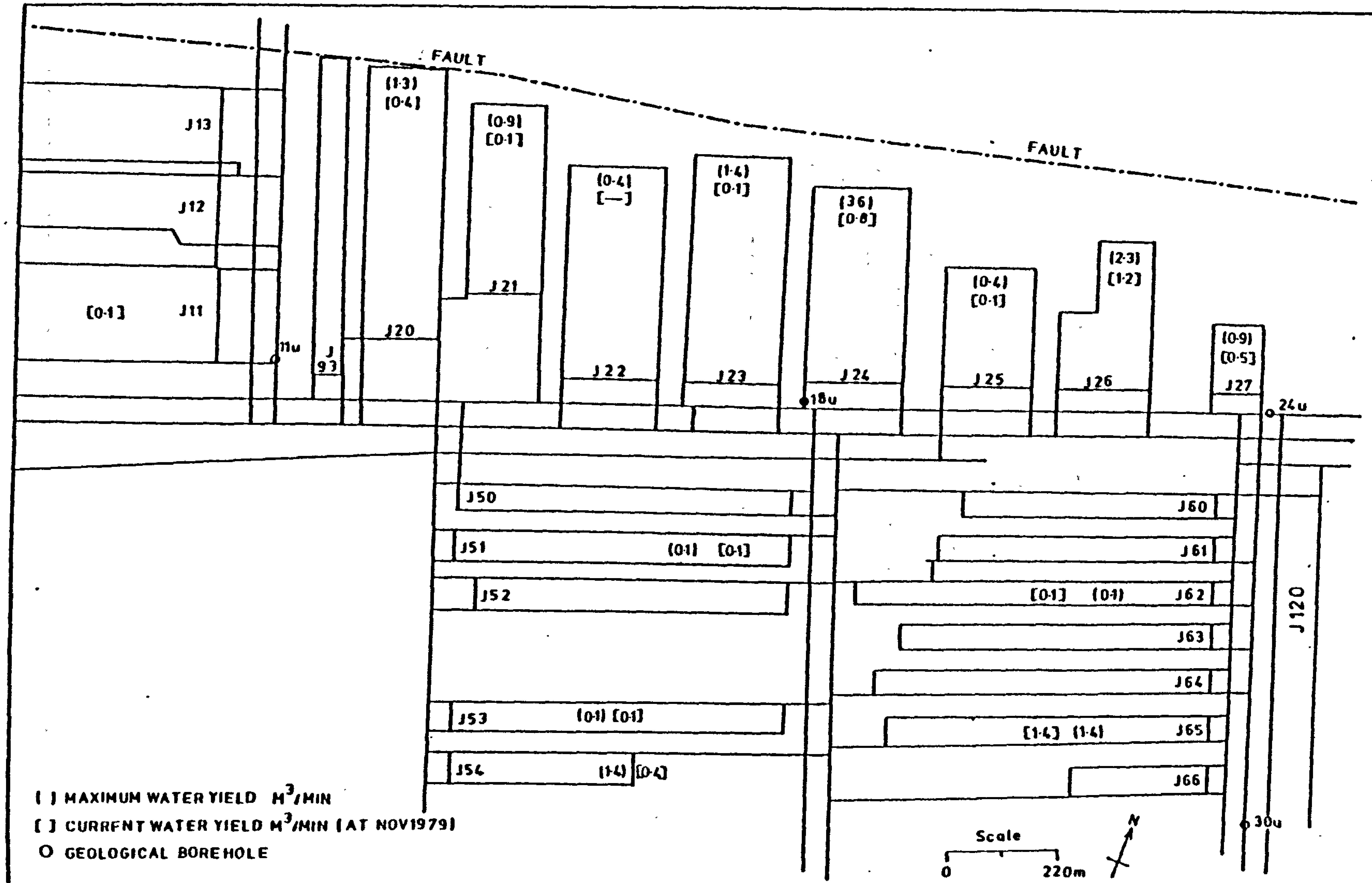


Figure 8.2 Southern Area, Low Main (J) Seam, Blackhall Colliery

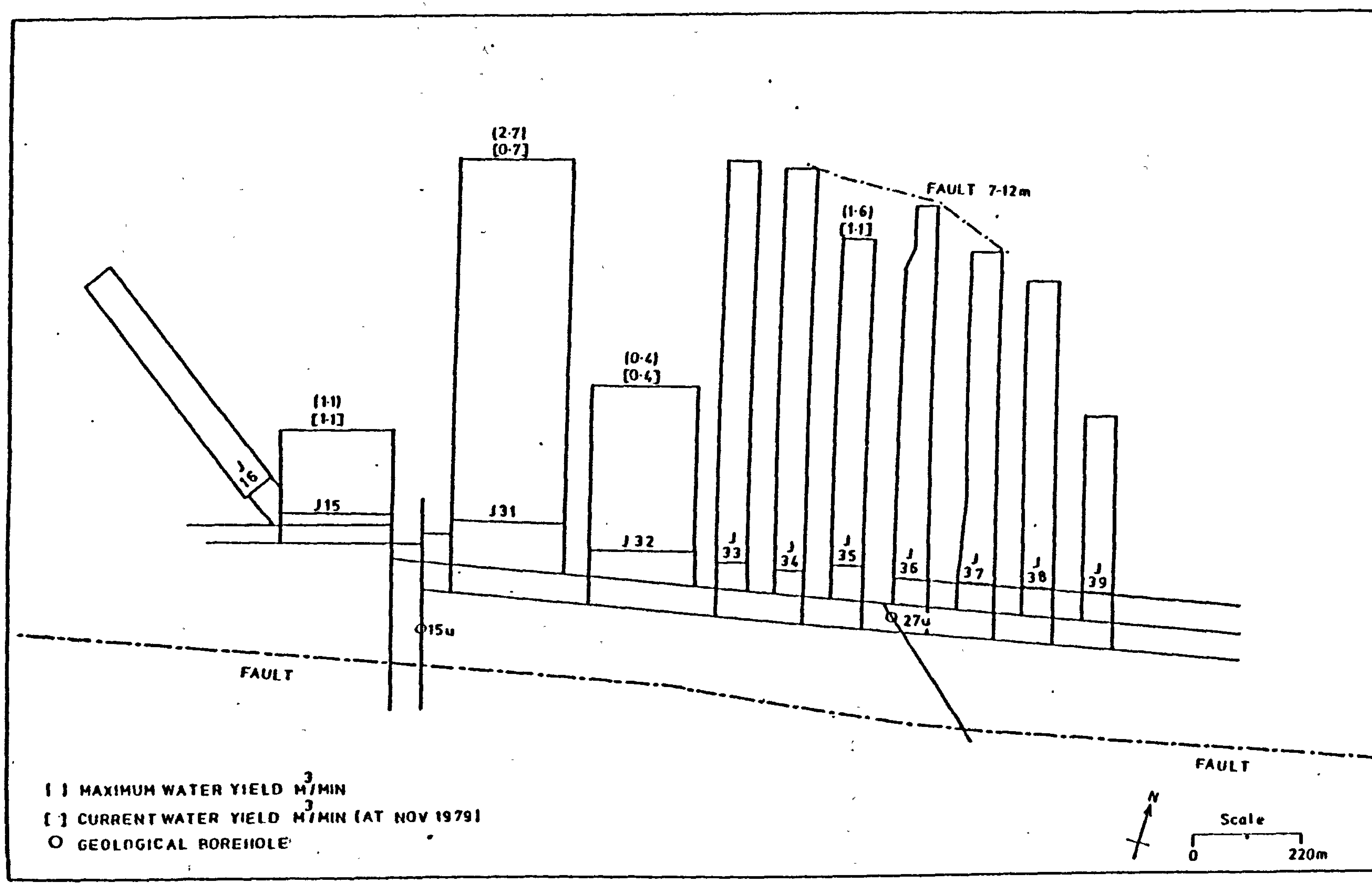


Figure 8.3 Northern Area, Low Main (J) Seam, Blackhall Colliery

Face widths of about 200 m were used for the J10 and J20 series faces. However, the severe water problems encountered on the J20 series, resulted in a face width reduction to 65 m for the J50 and J60 series.

In 1971-72, the major fault was crossed with little water being encountered. A J30 series of faces was then developed eastwards, running perpendicular to the fault and with initial face widths of 200 m. However, water problems encountered on some of these, together with those in the southern area, resulted in the face width being reduced to 65 m. Some of the J30 series faces terminated against a small East-West fault with a throw of between 7 - 12 m.

8.4.2 Mine Geology

Figure 8.4 shows geological sections across the northern and southern areas of the Blackhall workings. The Coal Measures strata is comprised of interbedded sandstones, siltstones, mudstones and seatearth/coal seams typical of a cyclothermal sequence. In the southern area, the amount of cover between the Low Main (J) seam and the base of Permian (BOP) thins to the east and south-east. North of the fault, the amount of Coal Measures cover increases rapidly eastwards. Localised strata lithology varies between the northern and southern areas. A thick sequence of sandstone above the High Main (E) seam in the north, is not as well developed across the southern area. In both areas, the strata dips gently eastwards.

The major fault is thought to be a 'hinge' type, with its pivot to the west and a throw which increases progressively eastwards

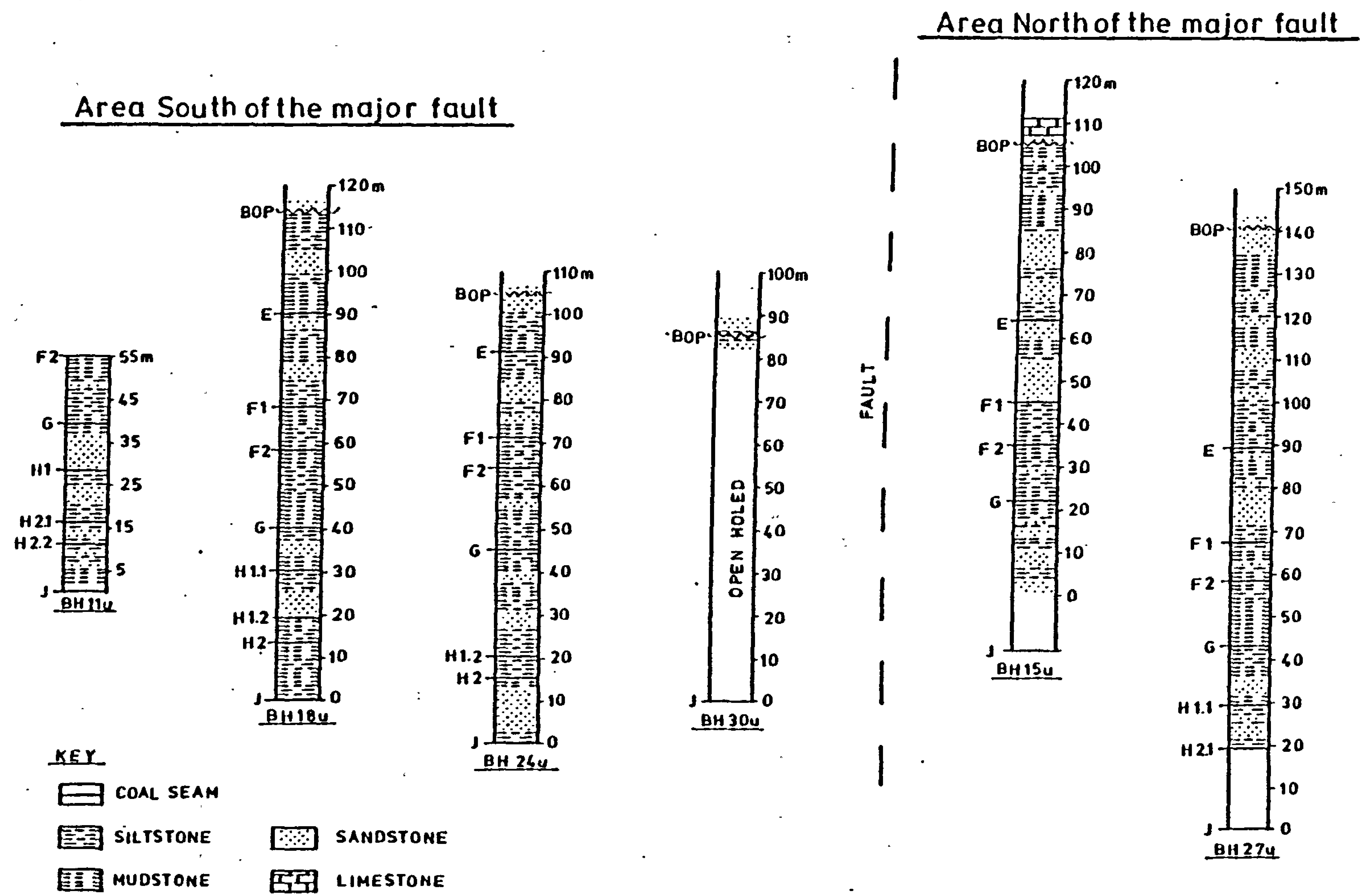


Figure 8.4 Geological Section across the Blackhall Colliery Workings

across the workings (13 - 37 m). Numerous small faults are found intersecting the workings, particularly in the southern area. Most of these appear as low angle, small displacement (< 1 m) faults, with a North-West to South-East trend. They are thought to be associated with a washout structure which crosses the area. Some larger faults (up to 5 m) have been encountered and these are thought either to be associated with the major fault or possibly one of the minor regional trends mentioned previously.

8.4.3 Occurrence of Water

8.4.3.1 Development Roadways

All water encountered during roadway development has been very small when compared with the quantities yielded by the production faces. It occurs as either small feeders or droppers and seldom constitutes more than a nuisance value.

8.4.3.2 Production Faces

All major water occurrences encountered within the workings have been associated with longwall faces. Figures 8.2 and 8.3 show the examined areas, along with the maximum and residual yields where applicable (November 1979). In each case, the water occurred as either intermittent or continuous droppers and/or feeders along the face line (Chapter 5, Section 5.2.2). Yields have also been encountered from the goaf area. Finally, during 1980, the J120 and J121 faces also experienced water and although no detailed figures are available, the combined yield was thought to be about 3.5 m^3/min.

8.4.4 Water Yield and Predicted Subsidence

A total of 35 faces within the Low Main (J) seam, worked between 1967 and 1979 have been examined. Of these, 18 (51%) were found to have experienced water in more than nuisance quantities. Distribution of the occurrences can be conveniently divided into areas north and south of the major fault and are summarised in Table 8.1.

8.4.4.1 Southern Area

Figure 8.2 shows that with the exception of J93, all the J20 series faces which worked perpendicular to the major fault encountered water in quantity. Predicted tensile strains at the base of Permian (BOP) of 6 - 8 mm/m and up to 2 mm/m at seabed can therefore be associated with these occurrences. However, not all the reorientated J50 and J60 series faces experienced water with tensile strains of 4 - 6 mm/m at BOP and up to 1 mm/m at seabed. Similarly, tensile strains of up to 7 mm/m at BOP on J11, J12 and J13 can only be associated with very small quantities of water on J11.

8.4.4.2 Northern Area

Figure 8.3 shows that only 4 faces encountered water and of these, 3 have tensile strains at BOP of between 6 - 7 mm/m, while the fourth, J35, has a predicted BOP strain of only 3 mm/m. The remaining J30 series dry faces have BOP strains varying between 2 - 5 mm/m. In each case, the predicted tensile strain at seabed is less than 2 mm/m.

	No of Faces	Current Yield	
		m^3/min	(gpm)
South of Fault - Wet	14	6.8	(1485)
South of Fault - Dry	10	-	-
North of Fault - Wet	4	3.4	(750)
North of Fault - Dry	7	-	-
	35	10.2	(2235)

Table 8.1 Quantity of Water Yielded to the 1967-79 Workings (at November 1979), Blackhall Colliery

	Total Predicted Tensile Strain (mm/m)	
Face Series	Base of Permian	Seabed
J10 (South of Fault)	up to 8 mm/m	up to 2 mm/m
J20 (South of Fault)	up to 9 mm/m	up to 2 mm/m
J50 and J60 (South of Fault)	up to 7 mm/m	up to 2 mm/m
J10 and J30 (North of Fault)	up to 7 mm/m	up to 2 mm/m

Table 8.2 Total Predicted Tensile Strains, Blackhall Colliery

8.4.4.3 Total Predicted Tensile Strain

Total strain values can be calculated for the interaction of faces worked in parallel and these are given in Table 8.2. In summary, maximum predicted tensile strains of between 7 - 9 mm/m at BOP can be associated with both wet and dry faces in both areas of the Blackhall workings.

8.4.5 Geological-Strain-Water Relationship

8.4.5.1 Southern Area

Figure 8.2 shows that the J20 series faces have been worked perpendicular to and beneath the hade of the major fault which bisects the workings. Geological evidence suggests that Permian strata in the north has been brought into contact with upper Coal Measures sequences in the south, Figure 8.5. Similarly, High Main (E) sandstones to the north will have been displaced against even lower lithological sequences in the south. The 'pivotal' nature of the fault suggests that the Permian-Coal Measures contact starts to occur in the vicinity of the J10 series faces and increases eastwards, with 30 m of contact in the vicinity of J27. This would explain the relatively dry nature of the Blackhall workings prior to the mid-1960's.

Two further points connected with the major fault also need to be considered. Firstly, recharge may be occurring directly along the fault plane. However, the variable nature of 'fault gouge' and the very small quantities of water encountered when crossing the fault, suggest that this is not happening. Secondly, all faults

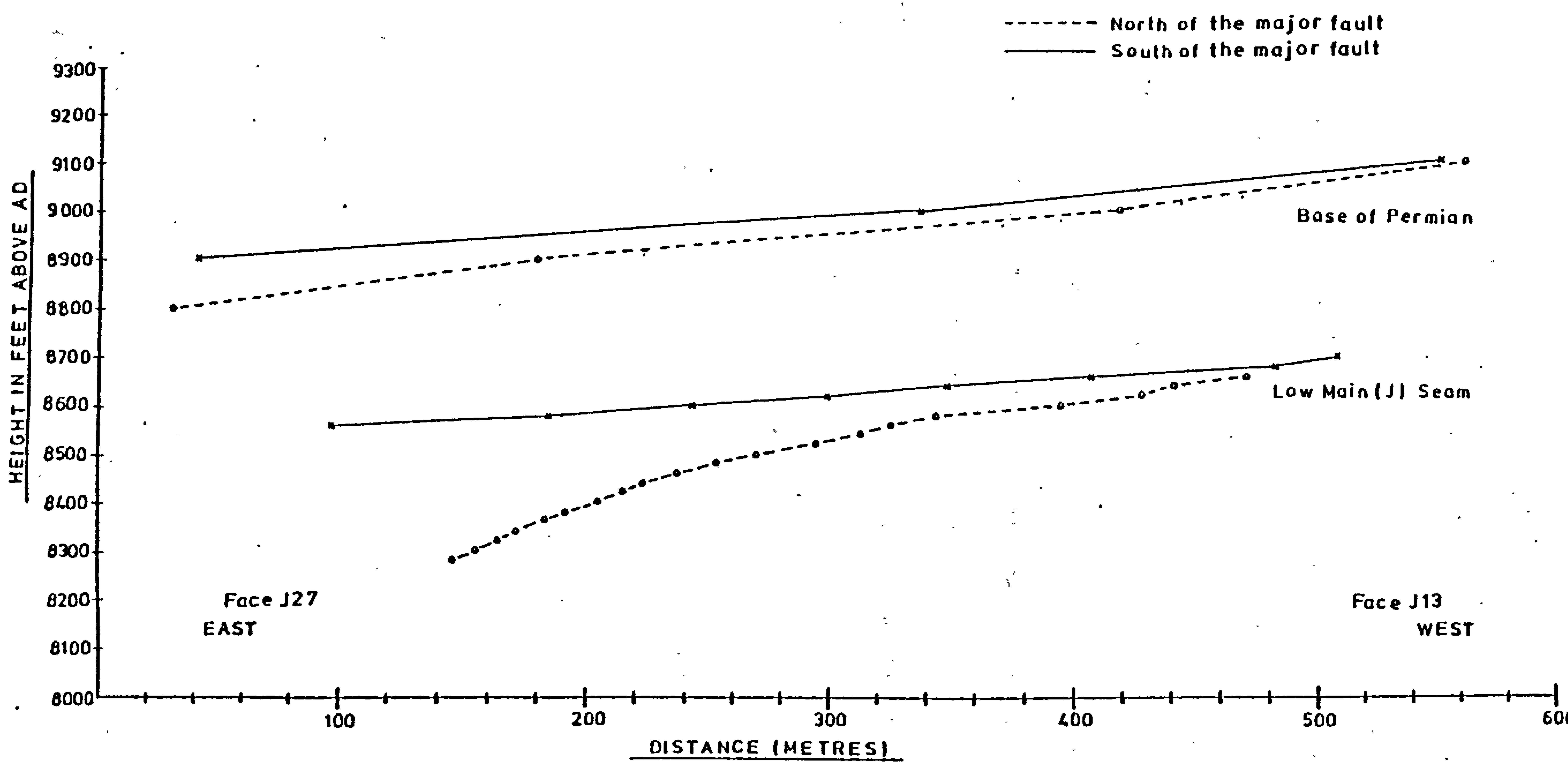


Figure 8.5 Section along the major fault, Blackhall Colliery

usually have a host of secondary structural features associated with them, Sherbourne Hills (99).

Water occurrences on the J20 series faces are thought to originate from the interaction of longitudinal and transverse tensile strains on saturated Coal Measures horizons in the vicinity of the major fault. These have been recharged over geological time, with water from the Permian and possibly the High Main (E) sandstones, and are probably intersected by additional structural features. Intermittent water problems on the J50 and J60 series faces may be related to the effects of tensile strain on a progressively thinning cover to BOP. However, more importantly the overlying Coal Measures probably contain discontinuity systems associated with structural and sedimentological features. In addition, secondary aquifer systems in the sandstone units are also likely to have formed, by direct vertical recharge from the Permian.

A natural flow regime can therefore be envisaged, which comprises a vertical component from the overlying Permian and a horizontal component from the Permian-Coal Measures contact at the fault. The formation of induced fissure networks onto this regime, will therefore allow water to be transmitted from the saturated horizons to the workings. However, the network must be of sufficient magnitude to penetrate the various strata types through which it passes.

Under ideal conditions, an initial flow will climb to a maximum and remain near this value if direct recharge is occurring from an infinite reservoir (the sea or Permian). Time dependent sealing effects may eventually reduce the magnitude of the yield. However, the variable nature of the Permian or Coal Measures strata might only allow the formation of 'secondary or finite' reservoirs. The tapping of such a reservoir, will result in a maximum yield which eventually decreases to some residual value, indicating steady-state conditions (water in = water out) or dry up.

In each southern area occurrence, an initial maximum yield has always decreased over a period of time to some residual value or dried up. This indicates that a reservoir draining effect is occurring which is supported by exploratory boreholes which have tapped feeders with either steady or decreasing yields before subsequent sealing.

8.4.5.2 Northern Area

Water occurrences on J31 and J35, Figure 8.3, are thought to be associated with the interaction of longitudinal strain profiles on geological-fault structures against which they have terminated. No faults exist near the J15 and J32 occurrences, although they may be related to secondary structural features associated with the major fault, especially since neither J16 or J31 experienced problems in this area.

8.5 Horden Colliery

8.5.1 Mine Development

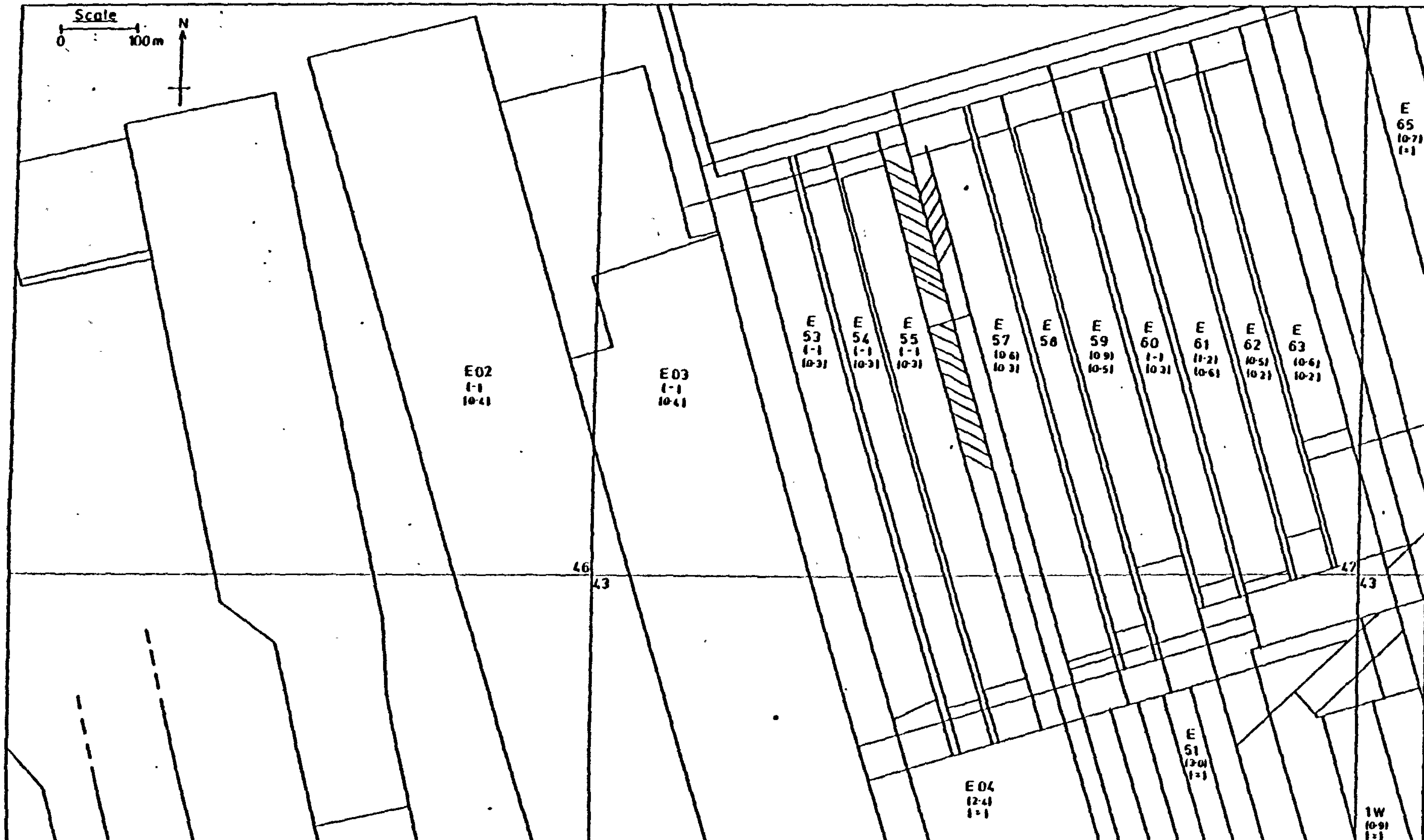
Coal production at Horden Colliery is currently from three seams, with longwall extraction occurring in the Main (F) and Yard (G) seams and herringbone partial extraction in the High Main (E) seam, Figure 8.7. Historical extraction has also occurred from the Low Main (J) and Hutton (L) seams.

Extraction has occurred progressively eastwards and the workings can be arbitrarily divided into areas north and south of the main arterial roadways, Figures 8.6 - 8.9. In the northern area, the panels have been worked in an approximate North-West to South-East direction and terminated in the vicinity of the Easington Fault, which forms the boundary between the Horden and Easington Collieries.

8.5.2 Mine Geology

Figure 8.10 shows two geological sections representative of the Coal Measures strata encountered in the Horden workings. The strata comprises typical cyclothermal sequences, although from Figure 8.10, a well developed sandstone, some 15 m thick, can be seen at the 'D' seam horizon.

No major faults bisect the workings, although the northern boundary coincides with the Easington Fault. The Easington Fault is regional in nature and increases in throw eastwards. Across the area of the E50 and E60 series faces, Figure 8.6, the throw varies from 45 - 70 m with a hade of 20^{o}. It does not exhibit a well defined shear plane, but rather a 'shatter' zone containing gouge material and fragmented rock. The downthrow side lies to the north of the fault.



( ) MAXIMUM WATER YIELD m³/min ( ) RESIDUAL WATER YIELD m³/min (-) or (-) NOT KNOWN

Figure 8.6 Northern (West) Area, High Main (E) Seam, Horden Colliery

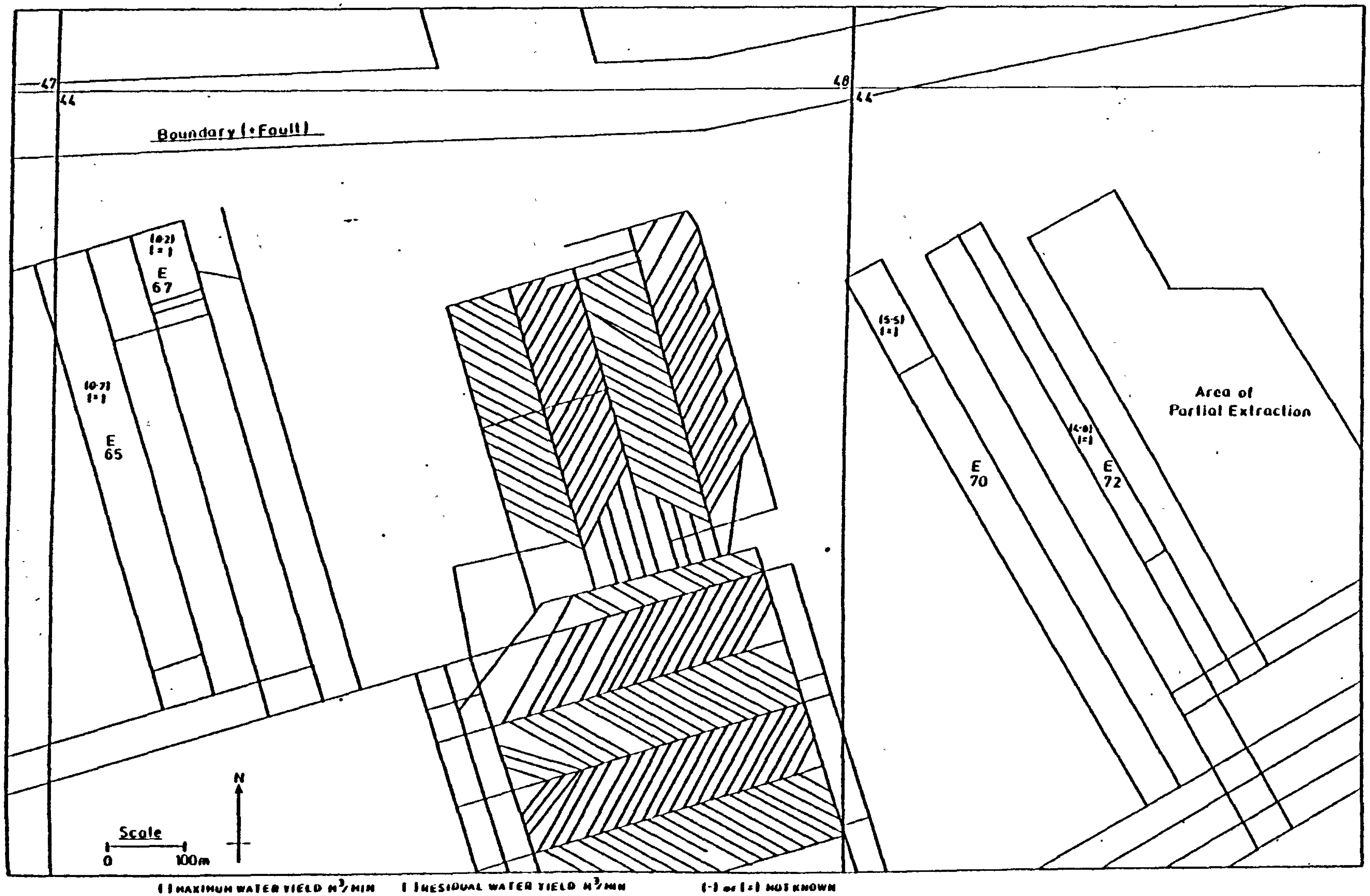


Figure 8.7 Northern (East) Area, High Main (E) Seam, Horden Colliery

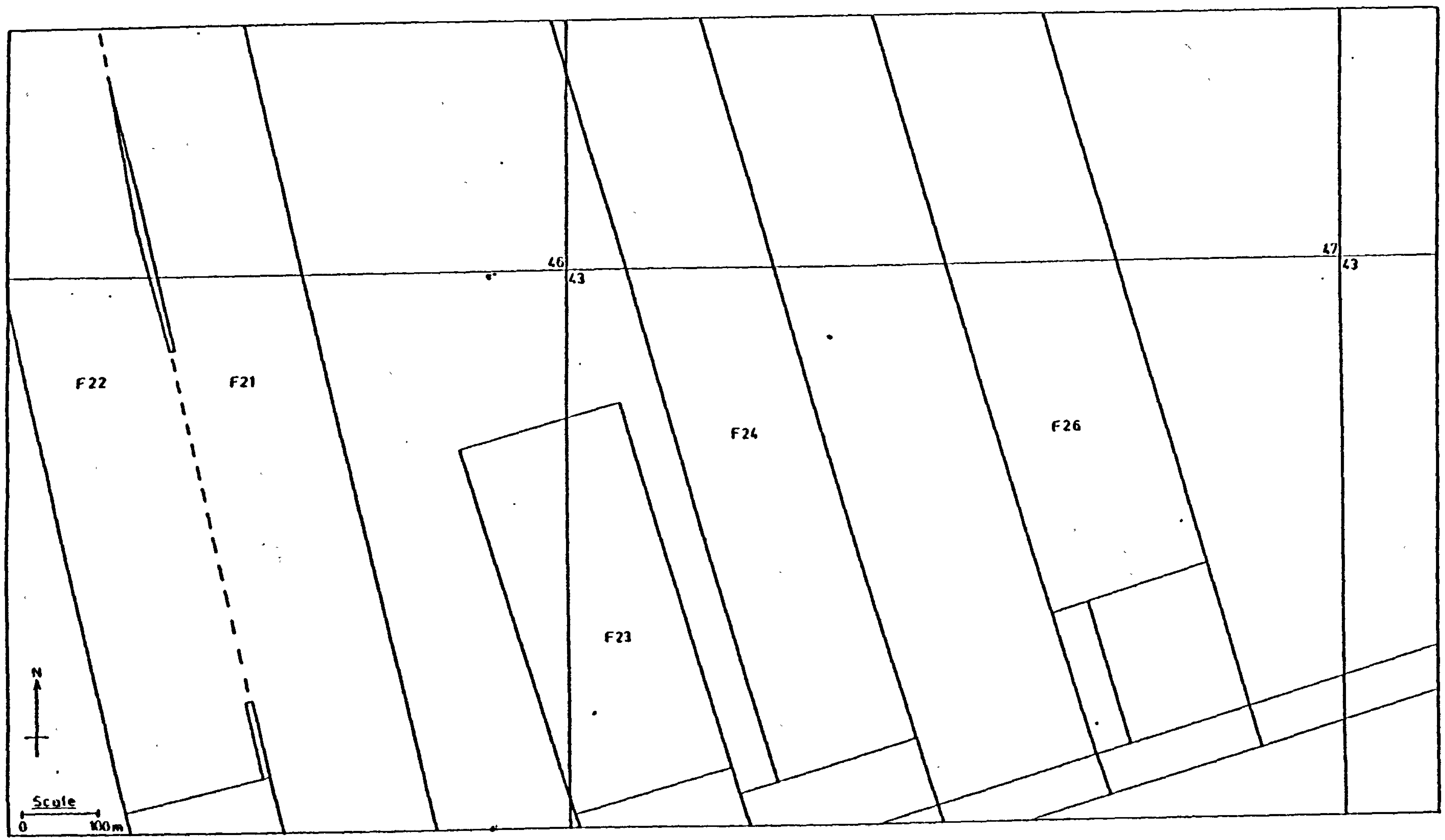


Figure 8.8 Northern Area, Main (F) Seam, Horden Colliery

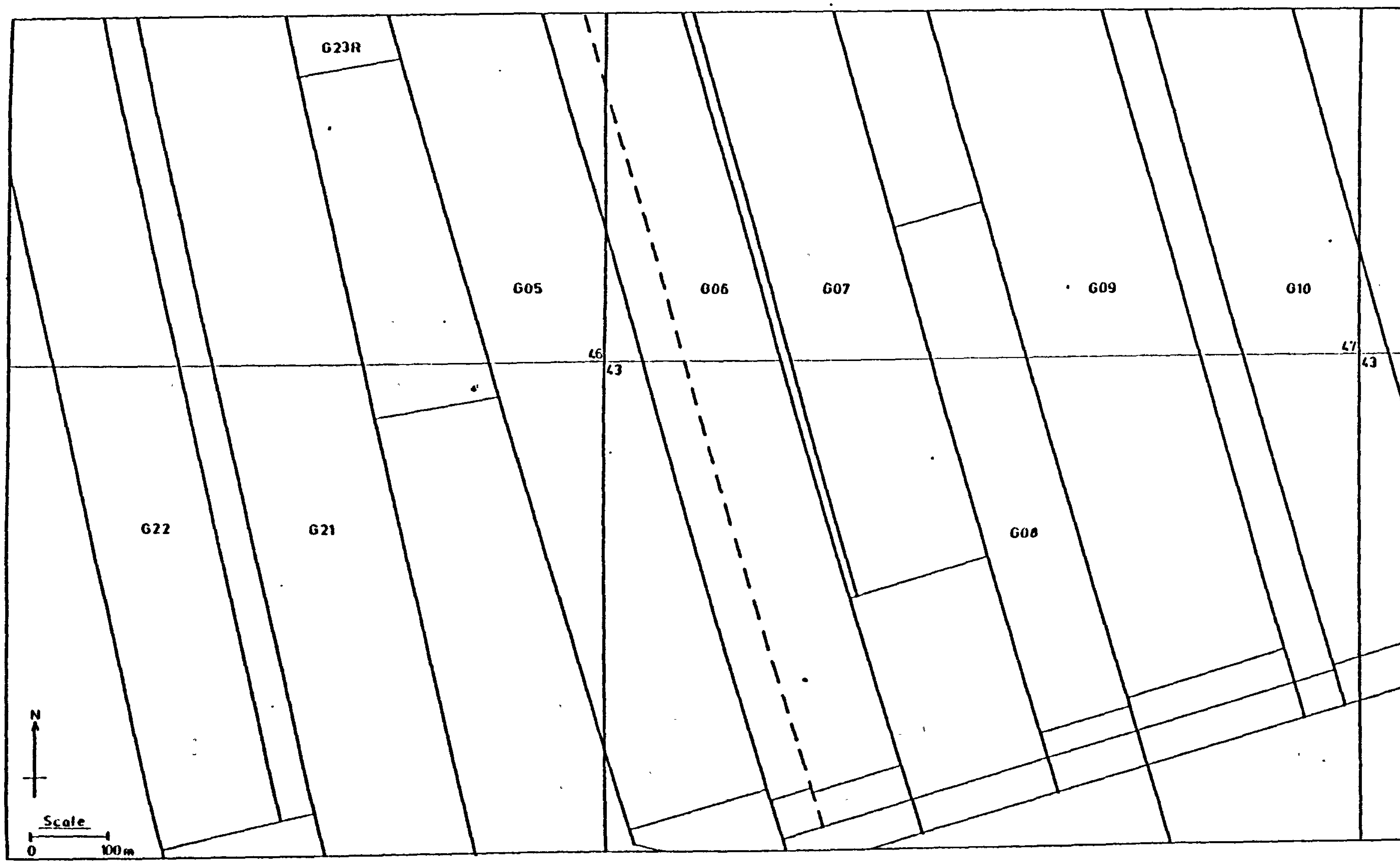


Figure 8.9 Northern Area, Yard (G) Seam, Horden Colliery

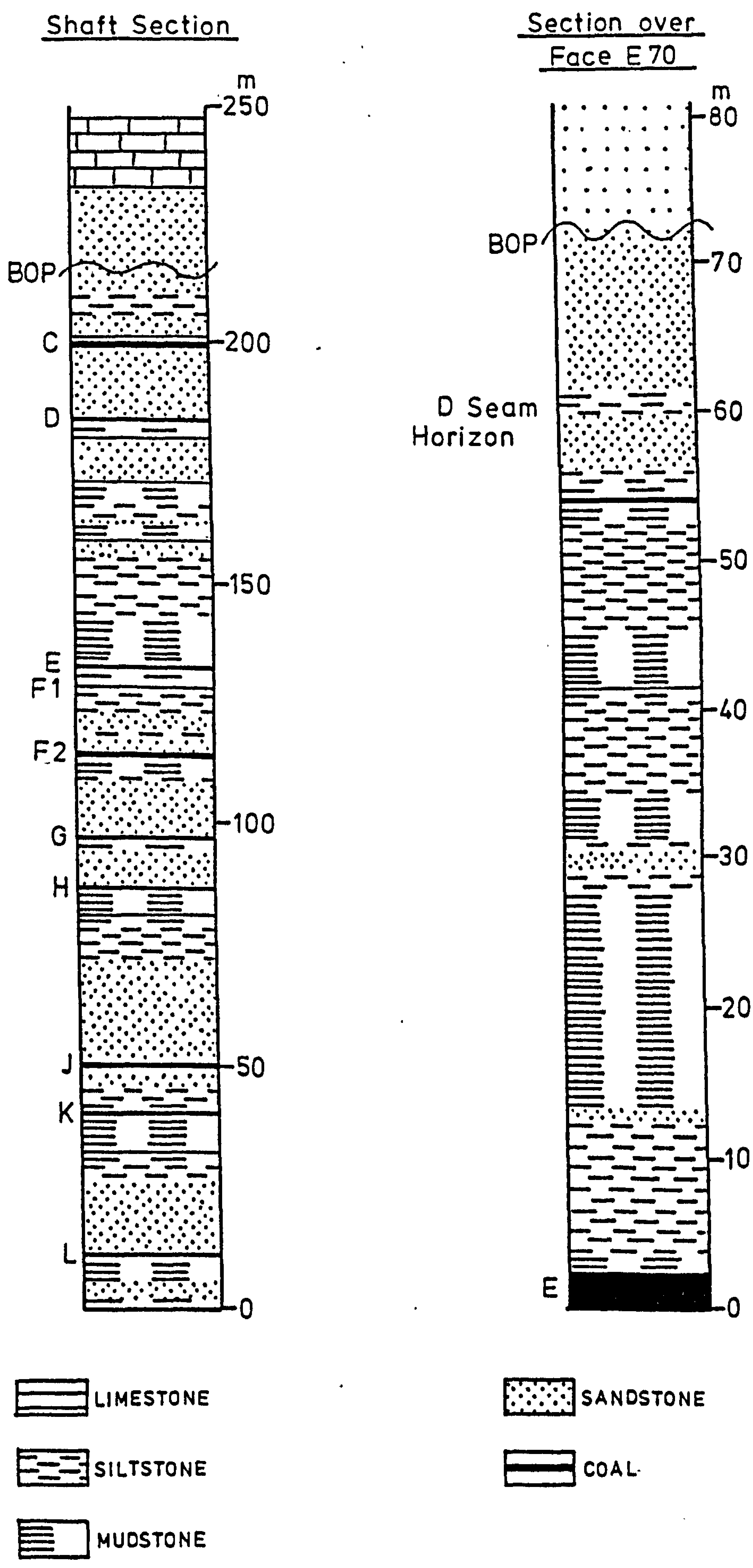


Figure 8.10 Geological Sections representative of Coal Measures Strata at Horden Colliery

8.5.3 Water Occurrence

Horden Colliery is currently pumping about 10.5 m^3/min of water, 8.3 m^3/min of which can be related to the High Main workings. Virtually all the water has occurred on longwall panels, although nuisance quantities have occasionally been encountered in drivages and the partial extraction areas.

8.5.3.1 Production Faces

Figures 8.6 and 8.7 show the High Main faces which have experienced water, along with their maximum and residual yields where available, Wilson (100) and Rowell (101). In each case, the water occurred as either feeders or heavy droppers along the face line or within the goaf area.

Although most of the water at Horden has occurred in the High Main seam, faces in the Main, Yard and Hutton seams have also experienced nuisance quantities. In the Main seam, the water was thought to originate from reservoirs in the overlying High Main. Small quantities of nuisance water were experienced in the Low Main seam until the workings reached their most easterly extent and extraction was started on an area in the vicinity of the Blackhall J15 and J16 faces. Water was then encountered in quantity, with amounts of up to 4 m^3/min being experienced.

8.5.4 Water Occurrences and Predicted Subsidence

A total of 40 faces in the Horden workings have been examined. In the High Main seam, 19 faces were examined and of those, 18 were found to have liberated water in significant quantities.

8.5.4.1 The High Main Faces

Examination of Figures 8.6 and 8.7 reveals that with the exception of E04, E51 and 1W, all were worked towards the Easington Fault and with the exception of E58 encountered water in quantity. Predicted tensile strains vary between 5 - 10 mm/m at BOP and 1 - 3 mm/m at seabed. The more southerly E04, E51 and 1W faces have predicted tensile strains of between 7 - 10 mm/m at BOP and up to 2 mm/m at seabed, for water quantities varying between 0.9 - 3.0 m^3/min.

It is interesting to note that the E58 face remained dry, even though the faces on either side experienced water. Predicted tensile strains for E58 are 5 mm/m at BOP and less than 1 mm/m at seabed.

8.5.4.2 Total Predicted Tensile Strain

Total predicted tensile strains for the interaction of parallel faces in the same and adjacent seams, indicate that for the High Main E50 and E60, Main F24 - 26 and Yard G05 - 10 series faces, up to 12 mm/m at BOP and 3 - 4 mm/m at seabed occurred.

However, in this area, the extraction sequence is particularly important, since the Main and Yard seam faces were worked after the High Main faces had already experienced water. Predicted tensile strains of up to 12 mm/m were therefore induced at BOP, without an increased yield from pre-existing feeders in the High Main workings.

8.5.5 Geological-Strain-Water Relationship

A total of 50% of the Horden faces examined have experienced water in quantity and of these all are located in the High Main (E) seam. In each case, the water occurred prior to extraction in the underlying Main and Yard seams, although historically complete extraction had already occurred in the Low Main and Hutton seams. It is normally assumed that complete extraction results in a zone of compression across an area, except in the vicinity of boundary and/or remnant pillars. Spatially, the High Main occurrences can be divided into two areas, those in proximity to the Easington Fault and those on the remaining E04, E51 and 1W faces.

8.5.5.1 Proximity to the Easington Fault

All faces worked in the vicinity of the Easington Fault, with the exception of E58, have experienced water in quantity. Examination of the area reveals that the fault has a throw of about 50 m in the Coal Measures and between 20 - 30 m in the Permian. Figure 8.10 shows that a well developed sandstone horizon, 15 - 20 m thick, lies close to the Permian contact. Recharge to this horizon can therefore occur both vertically from the overlying Permian and horizontally across the Coal Measure-Permian contact at the Easington Fault, Figure 8.11.1. The amount of unsaturated Coal Measures strata overlying the High Main seam, can therefore be reduced by up to 20 m. The intervening strata appears to consist of well developed mudstone and siltstone horizons, which should provide excellent impermeable barriers until disrupted. Water on the E50, E60 and E70 faces therefore originates from secondary aquifer systems formed in the 'D' seam

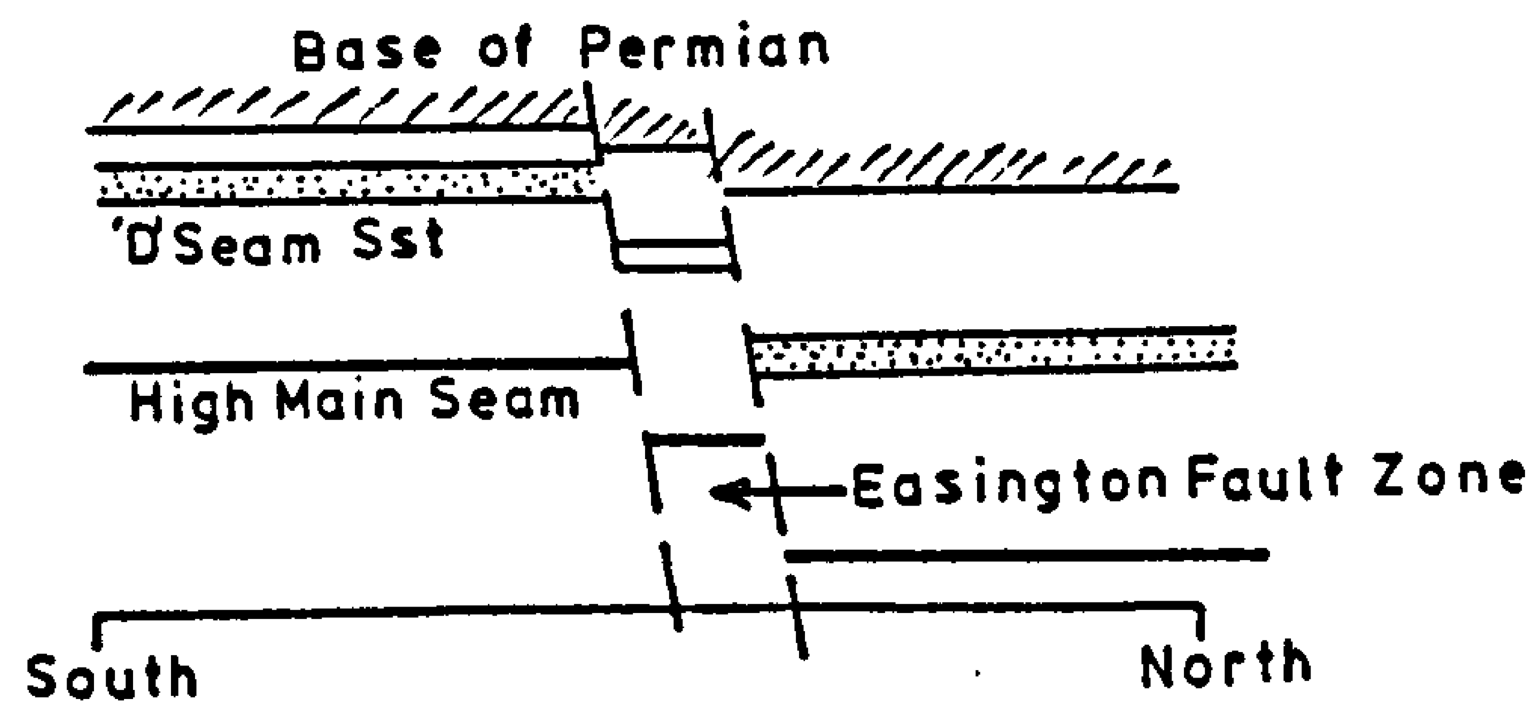


Figure 8.11.1 Schematic Geological Section across the Easington Fault, Horden Colliery

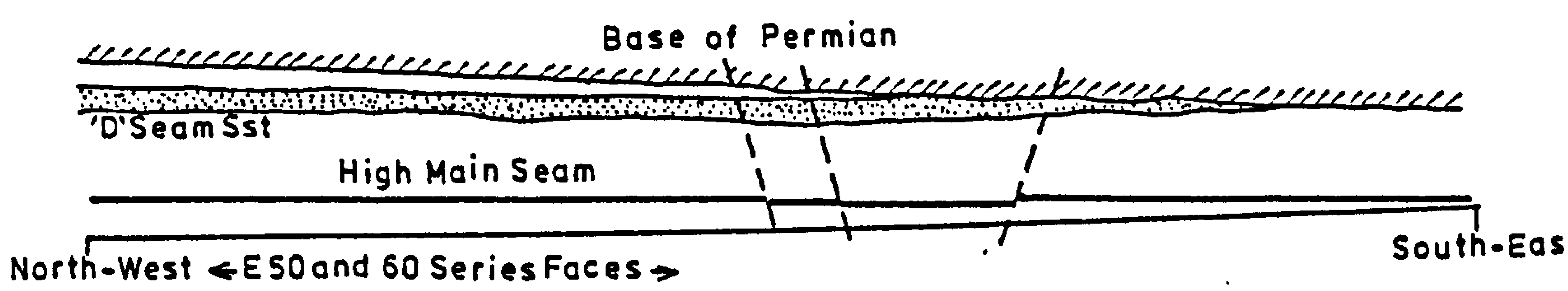


Figure 8.11.2 Schematic Geological Section N.W. to S.E. across the High Main (E) Seam Workings, Horden Colliery

sandstones, by recharge from the Permian over geological time, which have subsequently been disrupted by longwall working. Draining of a localised aquifer system by E57 and previous faces, resulted in the E58 face remaining dry. Similarly, the finite yield of an aquifer system can be associated with localised changes in lithology and hence permeability, transmissivity and the aquifer storage coefficient, Chapter 2, Section 2.5.

Recharge may also be occurring along the fault zone, from either the Permian or some other aquifer horizon. The E67 face started beneath the fault hade and encountered over 8 m^3/min of water after an advance of only 89 m. However, the other faces have all worked at greater distances from the fault zone and are therefore unlikely to have tapped this potential source of water. Secondary structural features associated with the Easington Fault, may act as focal points for strata disruption, allowing water to move along significant planes of weakness to the working horizon.

In underlying seams, faces which have worked towards the Easington Fault, have not experienced water in more than nuisance quantities for two main reasons. Firstly, no saturated horizons have been displaced into close proximity with strata overlying the workings, thus allowing secondary aquifer systems to form over geological time. Secondly, the fault zone flow regime has either been intercepted by the High Main workings or not penetrated in quantity to the underlying seam horizons. Alternatively, the workings have not worked sufficiently close to the fault zone to allow the access of water. Finally, workings in the Low Main seam which encountered significant quantities of water are also recorded as having worked in close proximity to a major fault. Although little information is available to the author, it is suggested that aquifer systems have again been moved into close proximity with the working horizon, allowing the formation of secondary aquifers in the intervening strata. Alternatively, the fault plane may have acted as the primary transmitter and this was either directly tapped or allowed secondary aquifer systems to form over geological time.

8.5.5.2 The E04, E51 and 1W Faces

Occurrences of water on these faces cannot be related to the proximity of the Easington Fault. However, the 'D' seam sandstones, which occur prominently above the High Main workings, incrop to Permian across this central area, Figure 8.11.2. Their estimated height above the 'E' seam is between 50 - 60 m. It is therefore proposed that water occurrences on the E04, E51 and 1W faces are directly related to mining induced stresses which tap these secondary aquifer systems. Recharge to the sandstone occurs principally from the overlying Permian and not from the strong horizontal component which is thought to exist nearer the Easington Fault.

8.6 Westoe Colliery

8.6.1 Mine Development

Coal production is currently by longwall mechanised working on 7 faces in 3 seams; the Main (F1), Maudlin (H) and Brass Thill (K). The method of working is either advance or retreat with face widths of about 200 m and an extraction height varying from 1.3 m in the Maudlin to 2.0 m in the Main seam. Development has currently extended some 10 km east and north eastwards from the shaft bottom. The workings can be conveniently divided into 3 areas: the northern, central and southern, by two large faults which cross the workings.

Figures 8.12 - 8.17 show the workings in each of the three seams.

8.6.2 Mine Geology

Figure 8.18 shows the geological sequence of Coal Measures strata across the Westoe workings, based on four offshore boreholes. Two boreholes are situated in the northern area and one in each of the central and southern areas.

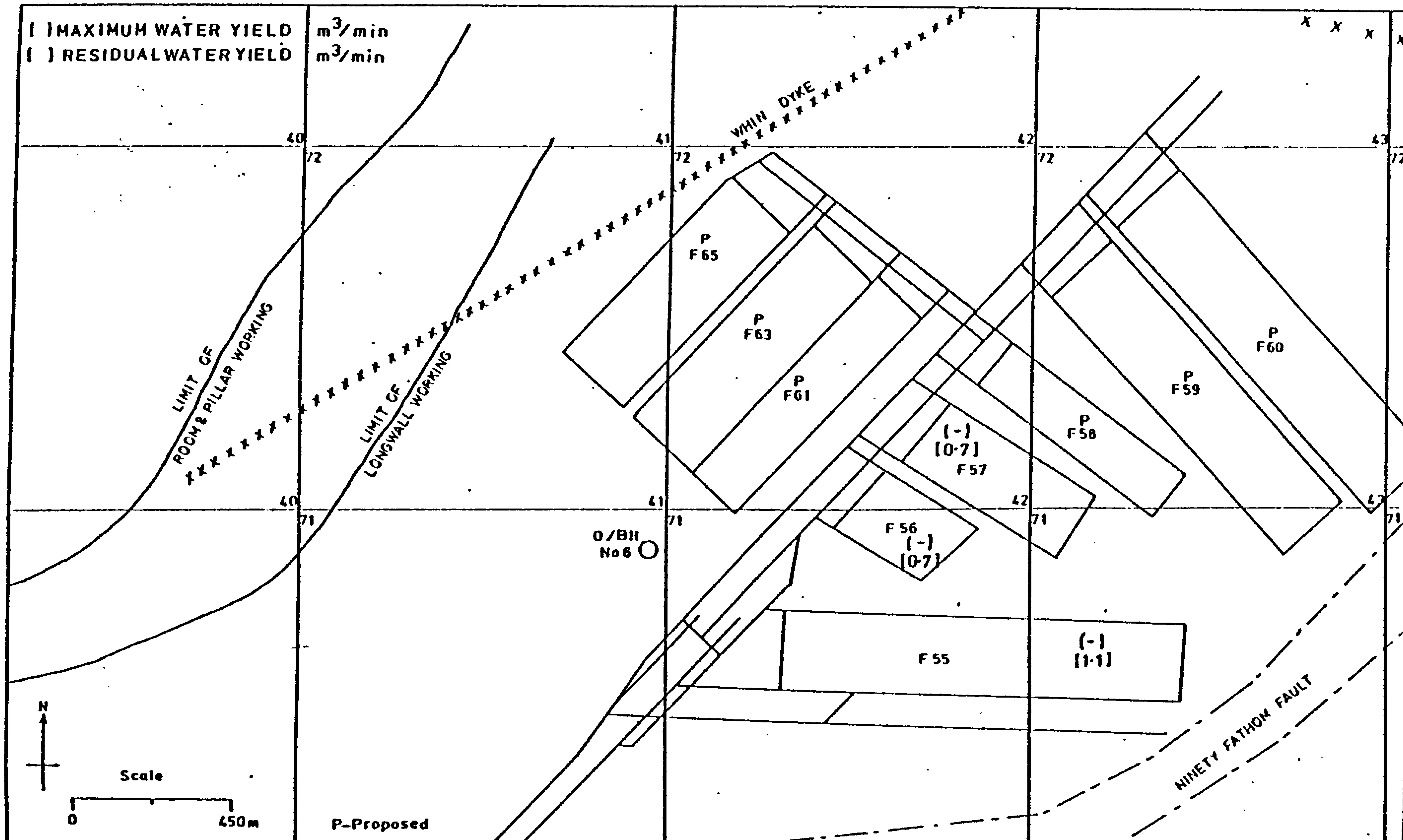
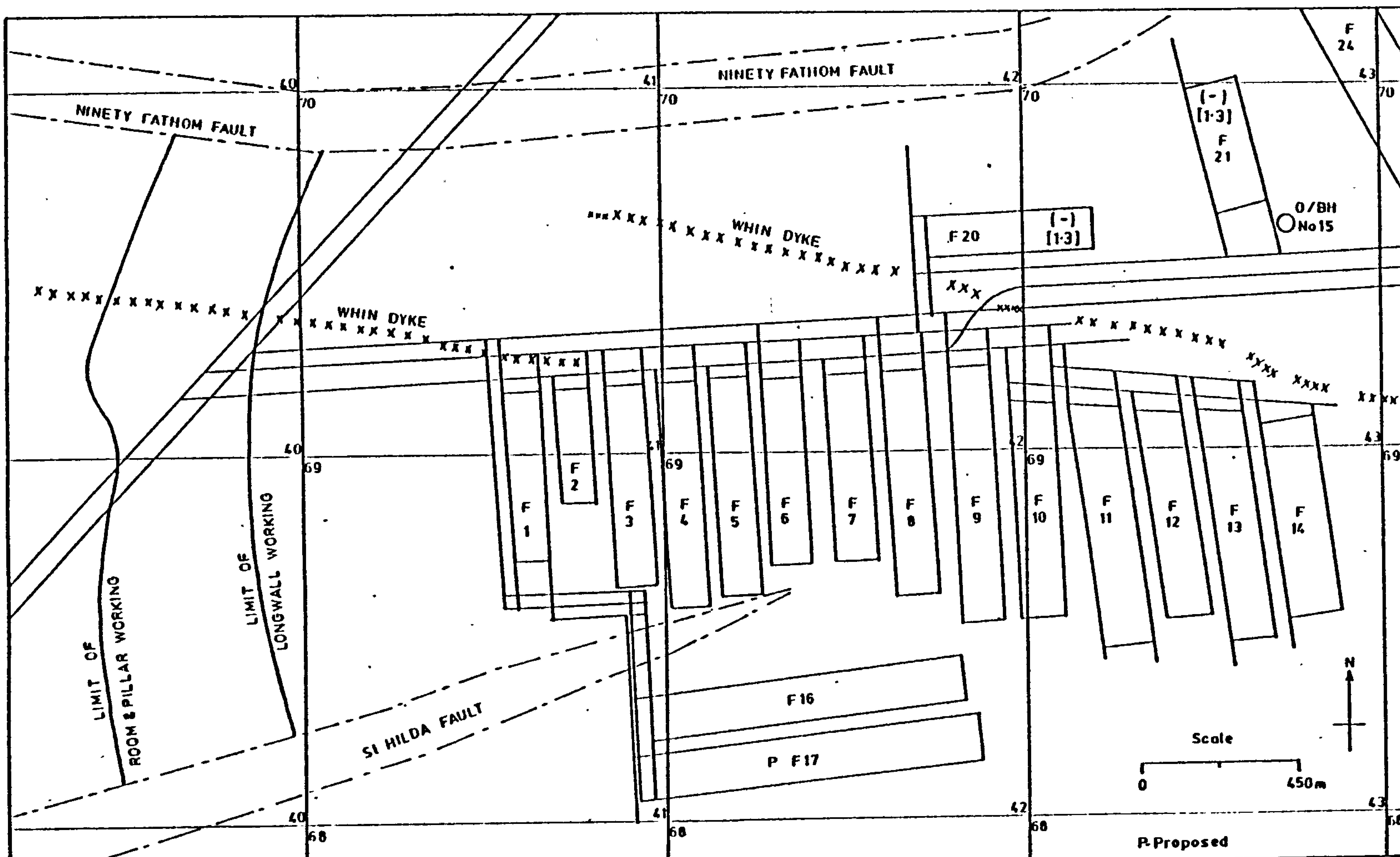


Figure 8.12 Northern Area, Main (F1) Seam, Westoe Colliery



( ) MAXIMUM WATER YIELD m³/min [ ] RESIDUAL WATER YIELD m³/min

Figure 8.13 Central and Southern Areas, Main (F1) Seam, Westoe Colliery

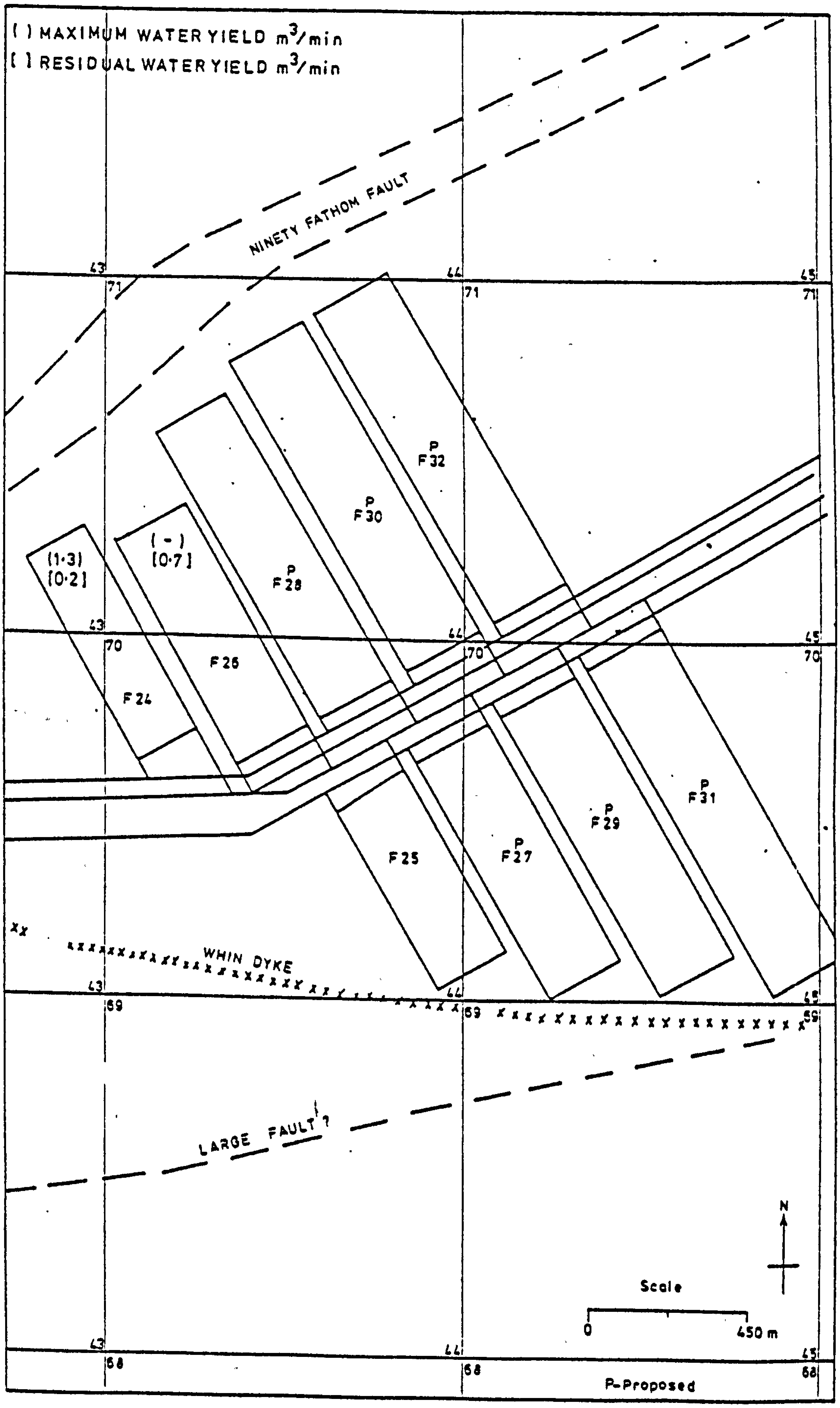


Figure 8.14 Central (East) Area, Main (F1) Seam, Westoe Colliery

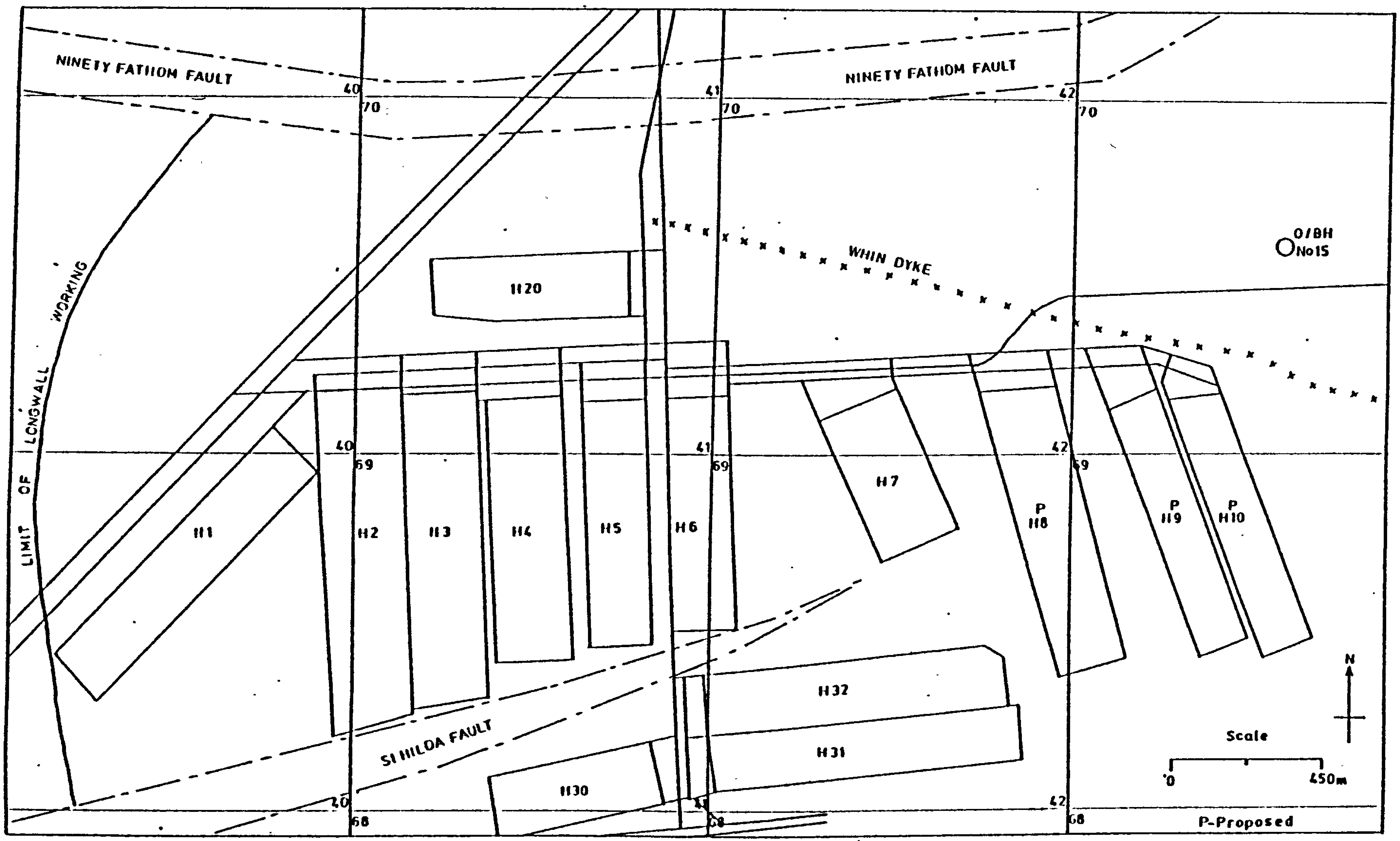


Figure 8.15 Central and Southern Areas, Maudlin (H) Seam, Westoe Colliery

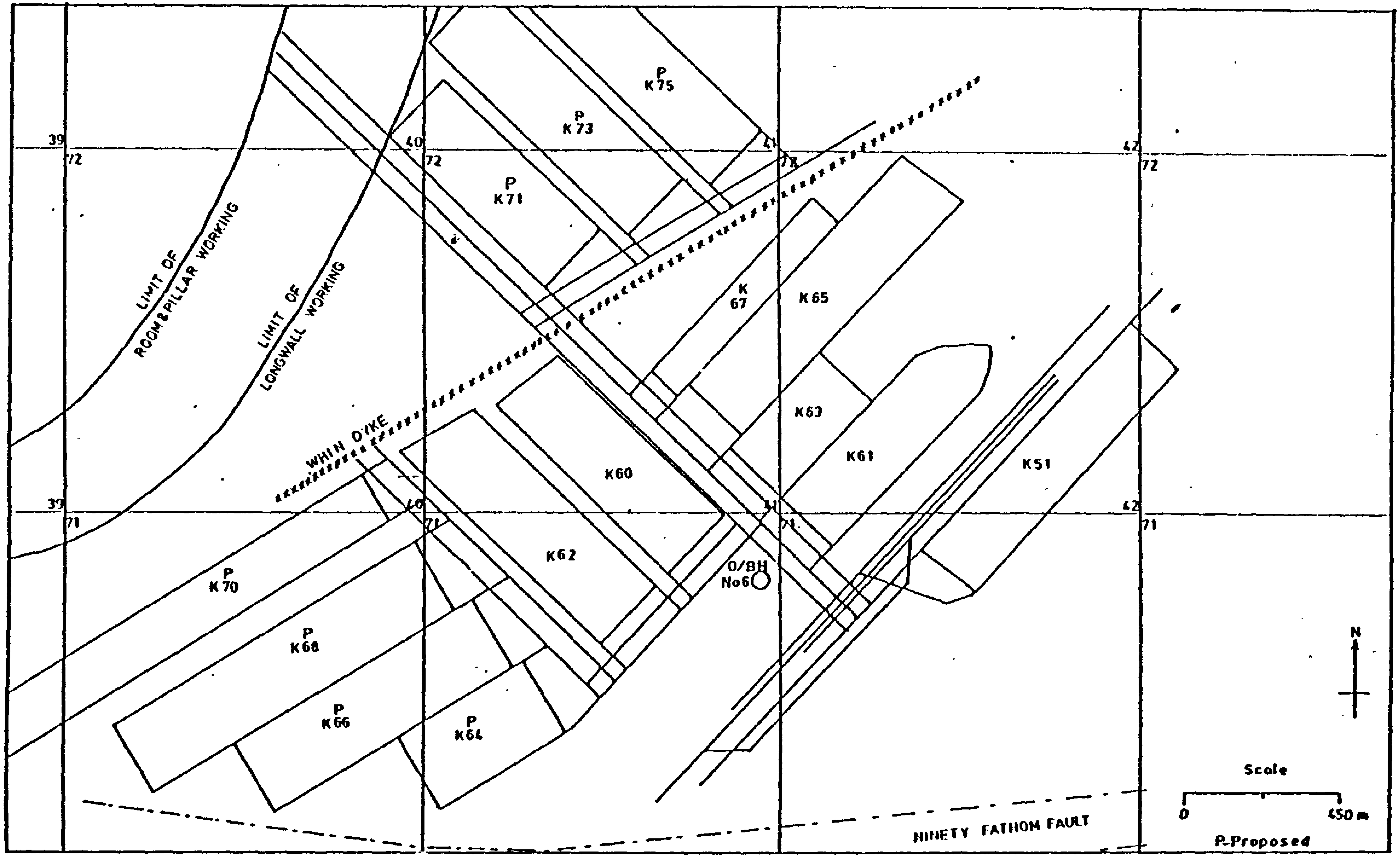


Figure 8.16 Northern Area, Brass Thill (K) Seam, Westoe Colliery

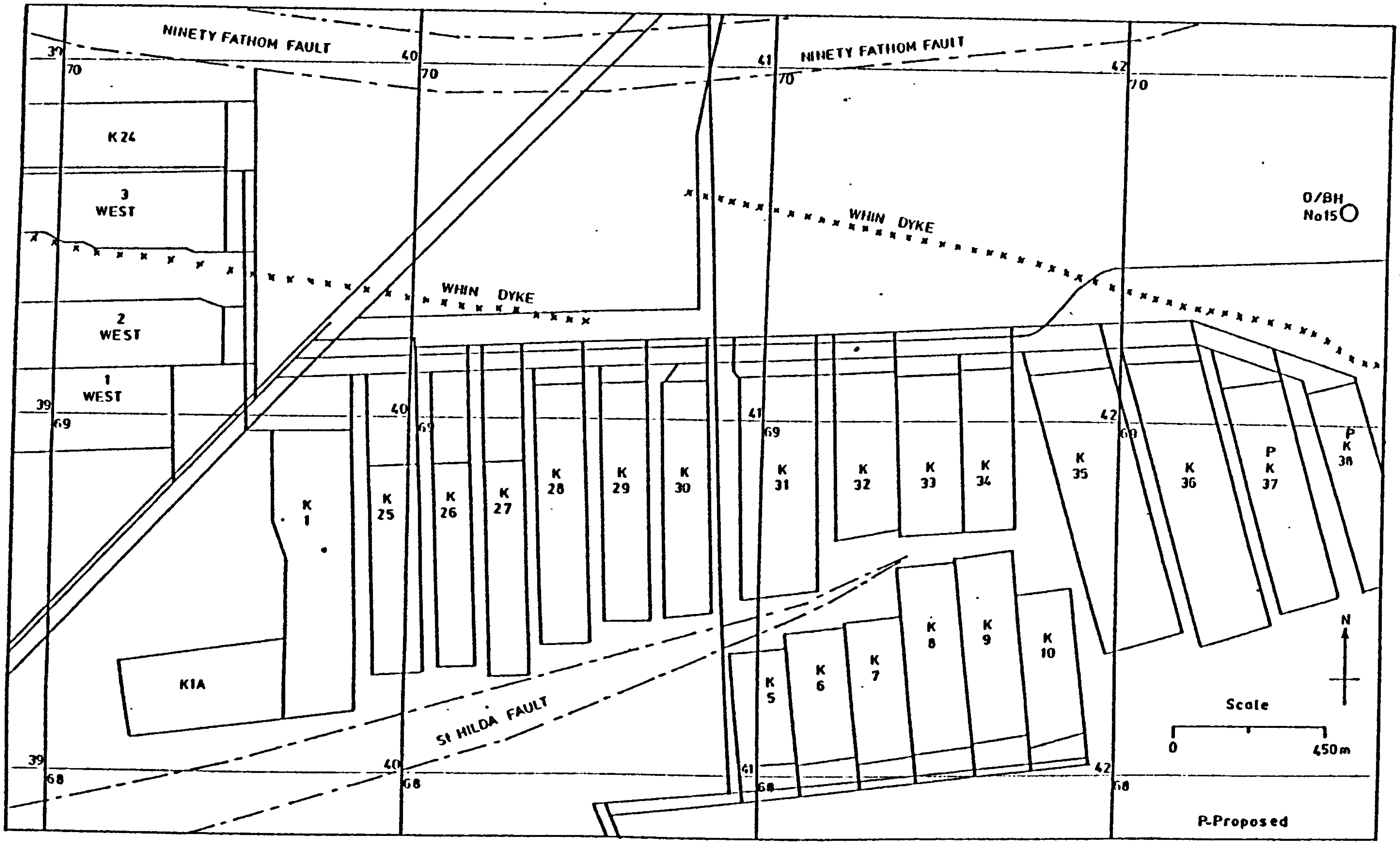


Figure 8.17 Central and Southern Areas, Brass Thill (K) Seam, Westoe Colliery

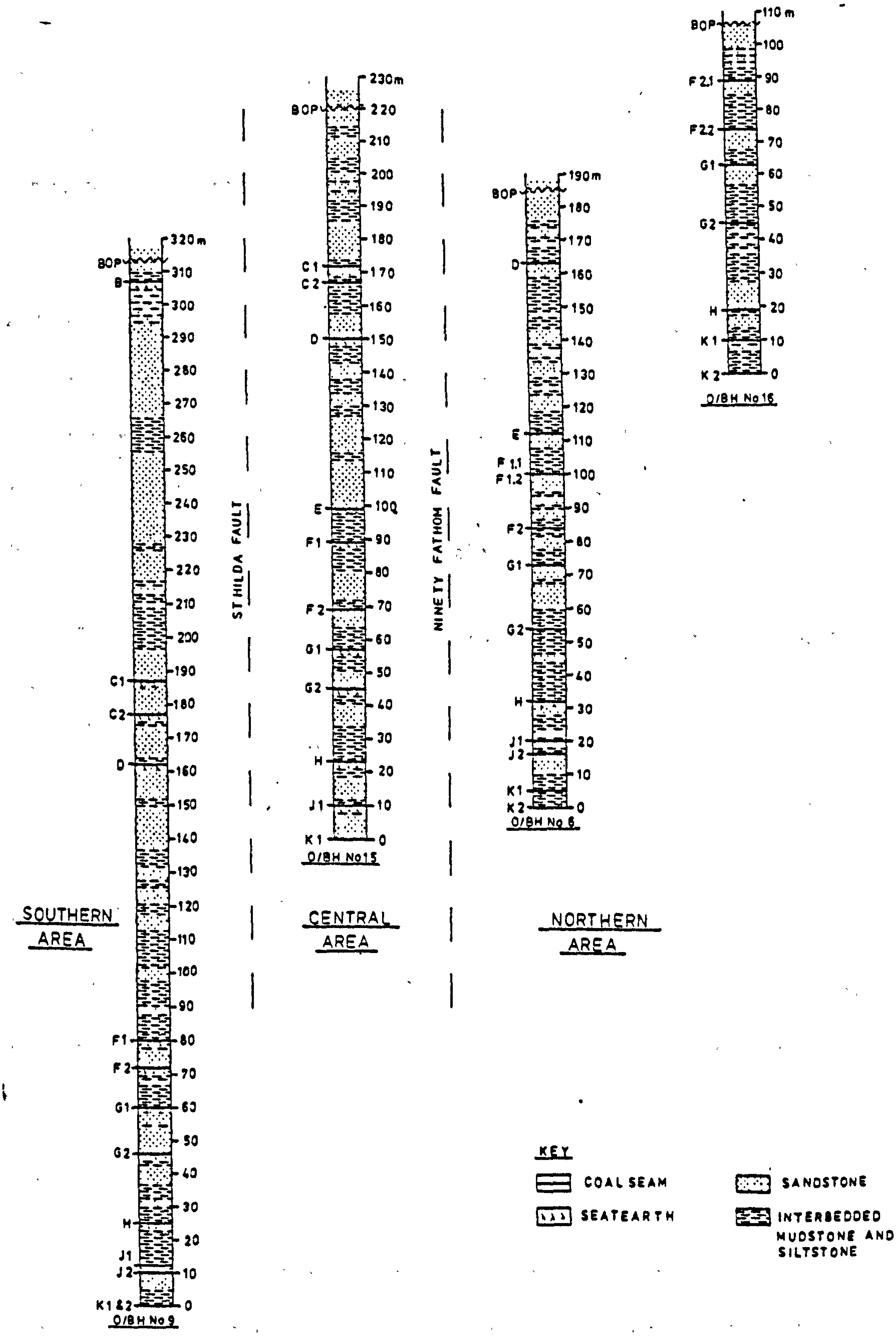


Figure 8.18 Geological Section across the Westoe Colliery Workings

The Coal Measures comprise a classical cyclothermal sequence of interbedded coal/seatearth, mudstone, siltstone and sandstone. A variable amount of cover, within the 1968 guidelines (2), exists between the Main seam and the base of Permian across the workings. Approximately 45 m of strata exists between the Main and Maudlin seams, with a further 30 m between the Maudlin and Brass Thill.

Two major faults cross the workings and both belong to the regional East-North-East to West-South-West trend. The Ninety Fathom Fault in the north, comprises a high angle 'normal' fault with a throw of 150 m, with the down throw side to the north. The St. Hilda Fault in the south, comprises a North-East to South-West 'pivot' fault which increases in throw westwards.

Two further important features associated with the Westoe workings are:

1) intersection of the workings by tertiary dyke systems.
2) an offshore buried river channel of the River Tyne.

8.6.3 Water Occurrence

Westoe Colliery is currently pumping about 8 m^3/min from the workings, of which 6.3 m^3/min is associated with the Main seam, Table 8.3.

8.6.3.1 Development Roadways

In general only nuisance water in the form of droppers and small feeders has been encountered. However, drivages across the Ninety Fathom Fault from the Main seam central area, have experienced large quantities of water which required the use of cement injection sealing techniques.

	Current Yield		
Seam	m^3/min	gpm	% Total
Main (F1)	6.3	1385	79
Maudlin (H)	0.2	50	3
Brass Thill (K)	1.5	325	18
	8.0	1760	100

Table 8.3 Quantity of Water Yielded to each Working Seam (at February 1980), Westoe Colliery

Seam	No of Faces	% Total
Main Seam (F1) Wet	7	11
Main Seam (F1) Dry	16	25
Maudlin (H) Wet	0	0
Maudlin (H) Dry	9	14
Brass Thill (K) Wet	4	7
Brass Thill (K) Dry	27	43
	63	100

Table 8.4 The Number of Wet and Dry Faces in each Seam, Westoe Colliery

Examination revealed that most of the water appeared on the north side of the fault and associated with a thick sandstone sequence in close proximity to the 'D' seam horizon. Only a small quantity of the total water encountered is thought to have been transmitted along the fault plane.

Water in nuisance quantities has also been encountered on Main seam drivages in the northern area associated with the F60 series faces and in the central area on the eastern drivages of the F30 series faces.

8.6.3.2 Production Faces

In general only nuisance water has been encountered on the majority of faces worked in all three seams, Table 8.4. However, several exceptions do exist in the Main and Brass Thill seams and these are discussed in Section 8.6.4.

8.6.3.3 Intersection of the Dyke System

Several dykes have been intercepted by either development drivages or production faces. In some cases they have proved dry, while in others very large quantities of water have been liberated. A dyke intercepted by a drivage in the region of face F11, Figure 8.13, produced a maximum yield of 4.5 m^3/min (1000 gpm) which eventually decreased to zero. Similarly, face 3W, Figure 8.17, in the Brass Thill had to be shortened when a dyke was intercepted which yielded a feeder of 2.3 m^3/min.

8.6.4 Water Occurrences and Predicted Subsidence

A total of 63 faces in the Westoe workings have been examined. In summary, only 18% of the total faces worked have experienced water in quantities greater than nuisance value. Both the Main and Brass Thill seams have encountered 'wet' faces, but none have occurred in the Maudlin.

8.6.4.1 Main Seam - Wet Faces

Of the seven Main seam faces which have liberated water, four are in the central area and three in the northern area. The three northern area faces F55, F56 and F57 have all liberated quantities varying between 0.6 m^3/min and 1.2 m^3/min. Each was worked perpendicular and in close proximity to the Ninety Fathom Fault. Predicted tensile strain at BOP varies between 7 - 10 mm/m and at seabed 3 - 5 mm/m. In the central area, four faces F20, F21, F24 and F26 have all experienced water in quantities varying from 0.6 to 1.2 m^3/min. Each face is near the Ninety Fathom Fault and for all but one, the direction of working is perpendicular to it. Predicted tensile strain at BOP indicates values of 7 - 8 mm/m and at seabed of 3 - 5 mm/m. In the case of F20, the direction of working is parallel to the Ninety Fathom Fault with a predicted tensile strain at BOP of 6 mm/m and seabed 3 mm/m. It is also recorded that a feeder of 0.6 m^3/min occurred in the vicinity of a 2 m fault intercepted by the face.

The remaining 16 Main seam dry faces indicate BOP tensile strains of 4 - 6 mm/m and seabed strains of 2 - 4 mm/m.

8.6.4.2 Brass Thill Seam - Wet Faces

Although only four out of 31 faces in the Brass Thill seam are recorded as wet, the K25 to 34 faces are known to have produced considerable amounts of nuisance water. Investigations at the time, concluded that the water rather than coming from an intervening aquifer horizon, originated from reservoirs in the previously worked Main seam F1 - 10 faces. The goaf areas acted as reservoirs for the collection of continuing amounts of nuisance water. The onset of working in the Brass Thill induced fracture zones which allowed drainage of the overlying Main seam goaf reservoirs. This theory is supported by the subsequent drying up of drainage points, installed in the Main seam goafs.

The four wet faces K24 and W1 - 3 are situated on the western boundary of the Brass Thill workings, with a cover to BOP of 115 m and to seabed of 135 m. Predicted tensile strains at BOP indicate values of 6 - 8 mm/m and at seabed 5 - 7 mm/m. However, it should be noted that water encountered on face 3W was associated with a dyke rather than roof droppers/feeders. Quantities varied from 0.9 to 2.3 m^3/min.

The 27 remaining Brass Thill dry faces indicate BOP tensile strains of 3 - 7 mm/m and seabed strains of 2 - 5 mm/m.

8.6.4.3 Total Predicted Tensile Strain

Total predicted tensile strains for the interaction of parallel faces in the same and adjacent seams, indicate that for the Main F1 - 10, Maudlin H2 - 6 and Brass Thill K25 - 37 series faces, maximum strains of 14 mm/m at BOP and 9 mm/m at seabed occurred. This suggests that in the case of multi-seam workings a total predicted BOP tensile strain of up to 14 mm/m can be induced without the incidence of water.

8.6.5 Geological-Strain-Water Relationship

A total of 82% of the faces in the Westoe workings have remained dry and in the case of multi-seam extraction up to 14 mm/m of tensile strain can be induced at BOP without the incidence of water. It is therefore proposed that the variable nature of the intervening Coal Measures strata allows the accommodation of induced tensile strains while still retaining its impermeable horizons. The remaining wet faces can be divided into two types, which are discussed separately.

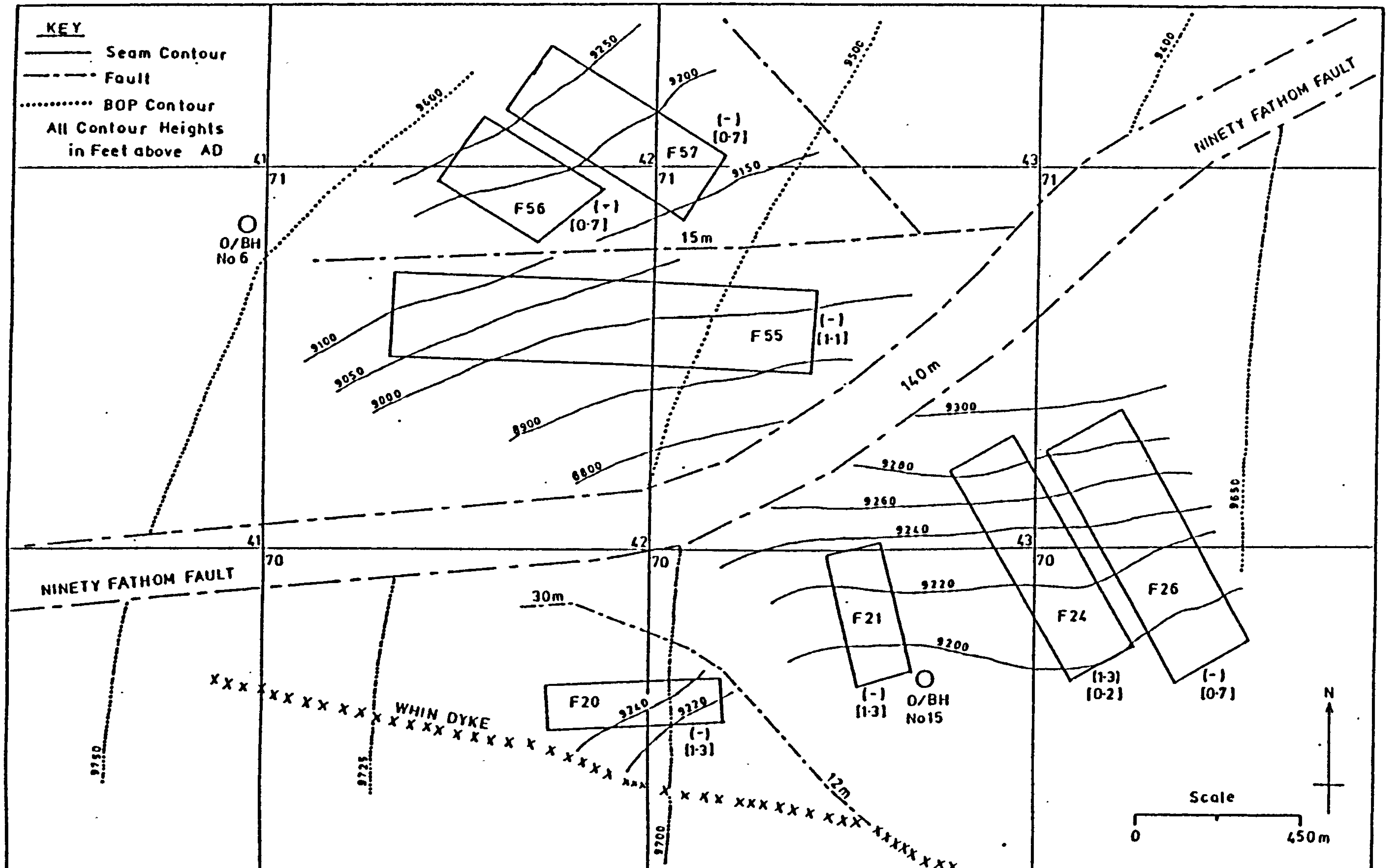
8.6.5.1 Proximity to the Ninety Fathom Fault

All Main seam faces worked in the vicinity of the Ninety Fathom Fault have experienced water and with the exception of F20, the direction of working has been perpendicular to it, Figure 8.19. A combination of three mechanisms can explain the occurrence of this water:

1) recharge across the fault from the saturated 'D' seam sandstones in the north to the overlying strata of the Main seam in the south.

2) recharge along the fault plane, either from the Permian or some intervening aquiferous horizon.

3) the existence of secondary structural features associated with the Ninety Fathom Fault, Sherbourne Hills (99).



( ) MAXIMUM WATER YIELD m^3/min [ ] RESIDUAL WATER YIELD m^3/min

Figure 8.19 Plan showing the 7 Main (F1) Seam Wet Faces, Westoe Colliery

The superimposition of a tensile strain zone onto any or all of these mechanisms will then provide an access route for water to the working horizon.

In the Main seam, the formation of tensile zones around the wet faces F55, F56 and F57 will have aggravated any secondary structural features associated with the Ninety Fathom Fault. This in turn will have aided the passage of water from the saturated 'D' seam sandstones and/or fault plane into the workings. On the central area wet faces, F20, F21, F24 and F26, the tensile strain zones may again have aggravated secondary structural features, which aided the passage of water from overlying secondary aquifer horizons. Recharge of water across the Ninety Fathom Fault from the 'D' seam sandstones, will have formed secondary aquifer horizons in the strata overlying the Main seam, during geological time.

At the St. Hilda Fault, where faces have been worked in close proximity but with no occurrence of water, two significant features do not occur here, which do exist at the Ninety Fathom Fault. Firstly, the fault increases in throw westwards, so the number and magnitude of any secondary structural features decreases eastwards towards the workings. Secondly, no potential aquifer horizons appear to have been displaced into close proximity with the working horizons, allowing recharge to form secondary aquifers in the overlying strata, during geological time.

Finally, all Maudlin and Brass Thill faces worked in close proximity to the Ninety Fathom Fault, have remained dry and this is due to two main reasons. Firstly, all water transmitted along the fault plane has been intercepted by the Main seam. Secondly, no saturated strata horizons exist between the Brass Thill and Main seam, to provide a recharge medium comparable with the 'D' seam sandstones.

8.6.5.2 Proximity to the River Tyne Buried River Channel

The Brass Thill faces K24 and W1 - 3 have all experienced water in quantity, even though no adjacent seam working has occurred. However, a buried channel of the River Tyne is known to cross the area with a possible depth of up to 60 m below seabed. A depth of cover of 75 m or less of Coal Measures strata may exist above these workings and consequently the predicted tensile strain at BOP of 6 - 8 mm/m will be too low.

A combination of shallow cover of unknown height and lithology, coupled with minor insitu structural discontinuities have therefore led to the occurrence of water with the onset of tensile zone formation. In the case of face 3W, water was associated with the interception of a dyke, rather than roof feeders/droppers.

8.7 Dawdon Colliery

8.7.1 Mine Development

Coal production is currently by longwall extraction in three seams; the High Main (E), Yard (G) and 'C' seam which is currently being developed. The Maudlin (H), Low Main (J) and Hutton (L) have also been historically worked by total extraction and longwall techniques. Some extraction has also occurred in the Main (F) seam.

The seams have been developed progressively eastwards, Figures 8.20 - 8.23. Development has occurred off the main arterial roadways in a generalised North-West to South-East direction.

8.7.2 Mine Geology

Figure 8.24 shows the typical Coal Measures sequences encountered in the Dawdon workings based on the shaft section. The Coal Measures consist of typical cyclotherm sequences of interbedded coal/seatearth, mudstone, siltstone and sandstone. The general seam intervals are given in Table 8.5.

	(m)	(m)
Base of Permian	0	0
Seam 'C'	146	146
High Main Seam 'E'	82	228
Main Seam 'F'	30	258
Yard Seam 'G'	15	273
Maudlin Seam 'H'	24	297
Low Main Seam 'J'	21	318
Hutton Seam 'L'	33	351

Table 8.5 General Seam Intervals, Dawdon Colliery

No major faults bisect the workings, although the northern boundary, separating the Dawdon and Vane Tempest Collieries, coincides with the Seaham Fault. The Seaham Fault is a major fault which belongs to the regional pattern, has a throw of approximately 240 m and comprises a fault zone rather than a well defined fault plane. The downthrow side lies to the north.

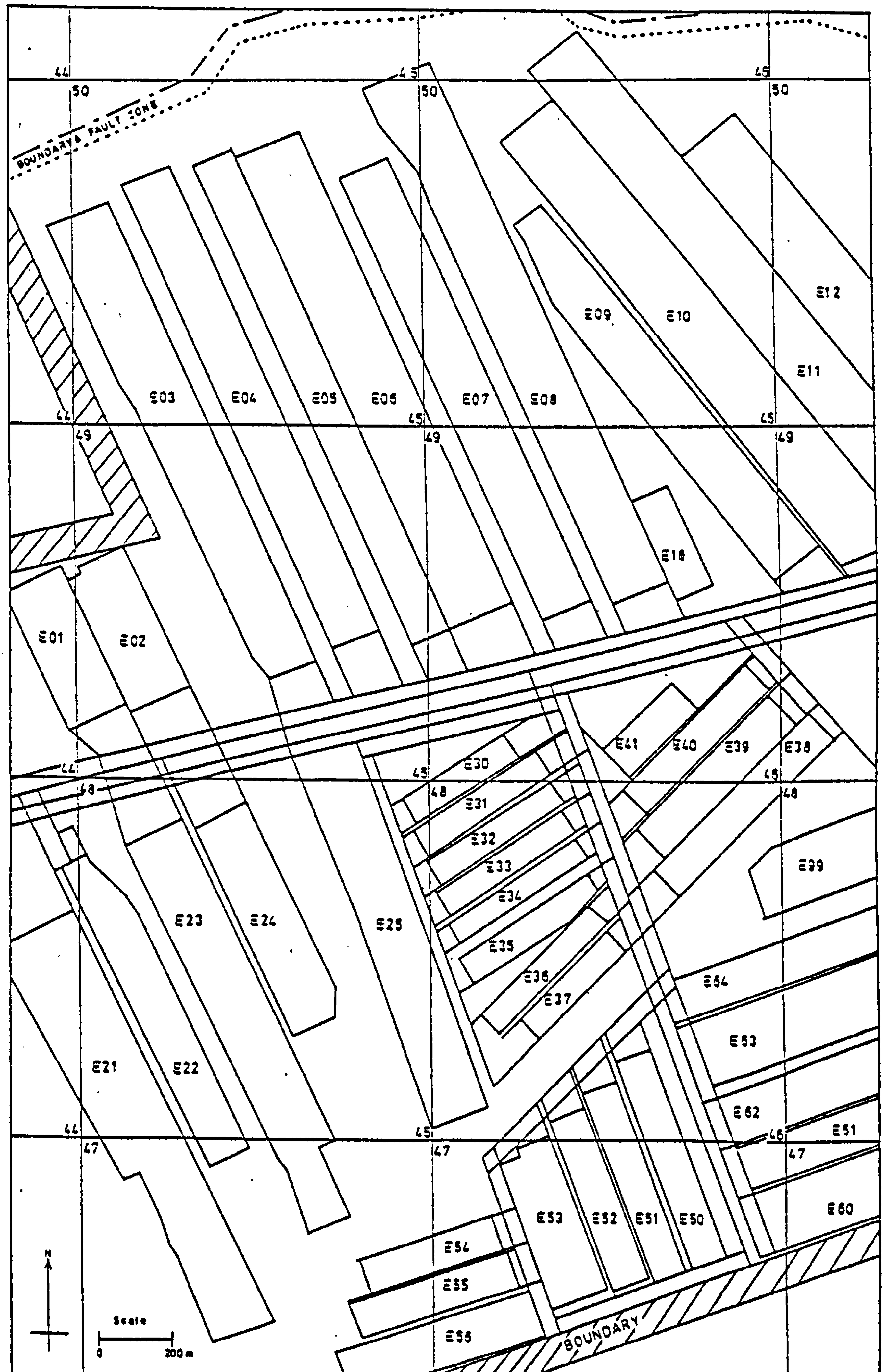


Figure 8.20 Western Area, High Main (E) Seam, Dawdon Colliery

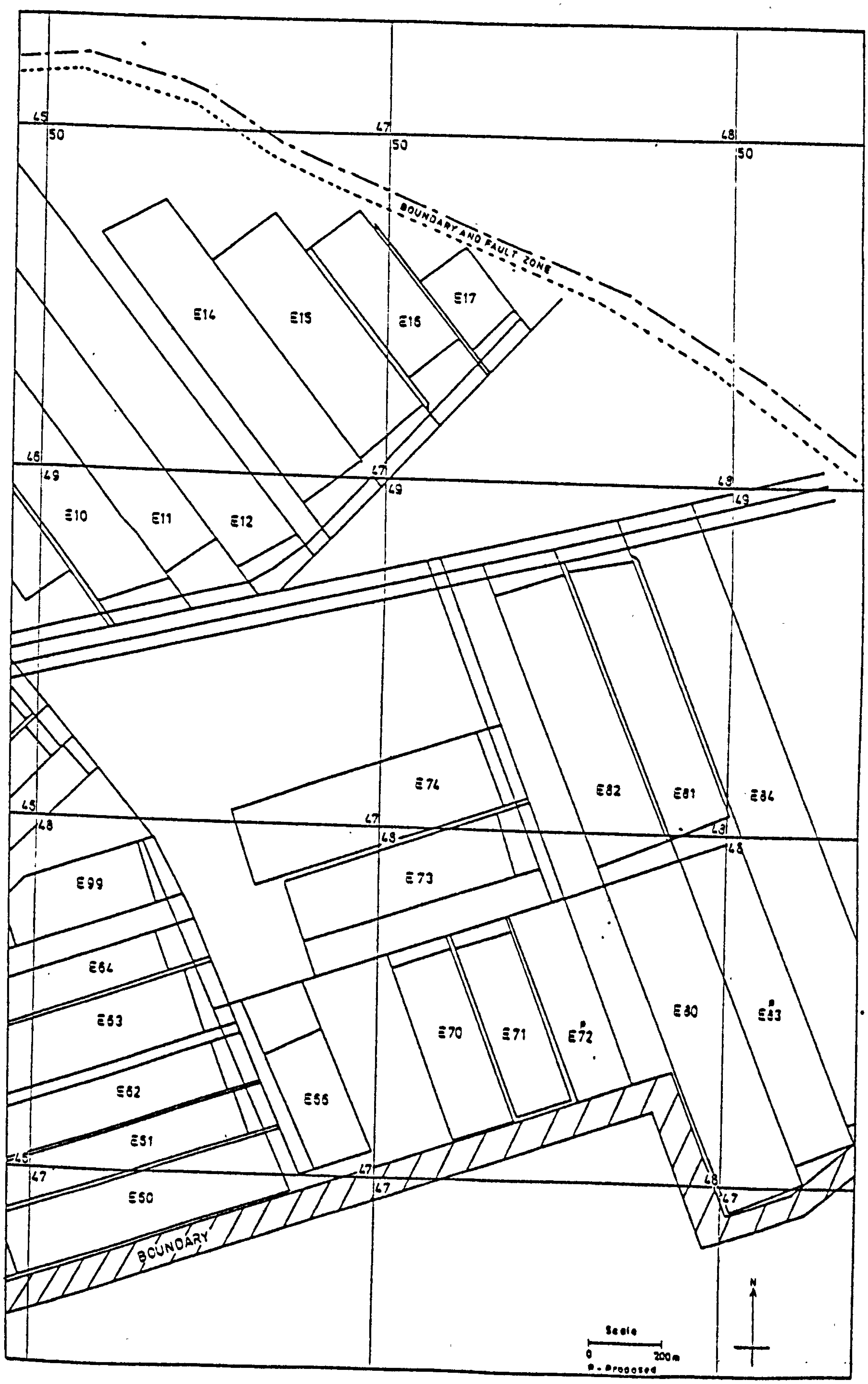


Figure 8.21 Eastern Area, High Main (E) Seam, Dawdon Colliery

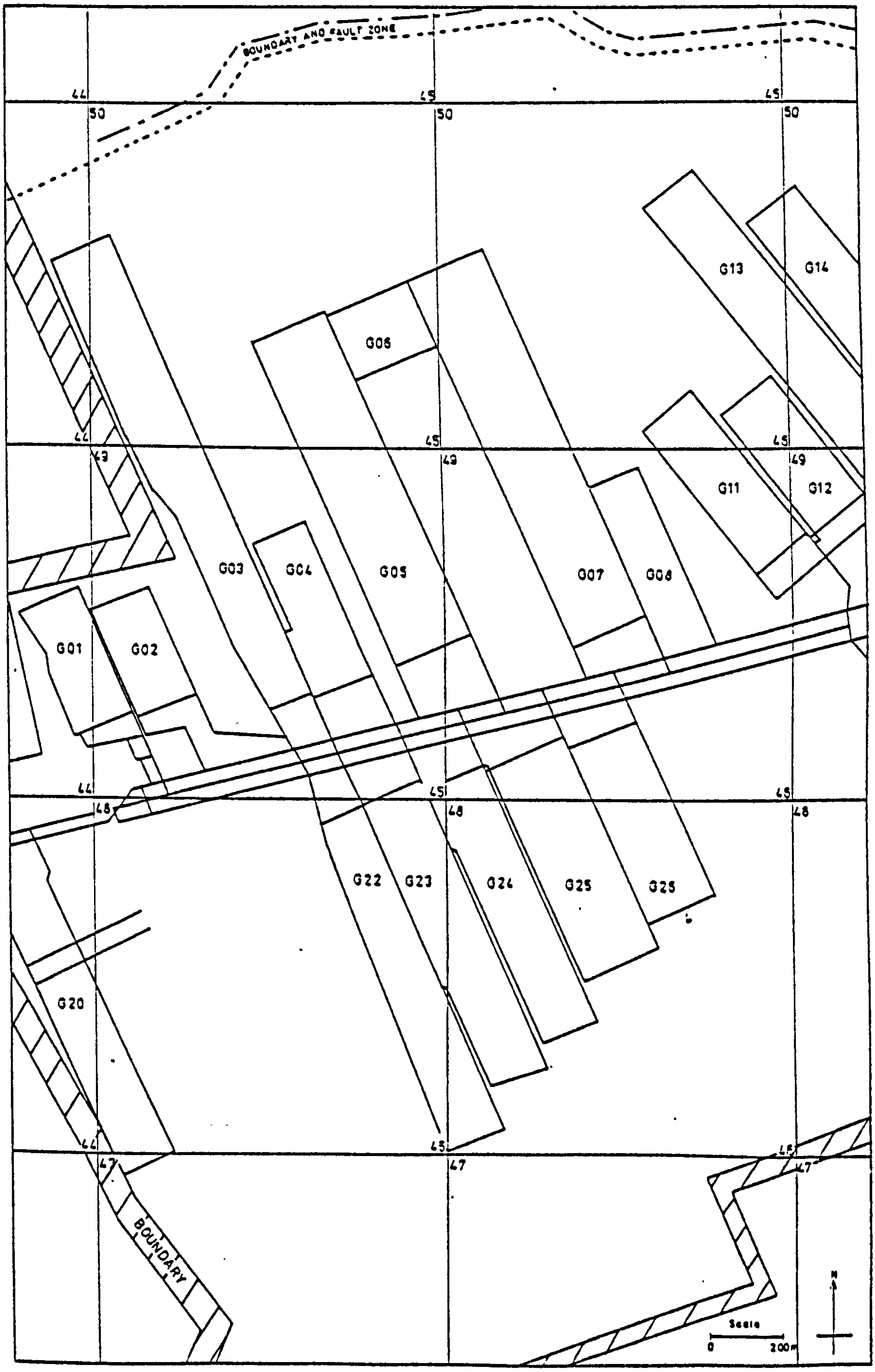


Figure 8.22 Western Area, Yard (G) Seam, Dawdon Colliery

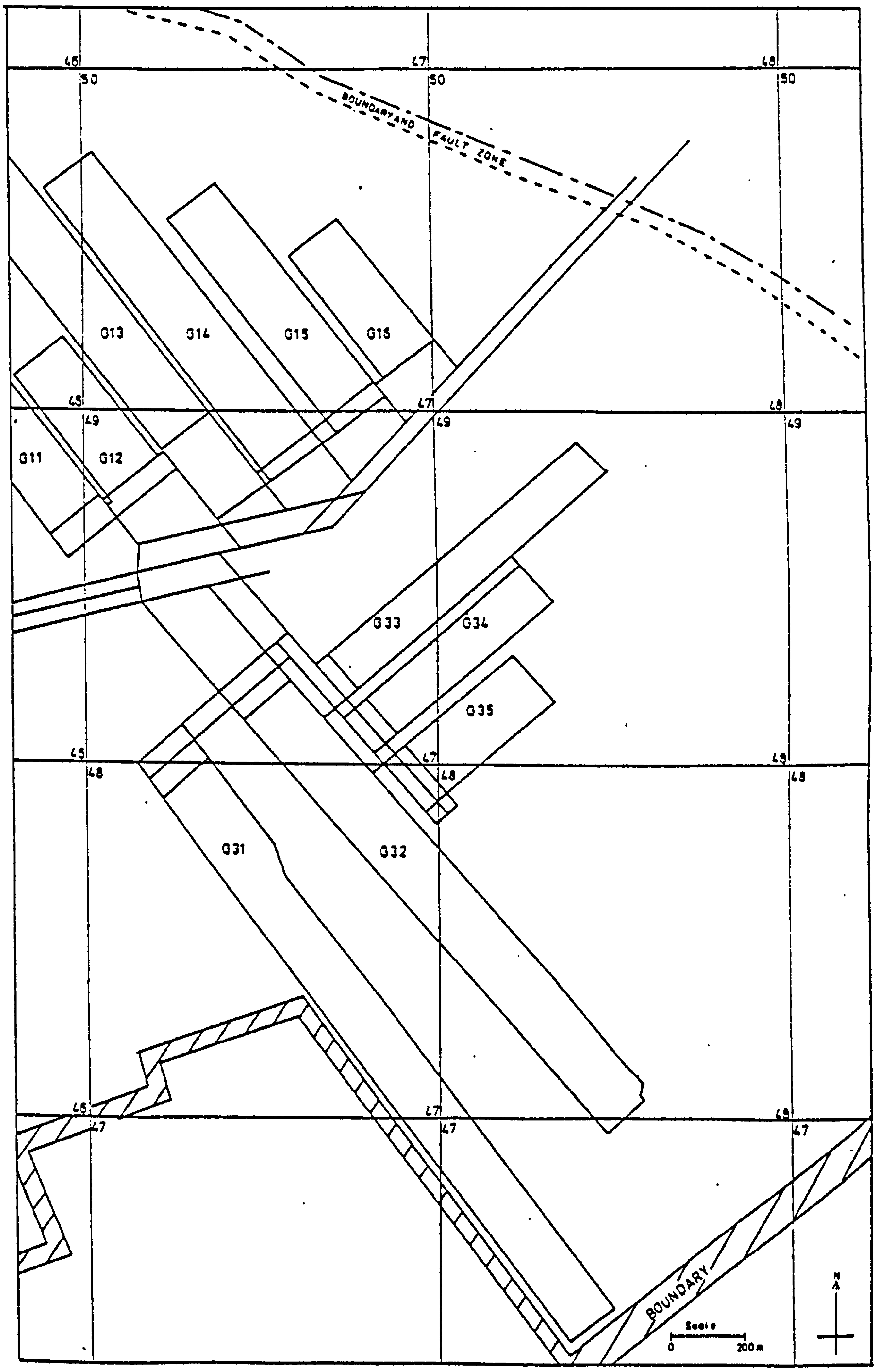


Figure 8.23 Eastern Area, Yard (G) Seam, Dawdon Colliery

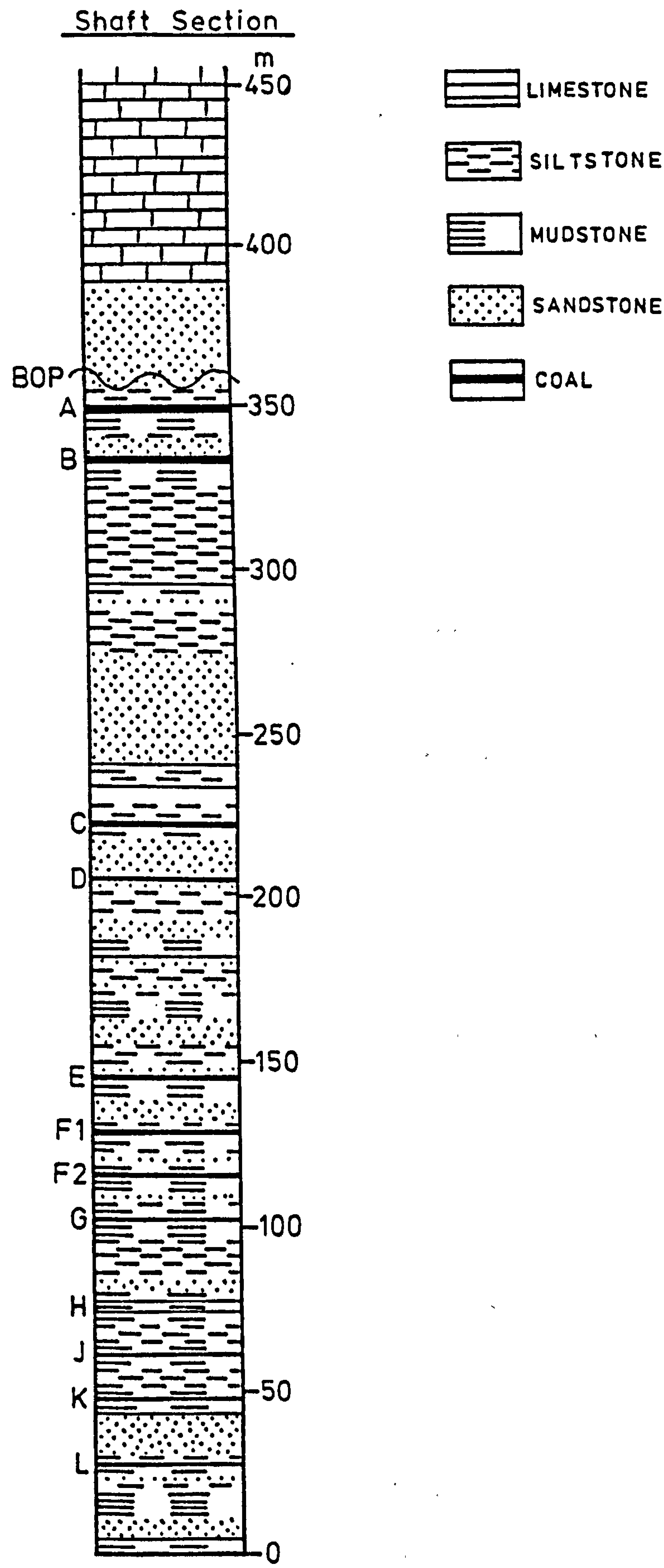


Figure 8.24 Geological Section representative of Coal Measures Strata at Dawdon Colliery

Minor faulting is encountered in the workings, but seldom exceeds 5 m in magnitude. Two main trends exist, North-East to South-West. No dyke structures have been encountered in the workings.

8.7.3 Water Occurrence

Dawdon Colliery is currently pumping about 0.5 m^3/min from the total area of workings and is normally considered to be 'dry'. No water occurrences have been experienced, even though extensive longwall extraction has occurred. Nuisance water has been encountered on some faces and in some seams, but has usually dried up within a few weeks. The water occurs as either small feeders or droppers occurring along the face line or within the goaf. There appears to be a tendency for water to be encountered around the time of first break, although it is seldom encountered again.

Nuisance water has been encountered principally in the High Main (E) seam. In the Main (F) seam, limited workings have encountered water, which is thought to have originated from ponded areas in the overlying High Main. Similarly, very small quantities of water were encountered in the Low Main (J) and Hutton (L) seams, oozing from the floor. The Yard (G) and Maudlin (H) seams are dry, even by Dawdon standards.

8.7.4 Water Occurrences and Predicted Subsidence

Of 80 faces in the Dawdon workings examined, none have experienced water occurrences, except in the form of temporary nuisance feeders.

8.7.4.1 Total Predicted Tensile Strain

Total predicted tensile strains for the interaction of parallel faces in the same and adjacent seams, indicate that maximum strains of between 5 - 7 mm/m at BOP and 3 - 5 mm/m at seabed have occurred.

8.7.5 Geological-Strain-Water Relationship

No longwall faces in any of the Dawdon seams have experienced significant water feeders. Therefore, predicted tensile strains at the base of Permian (BOP) of 5 - 7 mm/m and at seabed of 3 - 5 mm/m cannot be related to the incidence of water.

Two significant features are found in the Dawdon workings, which are not found at the Blackhall, Horden or Westoe Collieries. Firstly, the amount of cover between the base of Permian and the worked seams is greater than at any of the other examined collieries. Over 200 m of Coal Measures strata separates the uppermost worked seam from the Permian. A large amount of cover therefore exists which can accommodate the induced strains and provide impermeable barriers against the access of water.

Secondly, the workings are not bisected by large faults which have displaced potential aquifer horizons into close proximity with the working horizon, allowing secondary aquifer systems to form over geological time. At Dawdon, the northern boundary constitutes a major fault zone in the vicinity of which a considerable number of faces have terminated. However, no water occurrences have been experienced and two reasons can be put forward for this phenomena:

1) the faces have terminated short of the fault, so that induced longitudinal tensile strains have not interacted with the fault zone.

2) because of the depth of working, only small quantities of water have penetrated the fault zone and no potential aquifer horizons have been displaced into close proximity with the current workings.

8.7.5.1 Potential 'C' Seam Working

Development in the 'C' seam is currently occurring, where the amount of cover to the base of Permian is reduced to about 150 m. Examination of the available information suggests that well developed sandstone sequences exist between the 'B' and 'C' seam horizons as well as around the underlying 'D' seam horizon. The Seaham Fault which has a throw of 240 m, is thought to have displaced Permian strata in the north against Coal Measures strata in the south. A potential for creating secondary aquifer systems in the sandstones between the 'B' and 'C' seam horizons therefore exists.

It is thought that workings in the 'C' seam, which are kept away from the Seaham Fault, should only experience water in nuisance quantities, even if the 1968 guidelines were to be exceeded. However, when faces are worked in the vicinity of the fault, a much higher risk of water occurrences will exist. This is thought to be due primarily to the influence of tensile strain on potential aquifer horizons formed in the sandstone sequences between the 'B' and 'C' seam horizons, rather than on the base of Permian or the fault zone.

8.8 Discussion of Results

Normal statistical techniques have failed to identify any single parameter which controls the occurrence of water on a longwall face. Table 8.6 shows correlation coefficients for water occurrences in the Blackhall workings. The analysed data includes both wet and dry faces as well as those north and south of the major fault. A potential correlation appears to exist between maximum yield and face width (0.61), although the current yield and face width shows a much lower coefficient (0.42).

At the three wet collieries, some exploratory and drainage boreholes have failed to encounter water in any quantity, from potential aquifer horizons in the overlying strata. However, subsequent panel extraction has resulted in significant quantities of water being yielded. It is therefore proposed that potential aquifer horizons only dissipate their stored water once the strata has been disrupted.

Examination of data from the four collieries studied has revealed that water is encountered whenever,

'faces are worked in the vicinity of a major fault, where potential aquifer horizons (the Permian) have been displaced against Coal Measures strata overlying the working area'.

At each of the wet collieries, water was encountered in workings of the top most seam, on faces worked in the vicinity of a fault with the aforementioned conditions. Faces in underlying seams, which have also been worked in close proximity to the same type of fault have seldom encountered water in more than nuisance quantities.

	Maximum Yield (m^3/min)	Current Yield (m^3/min)
Maximum Yield (m^3/min)	1.00	0.78
Current Yield (m^3/min)	0.78	1.00
Height to Seabed (m)	0.04	0.10
Height to BOP (m)	0.01	-0.03
Panel Width (m)	0.61	0.42
Seam Thickness (m)	0.05	0.16
Seabed Width-Depth Ratio (w/h)	0.35	0.43
BOP Width-Depth Ratio (w/h)	0.53	0.33
Max. Tensile Strain at BOP (mm/m)	0.51	0.35
Max. Subsidence at BOP (mm/m)	0.57	0.40

BOP - Base of Permian

Table 8.6 Correlation Coefficients for Wet and Dry Faces in the North and South Area Workings at Blackhall Colliery

Figure 8.25 shows minimum distances between the fault and working panel against the maximum and current/residual yields on wet faces, at each of the three collieries. In general, water is encountered on faces which have been worked closer than 350 m to the fault. No evidence exists to suggest that 'the closer the fault, the greater the amount of water liberated to the face'. Similarly, a well developed sandstone horizon has been present in the strata sequence overlying the wet seam.

At Dawdon Colliery, a thick sequence of Coal Measures lies between the High Main (the top most worked seam) and the base of Permian. No aquifer horizons have been displaced into close proximity with the overlying strata, by the Seaham Fault. However, development is currently occurring in the 'C' seam, which is shallower and also contains a well developed sandstone horizon in the overlying Coal Measures sequence. The Seaham Fault might have displaced aquifer horizons (the Permian) against these sandstones, which has resulted in the formation of secondary aquifers over geological time. Therefore, comparing conditions in the Dawdon 'C' seam with those at Blackhall, Horden and Westoe, it is thought that water occurrences will be experienced on those faces worked in the vicinity of the Seaham Fault. Figure 8.25 indicates that provided a barrier pillar of 350 m is left around the fault zone, then the risk of water occurrences within the 'C' seam will be greatly reduced. The original guideline for pillars of one tenth depth appears inadequate for controlling the influx of water from secondary aquifer systems associated with this type of fault.

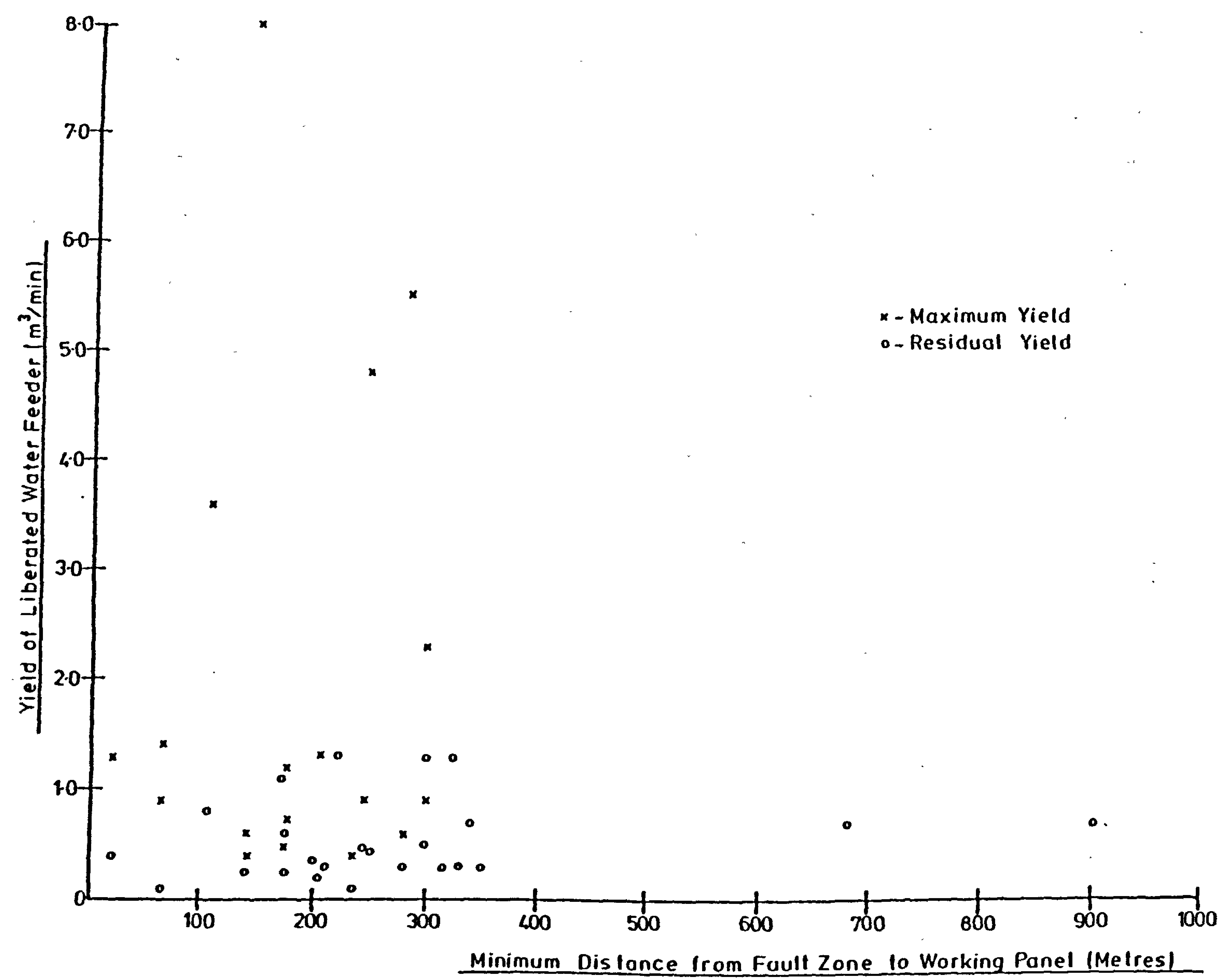


Figure 8.25 Minimum Distance of all Wet Panels from Fault Zone against Quantity of Water Experienced

No evidence was found during the investigation that the magnitude of tensile strain induced at the base of Permian affected the incidence of water on faces. At Horden and Westoe, multi-seam workings induced predicted tensile strains of up to 12 mm/m and 14 mm/m respectively, at BOP without influencing the occurrence of water in workings of the top most seam. It is therefore concluded that away from major fault zones, the predicted tensile strain of 10 mm/m can be exceeded without the increased likelihood of water.

In the vicinity of major faults, it is not apparent whether tensile strain or properties of the secondary aquifer systems control the amount of water yielded. However, at Blackhall and Horden, narrow faces and associated low tensile strains have resulted in water quantities as large as any of those experienced on wider faces. It is therefore concluded that properties of the secondary aquifers and intervening strata probably exert significant influence over the potential water yield.

8.9 Conclusions

Normal statistical techniques have failed to identify factors controlling the occurrence of water on longwall faces in the Durham coastal coalfield.

An assessment of geological, hydrogeological and induced tensile strains at four collieries has concluded that:

1) at all the collieries examined, faces have experienced water in the vicinity of a major fault, where aquifer horizons have been displaced against Coal Measures strata overlying the working horizon.

2) in underlying seams where aquifer horizons have not been displaced (by the major fault) into the vicinity of a working horizon, only occasional nuisance quantities of water are encountered.

3) away from fault zones, water is occasionally experienced in areas where the amount of cover to the base of Permian thins appreciably and/or a well developed sandstone horizon exists within the intervening Coal Measures sequence.

4) where neither conditions 1 or 3 exist, it appears that the incidence of water is unrelated to the 1968 guideline (2) value of 10 mm/m tensile strain at either seabed or the base of Permian. Up to 14 mm/m predicted tensile strain has been induced at the base of Permian, without the occurrence of water in the uppermost workings.

Finally, provided it is realised that an increased risk exists in the vicinity of major faults, there is no reason why the 10 mm/m guideline should not be relaxed. Current evidence suggests that the value could be raised to 15 mm/m without influencing the occurrence of water on a working horizon. Similarly, an increase to 20 mm/m could also be made for deeper working horizons. However, before a general increase to 20 mm/m was undertaken, a series of trials should be conducted on selected panels and the results evaluated.

CHAPTER 9

GENERAL CONCLUSIONS AND RECOMMENDATIONS

CHAPTER 9

GENERAL CONCLUSIONS AND RECOMMENDATIONS

9.1 Aspects of Mine Water Abstraction

In the future, amounts of water pumped from British coal-mines will increase. Primarily, this will be due to closure of uneconomical mines at which pumping has to be continued in order to prevent the flooding of adjacent workings. However, the potential yield from overlying aquifer systems is also expected to increase. At present, there appears to be a tendency for both individual mines and areas, which have water problems, to simply install either larger or more pumps, as further water is yielded to the workings. Although this solves the problem in the short term, it does not identify the cause, which may possibly have been avoided by a simple change in mine design or working practice.

It is proposed that a detailed study should be undertaken to determine the direct cost of mine water pumping per annum throughout British coalfields. Indirect expenditure resulting from corrosion, downtime and maintenance, although difficult to quantify, also represent hidden, but accepted costs. Such a study would probably reveal that several million pounds per annum are spent on mine water pumping and related problems. If a research programme was initiated, which in only 5 years resulted in a one percent drop in pumping costs, then not only would it have paid for itself, but it would also have made a profit.

Finally, environmental concern has led to increased legislation regarding the discharge of industrial effluent to natural water courses. In turn, this has resulted in capital expenditure on treatment plants, to remove potentially hazardous material. It is therefore essential to consider cost-effective schemes, whereby mine water is used for secondary purposes, such as coal washing, before final treatment and discharge.

9.2 Analytical and Laboratory Techniques

Historically, analytical techniques have been developed for the evaluation of well test data. However, in the majority of cases, Darcian conditions are assumed to exist around the well or test cavity. Such conditions are virtually unknown in the Coal Measures and seldom exist in nature. Darcian conditions do not strictly apply when a significant proportion of fracture permeability exists at a test horizon. Dynamic conditions around longwall panels develop an increased proportion of fracture permeability as the face approaches and passes the site. Normal analytical techniques cannot therefore be used to derive absolute permeability values for dynamic conditions. However, the nature of fracture permeability flow regimes are still far from being understood and current non-Darcian theories are both complex and require the insertion of assumed variables.

Recently, a technique derived by Barker (43) has enabled aquifer parameters to be calculated for a test cavity intersected by a single large fissure. The application of this technique to Coal Measures strata, where a test cavity is assumed to be intersected by numerous small fissures, therefore requires further investigation.

A dilemma exists between the use of various analytical techniques and the interpretation of results derived from a single set of field data. Ideally, as many different techniques as possible, should be used and the results evaluated on the basis of site conditions and the interpreter's experience.

Finally, a frequent review of current literature should be made to keep the Department abreast of developments at other establishments. Existing expertise and facilities can then be applied to assessing the potential application, of any developments, to the Coal Measures and mining related problems.

9.2.1 Laboratory Techniques

A considerable amount of work has been undertaken into the laboratory determination of rock permeability, notably Hsieh et al (102). However, there is still a need for the development of a cheap and simple technique which will allow the determination of permeability changes associated with increased stress levels within a specimen. Existing techniques are usually complex to operate and require large capital expenditure on equipment and resources.

With these considerations in mind, the author has derived an extremely simple and cheap technique for determining the permeability of a core specimen, Appendix C. In addition, the specimen can be pre-stressed to allow the determination of permeability changes, associated with increased stress levels.

If a series of tests are undertaken, where permeability is monitored against increasing strain on a specimen, data can be accumulated for insertion into models which determine strata permeability changes around a longwall panel. However, it should be noted that this technique is still very much in its infancy and requires considerable development and evaluation, which it is hoped will occur over the next few years within the Department.

9.3 Mining Subsidence

It is normally assumed that methods in the Subsidence Engineers Handbook (3) can be extrapolated to derive subsidence profiles for strata-strata as well as strata-surface horizons. However, when profiles were calculated for very shallow horizons, width-depth ratios of greater than 5.0, a lack of reliable data was found for even strata-surface horizons. An examination of predicted and actual profiles for case histories within the East Midlands was made, to determine whether an acceptable correlation existed. Unfortunately, a lack of data produced inconclusive results. Similar studies to assess the effect of surface geology on profile and fissure formation, again produced only tentative correlations. It is therefore proposed that supplementary data should be obtained from coalfields throughout the United Kingdom. Detailed analysis should then be undertaken and in the case of shallow working conditions, the Subsidence Engineers Handbook revised if necessary.

A shortage of data was also found concerning the development of strains around an advancing longwall panel. Figure 9.1, shows that subsidence and strain are usually monitored by a single

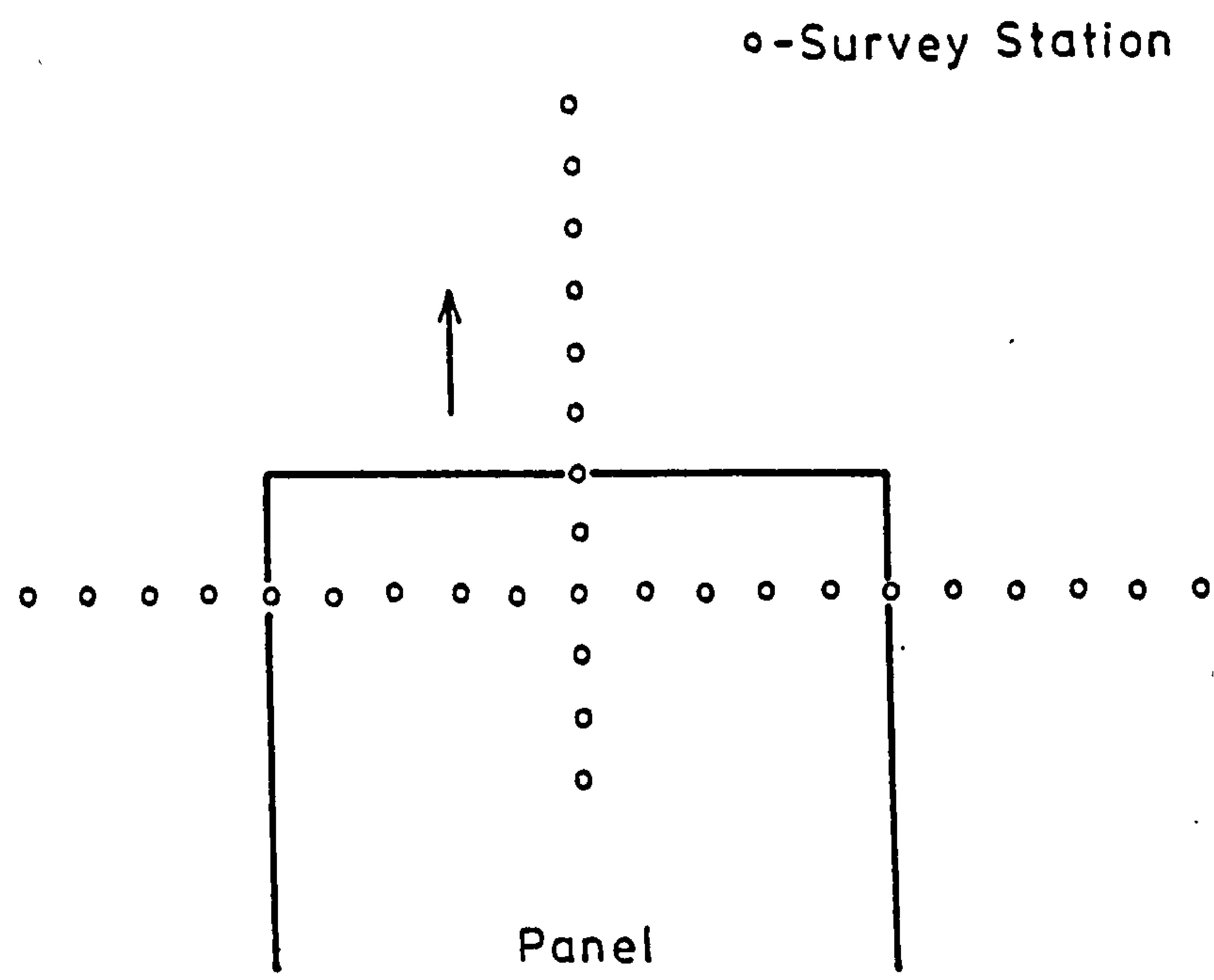


Figure 9.1 Current Field Instrumentation Scheme for Monitoring Subsidence Profile Formation

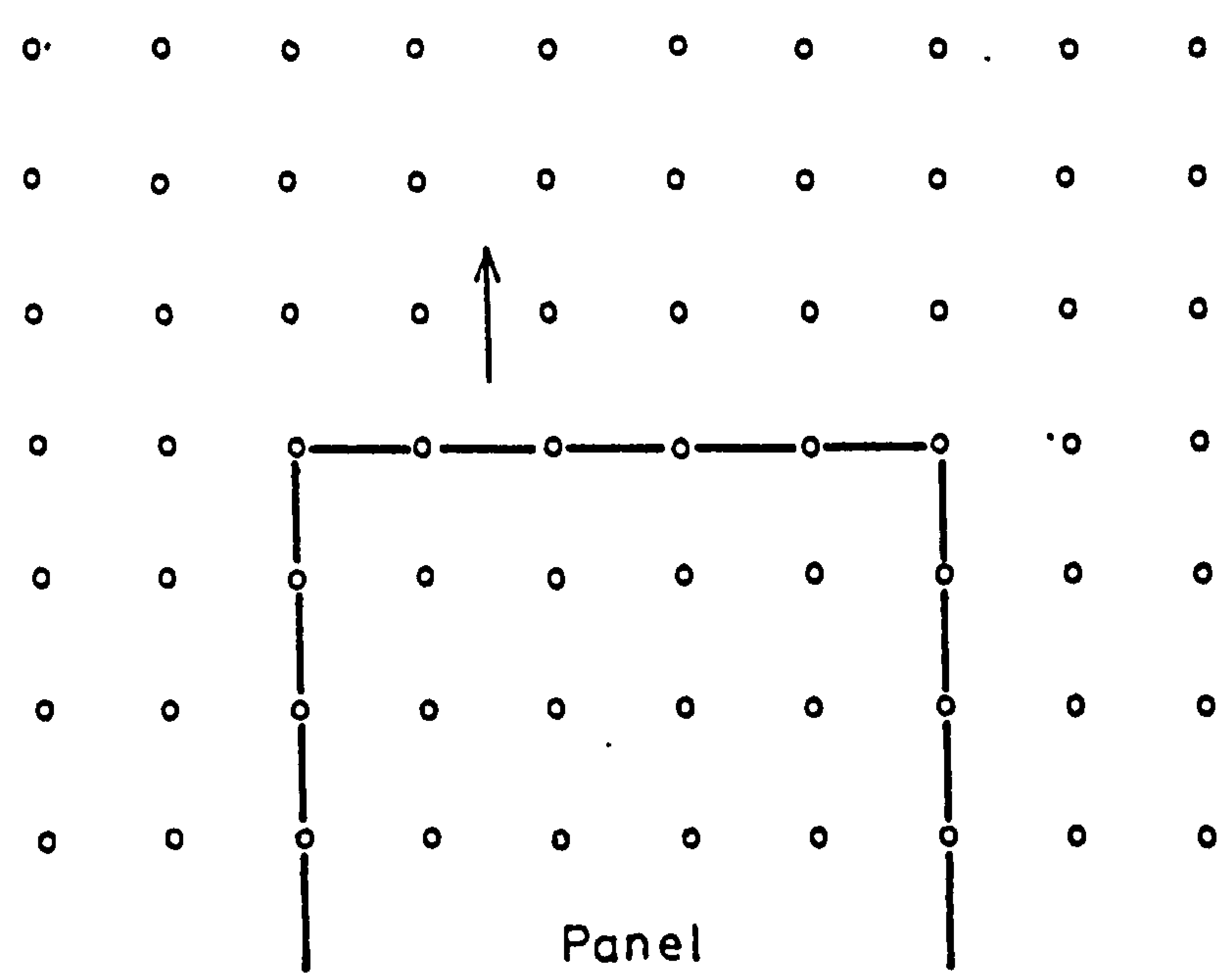


Figure 9.2 Proposed Field Instrumentation Scheme for Monitoring Subsidence Profile Formation

transverse and longitudinal survey line across the panel. Contours are then drawn between points of equal strain and subsidence around the edge of the panel. A lack of reliable data around the face-end areas has highlighted problems with the development of computer modelling techniques and the calibration of simulations. A comprehensive scheme is therefore proposed to monitor profile development in this region, Figure 9.2. A network of monitoring stations are used, which form a series of survey lines both transversely and longitudinally across the panel. Field instrumentation of this type is normally time consuming and expensive. However, recent advances in Electro-Magnetic Distance Measuring techniques and Automated Survey stations should make this type of scheme a feasible proposition.

Finally, computerised subsidence simulation and modelling techniques are under development by the Department. Work is currently progressing on the meshing of subsidence-strain profiles for adjacent workings in the same seam and it is hoped to extend this to multi-seam conditions.

9.4 Permeability Monitoring

Extensive work by the Department has concluded that changes in strata permeability ahead of a longwall face can be linked to the ground strains formed during subsidence profile formation. However, insufficient data exists to either quantify or fully comprehend the exact nature of such phenomena and either confirm or dispute the conceptual models proposed for such events. Data from a great many sites is still required before permeability changes can be predicted by empirical methods similar to those in the Subsidence Engineers Handbook (3).

Development of continuous monitoring equipment would in part overcome the lack of data imposed by insufficient sites. All boreholes should contain an instrumentation scheme to maximise the monitoring of induced longitudinal and transverse tensile strains. Similarly, from previous experience, it is important to assess the likelihood of a site being completely undermined.

Prediction of longitudinal tensile strain profiles by the Subsidence Engineers Handbook (3) and East Midlands Subsidence Unit methods, has found that the onset distances are considerably less than those at which permeability changes begin to occur. Therefore, either the current prediction methods need revision or another mechanism is causing the onset of change in permeability. Alternatively, current instrumentation is not sufficiently sensitive to monitor the very low induced strains which cause permeability change. If more sites are monitored, experience and the assessment of previous results, should allow the installation of instrumentation which will resolve these discrepancies.

Finally, five sites have monitored permeability changes around an advancing longwall face. Sufficient data should therefore exist to start modelling changes in strata permeability, using numerical methods.

9.4.1 Future Test Site Areas

No work has been initiated to monitor permeability changes occurring around steep seam workings, which are overlain by major aquifer systems. In steep seam conditions, an imbalance of strain develops across the profile, between the dip and rise side of the extraction. Exactly how this strain imbalance affects the overlying strata permeability is not known. However, it is an area of investigation which is considered sufficiently important to warrant prompt action, since at Seafield Colliery, Scottish Area, seams with gradients of up to 1 in 1.5 are being worked beneath the Firth of Forth.

Another area which also requires investigation is the change in permeability which can be associated with strata beneath the seam. At Dawdon Colliery, water is recorded oozing from the floor of several seams, although no reason for its presence could be ascertained. No instrumentation schemes have so far included test cavities to rectify this lack of knowledge. However, the author did propose a scheme at Whitwick Colliery, Section 5.3.2, which would have included this region had the site been successful.

9.5 Strata Control and Geological Considerations

Instrumentation techniques which involve monitoring roadway deformation and the installation of extensometers can be sufficient for determining strata behaviour around a longwall panel. However, work at Hickleton Colliery, Chapters 5, 6 and 7, has revealed that they cannot give a complete picture of what is occurring. The investigation should be supplemented by additional instrumentation, such as:

1) Permeability Monitoring
2) Dye-tracing Techniques
3) Borehole Orientation and Deviation
4) Geophysical Logging
5) Cored Boreholes
6) Drill Cutting Examination.

Detailed geological and geotechnical assessments are also essential for a comprehensive understanding of the problem and its ultimate solution. A cored borehole should be included in every scheme and the core evaluated by laboratory techniques. Preliminary on-site evaluation can be undertaken using mechanical logging methods, although certain limitations do exist. Finally, permeability monitoring can be used as a highly sensitive, alternative technique for assessing the caving characteristics of strata overlying a longwall panel.

9.5.1 Provision of Borehole Core

Normally cored boreholes are undertaken by the Area Drilling team. However, one way to overcome this problem would be to supply the colliery methane drillers with a short core barrel. This could then be used in pre-selected boreholes and/or certain strata horizons to provide core for a variety of uses.

The indiscriminate coring of all underground boreholes is considered impractical and wasteful in terms of money, time and resources. However, coring at pre-selected sites and strata horizons would provide extremely valuable information. Finally, in addition to providing core for geotechnical assessment it could also be used to monitor the coal quality of adjacent seams, with a view to potential working.

9.5.2 In-Seam Drilling Techniques

In-seam stress determination using drilling techniques to determine potential bump conditions, has for many years been successfully used in West Germany. Its potential application to British mining conditions has clearly not yet been recognised. A series of field trials should therefore be initiated as soon as possible, in order to assess the technique and establish its respectability as a method for monitoring high in-seam stress concentrations.

9.6 Undersea Coalfield Workings

Work undertaken in the Durham coastal coalfield has concluded that water principally occurs in an area where faces are worked in the vicinity of a major fault, and potential aquifer horizons (the Permian) have been displaced against Coal Measures strata overlying the working horizon.

In all the wet seams examined where this type of condition occurred, a well developed sandstone horizon was also found in the overlying Coal Measures strata. It is thought that over geological time, recharge occurs from the major aquifer across the fault zone, forming secondary aquifer systems in the adjacent sandstone horizons. Subsequent disruption of the secondary aquifer system by longwall mining releases the water from storage and liberates it to the working panel.

Figure 9.3 shows that all faces worked closer than 350 m to a fault zone (where the necessary conditions exist) have experienced water. Leaving a pillar around the fault zone which has a size based on one-tenth depth, is therefore inadequate to prevent the access of water from the secondary aquifer systems. However, in future workings where such conditions are likely to occur, notably the Dawdon 'C' seam, by leaving a pillar of 350 m around the fault zone, the incidence of water should be significantly reduced.

9.6.1 The N.C.B. 1968 Guidelines

Away from major faults, no evidence can be found to suggest that the incidence of water at a working horizon is related to the magnitude of the induced tensile strain at either the base of Permian or seabed. At the Horden and Westoe Collieries, predicted tensile strains of up to 12 mm/m and 14 mm/m respectively, occurred at the base of Permian, without influencing the potential occurrence of water in workings of the top-most seam. It is therefore concluded that away from major fault zones, predicted tensile strains in excess of 10 mm/m cannot be linked to the increased likelihood of water occurrences.

9.6.2 Hydrochemical Techniques

During the authors visits to the NCB North-East Area, it was found that a large amount of hydrochemical data existed, which

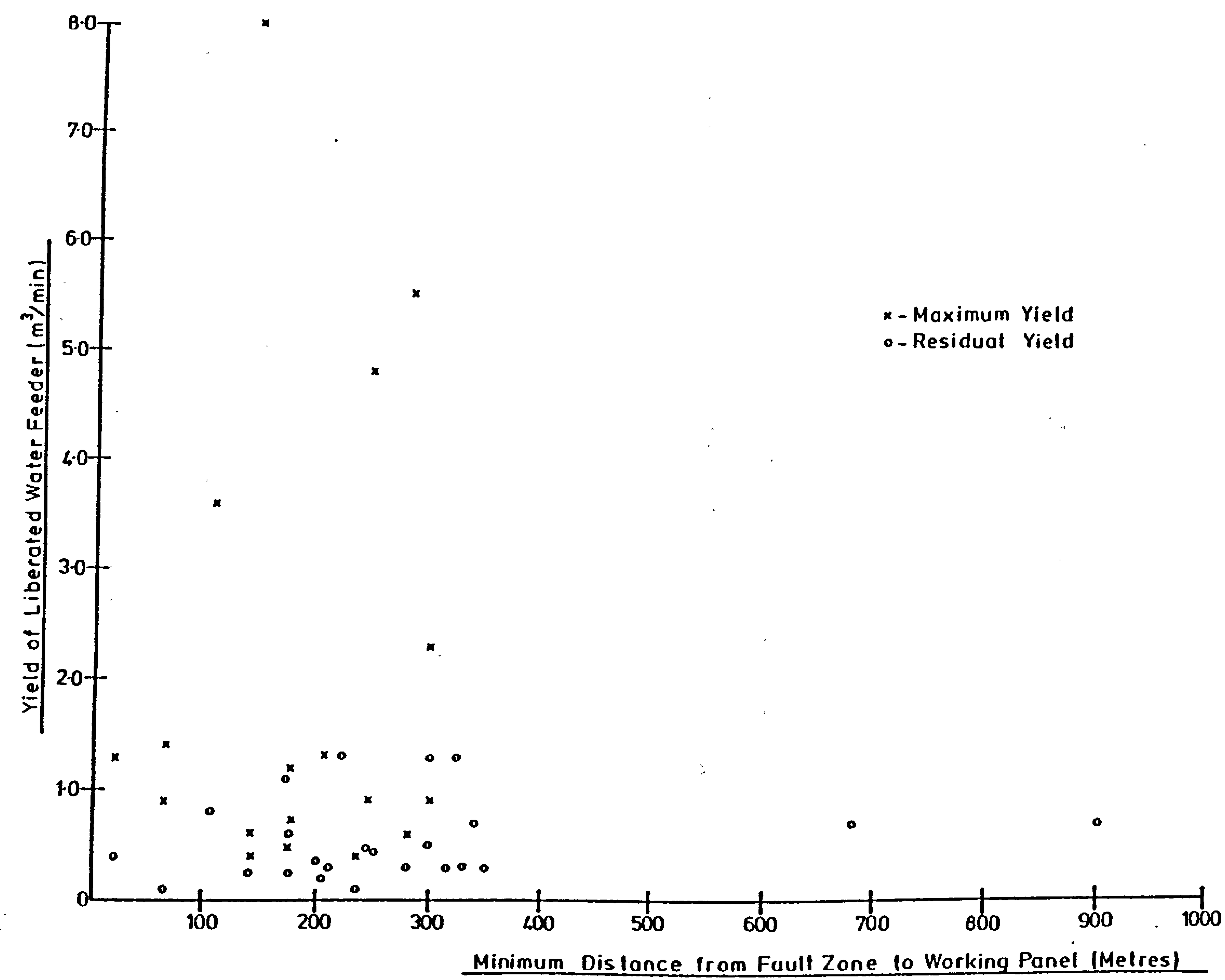


Figure 9.3 Minimum Distance of all Wet Panels from Fault Zone against Quantity of Water Experienced

related to mine water occurrences within the Durham coastal collieries. It has been accumulated over the past 30 years on either a systematic or irregular basis, depending upon conditions prevalent at the time. Only one major analysis of the North-East hydrochemistry has been undertaken, Edmunds (103), although limited NCB in-house analysis has also occurred, Ellis (104).

Edmunds (103) has evaluated the data, by principally using logarithmic cross-plots. However, Zaporozec (105) discusses a wide range of analytical approaches which could be used on the data. The two most widely accepted methods are the Piper and Durvo plots, Piper (106) and Chilingar (107, 108). However, with the increased availability of computer facilities and the use of statistical procedures, recent work by Keet (109) could provide a worthwhile avenue for investigation.

Sufficient data is thought to be available to undertake a comprehensive evaluation of potential source horizons, without initiating additional sampling schemes. The object of the investigation would be to determine a source horizon for the water occurrences experienced and evaluate the localised flow regime around the working horizon. In addition, it would also help to either confirm or dispute the mechanisms proposed in Chapter 8.

Finally, the technique of tritium dating water samples, Langley (110) and Downing et al (111) should be undertaken at carefully selected sites, to provide supplementary data concerning the localised flow regimes around wet seams.

ACKNOWLEDGEMENTS

The author would like to record his thanks to the following:

Department of Mining, Nottingham University.

Professor T. Atkinson for provision of the facilities and his interest in the work.

Dr. B.N. Whittaker for his guidance, advice, practical help and encouragement.

Dr. R.N. Singh for many useful discussions and proof reading this thesis.

Dr. F.P. Hassani for proof reading and commenting on certain chapters.

Mr. S.F. Smith for his comments when proof reading this thesis.

Mr. M. Turney, Mr. V. Cassapi and the departmental technical staff for their help with the design and construction of test equipment.

To the other post-graduates for their advice, criticism and general good humour. In particular, I should like to thank Dave Reddish, Steve Jukes, Barrie Wells, Paul Buddery, Robin Miller and Chris Neate.

National Coal Board.

The National Coal Board for sponsoring this research.

Mr. C. Shadbolt and staff at the East Midlands Subsidence Unit, Huthwaite.

Mr. S. Lewis and members of the Coal Face Mining Branch, Strata Control and Geological Sections, NCB Doncaster Area.

Mr. J. Teare and members of the Subsidence, Survey and Geological Sections, NCB North-East Area. In particular I would also like to thank the Survey and Geological Staff at the Blackhall, Horden, Dawdon and Westoe Collieries.

Mr. F. Tulley, Mr. N. Brownbridge and all the members of Hickleton Colliery, NCB Doncaster Area.

Severn-Trent Water Authority.

Mr. J. Black and other members of the Severn Area Unit, Malvern.

Members of the Trent Area Unit, Nottingham.

Finally, I should like to thank Mrs Margaret Carter for her patience and care in typing this thesis.

REFERENCES

1. SINGH, M.M. and KENDORSKI, F.S. Strata Disturbance Prediction for Mining beneath Surface Water and Waste Impoundments. Proceedings 1st Annual Conference on Ground Control in Mining, West Virginia University, Morgantown, USA, July 1981.

2. N.C.B. 'Working Under the Sea', Production Department Instruction PI/1968/8.

3. N.C.B. Subsidence Engineers Handbook, Mining Department, London 1975.

4. BABCOCK, C.O. and HOOKER, V.E. Results of Research to develop Guidelines for Mining near Surface and Underground Bodies of Water. U.S. Department of the Interior, Bureau of Mines, 1977.

5. WARD, R.C. Principles of Hydrology, 2nd Edition, McGraw-Hill Book Company (UK) Limited, 1975.

6. BOUWER, H. Groundwater Hydrology, McGraw-Hill Book Company, 1978.

7. WOODWARD, H.B. The Geology of Water Supply, Edward Arnold (London), 1910.

8. BENNISON, G.M. and WRIGHT, A.E. The Geological History of the British Isles, Edward Arnold (Publishers) Ltd, London, 1975.

9. GREIG, D.A. The South of Scotland, British Regional Geology, Institute of Geological Sciences, HMSO Edinburgh, 3rd Edition 1971.

10. TAYLOR, B.J., BURGESS, I.C., LAND, D.H., MILLS, D.A.C., SMITH, D.B. and WARREN, P.T. Northern England, British Regional Geology, Institute of Geological Sciences, HMSO London 4th Edition, 1971.

11. EDWARDS, W. and TROTTER, F.M. The Pennines and Adjacent Areas, British Regional Geology, Institute of Geological Sciences, HMSO London, 3rd Edition, 1975.

12. HAINES, B.A. and HORTON, A. Central England, British Regional Geology, Institute of Geological Sciences, HMSO London, 3rd Edition, 1975.

13. GALLOIS, R.W. The Wealden District, British Regional Geology, Institute of Geological Sciences, HMSO London, 4th Edition, 1968.

14. INESON, J. Groundwater Conditions in the Coal Measures of the South Wales Coalfield, NERC Hydrological Report No. 3, Institute of Geological Sciences, London, 1967.

15. TRUEMAN, A. The Coalfields of Great Britain. Edward Arnold Ltd, London, 1954.

16. MACGREGOR, M. The Coalfields of Scotland. Chapter XVI, p.325 - 381 in the Coalfields of Great Britain Edited by Sir A. Trueman, Edward Arnold Ltd, London, 1954.

17. ORCHARD, R.J. Working Under Bodies of Water, The Mining Engineer, Vol. 134, p.261 - 270, 1975.

18. LEVITT, E. Water Disposal from Underground Coal Mining Activities. Part 1 - Sources, Symposium on Mining and Water Pollution, Institute of Water Engineers and Scientists, Nottingham, June 1981.

19. ROGOZ, M. Water Capacity of Abandoned Workings in Underground Coal Mines, Symposium on Water in Mining and Underground Works, SIAMOS 1978, Granada, Spain, Vol. 2, p.1291 - 1302.

20. HEADWORTH, H.G., PURI, S. and RAMPLING, B.H. Contamination of a Chalk Aquifer by Mine Drainage at Tilmanstone, East Kent, U.K. Q.J. of Engineering Geology, London, 1980, Vol. 13, p.105 - 117.

21. GLOVER, H.G. Water Disposal from Underground Coal Mining Activities. Part 2 - Control. Symposium on Mining and Water Pollution, Institute of Water Engineers and Scientists, Nottingham, June 1981.

22. WOODWARD, G.M. The Effect of Coal Mining on Water Quality. Symposium of Mining and Water Pollution, Institute of Water Engineers and Scientists, Nottingham, June 1981.

23. EDWARDS, R.W. The Impact of Coal Mining on River Ecology. Symposium on Mining and Water Pollution, Institute of Water Engineers and Scientists, Nottingham, June 1981.

24. GISMAN, S. and SZCZYPA, H. Trends in Utilization of Mine Waters in the Upper Silesian Coal Basin, 1st International Mine Water Congress, Budapest, Hungary, Proceedings D, p17 - 23, April 1982.

25. WALTON, W.C. Groundwater Resource Evaluation, McGraw-Hill Kogakusha Ltd, 1970.

26. Symposium of Water in Mining and Underground Works, SIAMOS 1978, Granada, Spain, 3 volumes.

27. 1st International Mine Drainage Symposium, Denver, USA, 1979, 1 Volume.

28. 1st International Mine Water Congress Proceedings, Budapest, Hungary, April 1982.

29. DARCY, H. Les Fontaines Publiques de la Ville de Dijon, Dalmont, Paris, 1856.

30. SCHEIDEGGAR, A.E. The Physics of Flow Through Porous Media. 3rd Edition, University of Toronto Press, Toronto, 1974.

31. LOOFBOUROW, R.L. Controlling Mine Water. Proceedings 1st International Mine Drainage Symposium, Denver, USA, 1979, Section 3, p.507 - 528.

32. KRUSEMAN, G.P. and de RIDDER, N.A. Analysis and Evaluation of Pumping Test Data, Bulletin 11, International Institute for Land Reclamation and Improvement, The Netherlands, 1979.

33. BEAR, J. Hydraulics of Groundwater, McGraw-Hill, 1979, Chapter 8, p.300 - 379.

34. THEIS, C.V. The Relationship between the Lowering of the Piezometric Surface and the Rate and Duration of Discharge of a Well Using Groundwater Storage. Transactions, American Geophysical Union, 1935, p.519 - 524.

35. EAGON, H.B. and JOHE, D.E. Practical Solutions for Pumping Tests in Carbonate Rock Aquifers. Groundwater, Vol. 10, No. 4, p.6 - 13, 1972.

36. CLARK, L. The Analysis and Planning of Step Drawdown Tests. Q. J. Engineering Geology, 1977, Vol. 10, p.125 - 143.

37. RADHAKRISHNA, I. and VENKATESWARLU, T. Aquifer Response in the Ultramafic Field Area of Sukhinda Valley - Orissa (India). Groundwater Vol. 18, No. 5, p.479 - 485, 1980.

38. HVORSLEV, M.J. Time Lag and Soil Permeability in Groundwater Observations. Bulletin No. 36, Waterways Experiment Station, Corps of Engineers, US Army, Vicksburg, Mississippi, 1951.

39. HOEK, E. and BRAY, J. Rock Slope Engineering, 2nd Edition, Institute of Mining and Metallurgy Publication, London, 1977, Chapter 6, p.127 - 148.

40. WHITTAKER, B.N. and SINGH, R.N. Design aspects of barrier pillars against water logged workings in coal mining operations. Symposium of Water in Mining and Underground Works, SIAMOS 1978, Granada, Spain, Vol. 1, p.675 - 692.

41. NEATE, C.J. Effects of Mining Subsidence on the Permeability of Coal Measures Rocks. Ph.D. Thesis, Nottingham University, October 1980.

42. PRICE, M., MORRIS, B. and ROBERTSON, A. A Study of Intergranular and Fissure Permeability in Chalk and Permian Aquifers, Using Double-Packer Injection Testing, J. of Hydrology, Vol. 54, 1982, p.401 - 423.

43. BARKER, J.A. A Formula for Estimating Fissure Transmissivities from Steady-State Injection-Test Data. J. of Hydrology, Vol. 52, 1981, p.337 - 346.

44. SHADBOLT, C.H. 'Mining Subsidence' Chapter 6 in "Site Investigations in areas of Mining Subsidence", Edited by F.G. Bell, Newnes-Butterworths, London, 1975, p.109 - 124.

45. WARDELL, K. and BEEVERS, C. Recent Research in Mining Subsidence. Trans. Inst. Min. Eng., Vol. 114, 1954-55, p.223 - 244.

46. ORCHARD, R.J. Surface Effects of Mining - The Main Factors. Trans. Inst. Min. Eng., Vol. 116, 1957, p.942 - 955.

47. OLIVER, M.E. A Study of Surface Displacements and Strains in Mining Subsidence Resulting from Multi-Seam Workings. M.Phil. Thesis, Nottingham University, May 1980.

48. BREEDS, C.D. A Study of Mining Subsidence Effects on Surface Structures with Special Reference to Geological Conditions. Ph.D. Thesis, Nottingham University, October 1976.

49. Personal Communication, East Midlands Subsidence Unit, Huthwaite, Nr. Mansfield.

50. REDDISH, D.J. Computer Aspects of Mining Subsidence Simulation. Ph.D. Thesis in Preparation, Nottingham University.

51. ASTON, T.R.C. and REDDISH, D.J. A Comparison of Actual and Calculated Subsidence Profiles for Shallow Coal Workings in the East Midlands Coalfield. Progress Report, November 1981, University of Nottingham.

52. WHITTAKER, B.N. and BREEDS, C.D. The Influence of Surface Geology on the Character of Mining Subsidence. The Geotechnics of Structurally Complex Formations - Proceedings Associazione Geotecnica Italiana, Capri, 1977.

53. FORRESTER, D.J. An Investigation into the Effect of Undermining Colliery Spoil Heaps. Ph.D. Thesis, University of Nottingham, May 1974.

54. WILLIAMSON, W.H. Hydrological Aspects of Coal Mining under Stored Waters near Sydney, Australia. Proc. Symposium on Water in Mining and Underground Works, Granada, Spain, 1978, p.309 - 328.

55. McPHERSON, M.J. The Occurrence of Methane in Mine Workings. J. Mine Vent. Soc. of S.A., Vol. 28, No. 8, August 1975, p.118 - 128.

56. NEATE, C.J. and WHITTAKER, B.N. Influence of Proximity of Longwall Mining on Strata Permeability and Ground Water. Proc. 20th Annual Symp. Rock Mech., Austin, Texas, 1979.

57. WHITTAKER, B.N., SINGH, R.N. and NEATE, C.J. Effect of Longwall Mining on Ground Permeability and Subsurface Drainage. Proc. 1st International Mine Drainage Symposium, Denver, USA, 1979, p.161 - 183.

58. WHITTAKER, B.N. and SINGH, R.N. Evaluation of the Design Requirements and Performance of Gate Roadways. Trans. Inst. Min. Eng., February 1979, p.101 - 119.

59. MILLER, R.R. A Study of Design and Structural Stability Aspects of Mine Roadway Support Systems. Ph.D. Thesis, University of Nottingham, March 1981.

60. Sperry-Sun Magnetic Multishot Directional Survey Equipment, Instruction Manual, Surveying Department, N.C.B. Doncaster Area.

61. HUMPHRYS, H.J. Barnborough Main Colliery, South Yorkshire, Report on the Causes of and Circumstances attending the Upheaval of Floor which occurred on the 24th April 1942, at the Barnborough Main Colliery, S. Yorks., HMSO Ministry of Fuel and Power, 1943, Cmd. 6414.

62. BRYSON, J.F. Method of Eliminating Coal Bumps or Minimizing their Effects. Trans. Am. Inst. of Min. and Metall. Eng., Coal Division, Vol. 119, 1936, p.40 - 57.

63. Prevention of Rockbursts. Proc. of Symposium given by the Rockburst Prevention Development Committee at Steinkohlenbergbauverein, Essen, on 19th April 1978. Gluckauf and Translation 114 (1978) Nr. 17, p.354 - 375.

64. The Occurrence of Bumps in the Thick Coal Seam of South Staffordshire. First Progress Report of the Safe Working of Mines Committee of the South Staffordshire and Warwickshire Institute of Mining Engineers. Trans. Inst. of Min. Eng. Vol. 85, 1932-33, p.116 - 147.

65. SHEPHERD, R. and KELLET, W.H. Strata Behaviour - A study of faces and roadways in workings under a sandstone roof liable to rock bumps. Colliery Guardian, March 1973, p.93 - 102 and 195.

66. JOHNSON, J.E. East Parkgate Seam - "Weight Bumps", Hickleton Colliery, Doncaster Area. N.C.B. Yorkshire Regional Geological Services Report, May 1974.

67. Proposed Working of the P30 and P28 Panels, Parkgate Seam, Hickleton Colliery, NCB Doncaster Area, M.R.D.E. Bretby, Internal Report, April 1980.

68. PHILLIPS, D.W. Rock Bursts or 'Bumps' in Coal Mines. Trans. Inst. Min. Eng., Vol. 104, 1944-45, p.55 - 91.

69. RICE, G.S. Bumps in Coal Mines - Theories of Causes and Suggested Means of Prevention or of Minimizing Effects. Trans. Am. Inst. of Min. and Metall. Eng., Coal Division, Vol. 119, 1936, p.11 - 39.

70. MITCHELL, G.H., STEPHENS, J.V., BROMEHEAD, C.E.W. and WRAY, D.A. Geology of the Country around Barnsley - Explanation of Sheet 87. Memoirs of the Geological Survey of Great Britain, HMSO London, 1954 Reprint.

71. HEDLEY, D.G.F. Design Criteria for Multi-Wire Borehole Extensometer Systems. Proc. 1st Canadian Symposium on Mine Surveying and Rock Deformation Measurement, Fredericton, October 1969, p.349 - 377.

72. WHITTAKER, B.N. and HODGKINSON, D.R. Strata Displacement Measurement by Multi-Wire Borehole Instrumentation. Colliery Guardian, September 1970, p.445 - 449.

73. Minutes of Meeting, Coalface Mining Operations Branch, H.Q. Mining Department, Doncaster, June 1980.

74. WHITTAKER, B.N. Communication to Coalface Mining Operations Branch, H.Q. Mining Department, Doncaster, June 1980.

75. Personal Communication - Mine Geologist, Hickleton Colliery.

76. Minutes of Meeting, Hickleton Colliery, N.C.B. Doncaster Area, 5th September 1980.

77. JAEGER, J.C. and COOK, N.G.W. Fundamentals of Rock Mechanics, Chapman and Hall, London, 1976.

78. VUTUKURI, V.S., LAMA, R.D. and SALUJA, S.S. Handbook on Mechanical Properties of Rocks - Testing Techniques and Results - Volume 1. Trans. Tech. Publication, USA First Edition 1974.

79. FRANKLIN, J.A., BROCH, E. and WALTON, G. Logging the Mechanical Character of Rock. Trans. Inst. Min. Metall. (Section A), Vol. A.80, 1971, p.A1 - A9.

80. DEERE, D.U. Technical Description of Rock Cores for Engineering Purposes. Int. J. Rock. Mech. Min. Sci., Vol. 1, 1964, p.16 - 22.

81. BIENIAWSKI, Z.T. The Point-Load Test in Geotechnical Practice. Engineering Geology, Vol. 9, 1975, p.1 - 11.

82. HASSANI, F.P. A Study of the Physical and Mechanical Properties of Rocks and their Discontinuities associated with Opencast Coal Mining Operations. Ph.D. Thesis, University of Nottingham, December 1980.

83. ELKINGTON, P.A.S. Wireline Logging and Coal Measures Rock Strength. Ph.D. Thesis, University of Nottingham, May 1981.

84. BROCH, E. and FRANKLIN, J.A. The Point-Load Strength Test. Int. J. Rock. Mech. Min. Sci., Vol. 9, 1972, p.669 - 697.

85. HASSANI, F.P., SCOBLE, M.J. and WHITTAKER, B.N. Application of the Point-Load Index to Strength Determination of Rock and Proposals for a New Size Correction Chart. Proc. 21st Annual Symp. Rock. Mech., Univ. of Missouri, 1980.

86. HASSANI, F.P., WHITTAKER, B.N. and SCOBLE, M.J. Strength Characteristics of Rocks associated with Opencast Coal Mining in the U.K. Proc. 20th Annual Symp. Rock Mech., Austen, Texas, 1979.

87. JUKES, S.G. - Personal Communication, Research Student, Dept. Of Mining, University of Nottingham.

88. SINGH, R.N., BUDDERY, P.S. and WELLS, B.T. Geotechnical Investigations and Appraisal of Ground Control Practice for Roof Support Design. Proc. of 1st Int. Conf. on Stability in Underground Mining, Vancouver, B.C., Canada, August 1982.

89. Steps to Reduce the Risk of Rockbursts. Mines Inspectorate of North Rhine Westphalia, Rockburst Guidelines April 1976, Doc. No. 3458/78e.

90. BRAUNER, G. Personal Communication, Steinkohlenbergbauverein, 4300 Essen-Kray, Franz-Fischer-Weg 61.

91. GOTTIG, P. Winning and Working in Rockburst Conditions. Gluckauf and Translation 114 (1978) Nr. 17, p.360 - 363.

92. HESS, H. Papers from Joint Organisations and Member Companies engaged in Rockburst Prevention. Gluckauf and Translation 114 (1978) Nr. 17, p.355 - 359.

93. WATSON, H.F. Undersea Coal Mining in North-East England. Proc. 10th World Mining Congress, Istanbul, September 1979, Vol. 3, Paper 3.

94. SHEARD, R.L. and HURST, K.G. A History of Water Problems in the S. Lancs. Coalfield. Trans. Inst. of Min. Eng., Vol. 132, 1973, p.557 - 574.

95. PLUMPTRE, J.H. Underground Waters in the Kent Coalfield. Trans. Inst. of Min. Eng., Vol. 119, 1959-60, p.155 - 169.

96. SAUL, H. Current Mine Drainage Problems. Trans. Inst. of Min. Eng., Vol. 129, 1970, p.643 - 657.

97. GARRITTY, P. Effects of Mining on Surface and Sub-Surface Water Bodies. Ph.D. Thesis, University of Newcastle-upon-Tyne, October 1980.

98. SMITH, D.B. and FRANCIS, E.A. Geology of the Country between Durham and West Hartlepool - Explanation of Sheet 27. Memoirs of the Geological Survey of Great Britain, HMSO London, 1967.

99. SHERBOURNE HILLS, E. Elements of Structural Geology. Published Chapman and Hall Ltd, 1975, p.164 - 215.

100. ROWELL, D. Personal Communication, Mine Surveyor, Horden Colliery.

101. WILSON, K. Working of Main Seam 'F' and Yard Seam 'G' under the High Main 'E'. N.E. 4 District, Horden Colliery. N.C.B. Internal Report, South Durham Area, December 1971.

102. HSIEH, P.A., TRACY, J.V., NEUZIL, C.E., BREDEHOEFT, J.D. and SILLIMAN, S.E. A Transient Laboratory Method for Determining the Hydraulic Properties of 'Tight' Rocks. Int. J. Rock. Mech. Min. Sci., Vol. 18, 1981, p.245 - 258.

103. EDMUNDS, W.M. Geochemistry of Brines in the Coal Measures of North-East England. Trans. Inst. Min. Metall. (Section B), 1975, B39 - B52.

104. ELLIS, J.V. A Review of the work done in developing a graphical system for the plotting of water analysis and its application at the Easington, Horden and Blackhall Collieries. Contribution to a Seminar on Water Analysis. Geology Branch, North-East Area Mining Branch, National Coal Board.

105. ZAPOROZEC, A. Graphical Interpretation of Water Quality Data. Groundwater, Vol. 10, No. 2, March-April 1972.

106. PIPER, A.M. A Graphical Procedure in the Geochemical Interpretation of Water Analyses. Trans. Am. Geophys. Union, 1944, p.914 - 923.

107. CHILINGER, G.V. Durvo's Classification of Natural Waters and Chemical Composition of Atmospheric Precipitation in USSR: A Review. Trans. Am. Geophys. Union, Vol. 37, No. 2, April 1956, p.193 - 196.

108. CHILINGER, G.V. Cl' and SO_4" Content of Atmospheric Precipitation in USSR: A Summary. Trans. Am. Geophys. Union, Vol. 37, No. 4, August 1956, p.410 - 412.

109. KEET, B. Statistical Hydrochemistry as a Tool to Facilitate the Control of Mine Water. Int. J. of Mine Water, Vol. 3, September 1982 (In Preparation).

110. LANGLEY, K.M. Geochronological Investigation of Selected Sediments, Faults and Waters in British Coalfields. Ph.D. Thesis, University of Sheffield, October 1976.

111. DOWNING, R.A., SMITH, D.B. and WARREN, S.C. Seasonal Variations of Tritium and Other Constituents in Groundwater in the Chalk near Brighton, England. J. Inst. Water Engineers and Scientists, Vol. 32, No. 2, March 1978, p.123 - 136.

112. DEENEY, A.C. Worfe Recharge Investigation - Preliminary Report on the Suitability of an area north of the Mad Brook. Severn Area Unit, Severn-Trent Water Authority, Interim Report, 1978.

113. BLACK, J. Personal Communication. Area Hydrogeologist, Severn Area Unit, Severn-Trent Water Authority, Malvern, Worcs.

114. ASTON, T.R.C. and SMITH, S.F. Hydrogeological Aspects Associated with British Ironstone Mining. Proc. First International Mine Water Congress, Budapest, Hungary, April 1982, Vol. A, p.9 - 26.

APPENDIX A

CALCULATION OF SUBSIDENCE PROFILES FOR WIDTH-DEPTH RATIO'S GREATER THAN 3.0

APPENDIX A

CALCULATION OF SUBSIDENCE PROFILES FOR WIDTH-DEPTH RATIO'S GREATER THAN 3.0

A.1 Introduction

The Subsidence Engineers Handbook (SEH) (3) only allows subsidence profiles to be calculated for workings with width-depth (w/h) ratios of less than 3.0. However, work by the author, Chapters 3 and 8, has necessitated the calculation of profiles with w/h ratios greater than 3.0. It is therefore proposed to outline the means by which the Subsidence Engineers Handbook methods were extrapolated.

A.2.1 Maximum Subsidence

Maximum subsidence is calculated by multiplying the extracted seam height by an s/m factor which is derived from width and depth values of the working. Figure A.1 shows the relationship of subsidence to width and depth, and reveals that s/m factors can be readily derived for workings with width and depth values as low as 50 m. Similarly, an s/m factor can be obtained for workings with values as low as 25 m by carefully extrapolating the s/m lines.

A.2.2 Distance of Subsidence Profile Points (s/S) from Face Line

A.2.2.1 Transverse Profile

The SEH graph for predicting the location of s/S subsidence profile points is given in Figure A.2, while the tabulated values

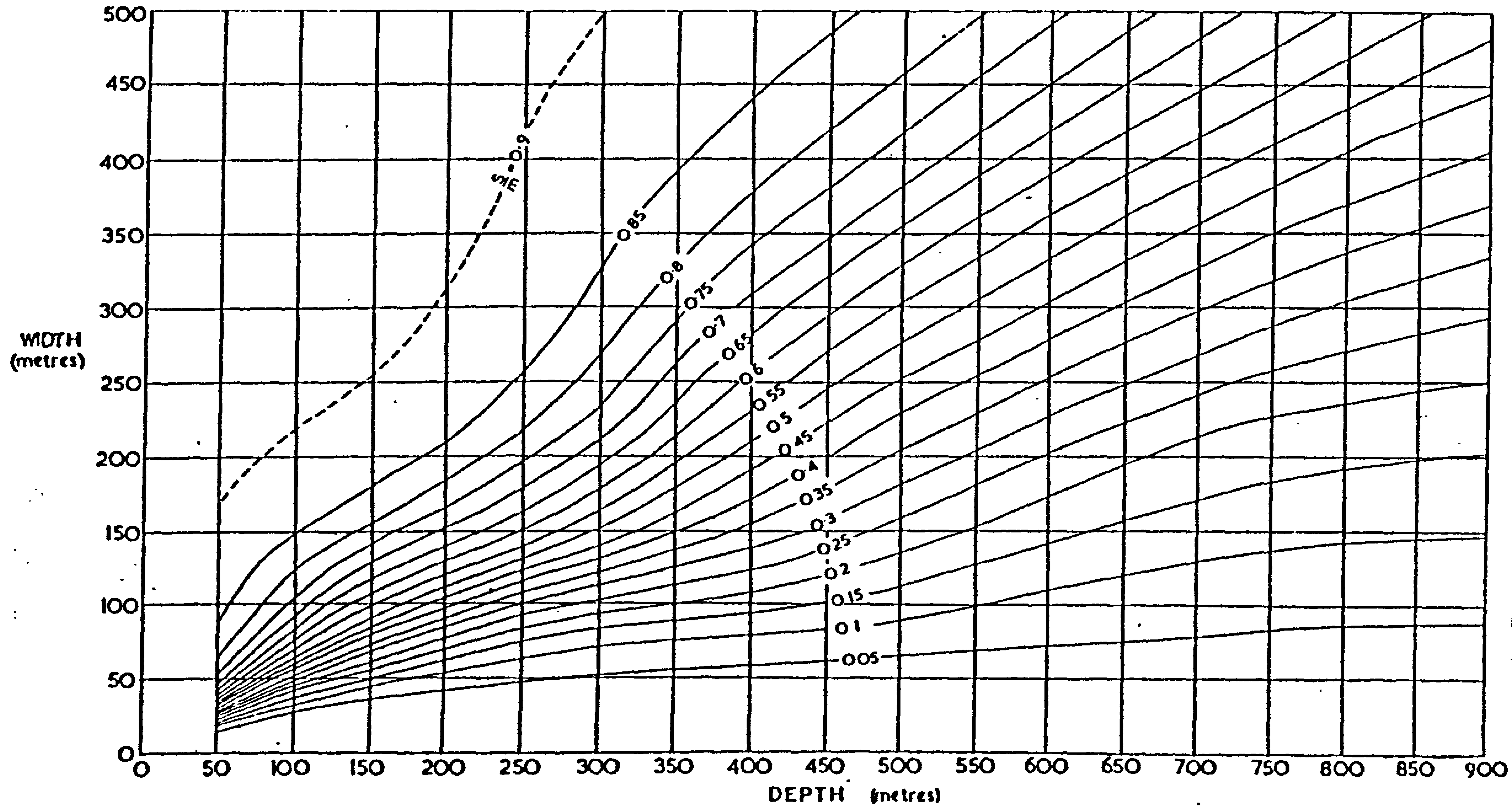


Figure A.1 <u>The Relationship between Subsidence and the Width and Depth of a Working Panel</u>
(after Subsidence Engineers Handbook (3))

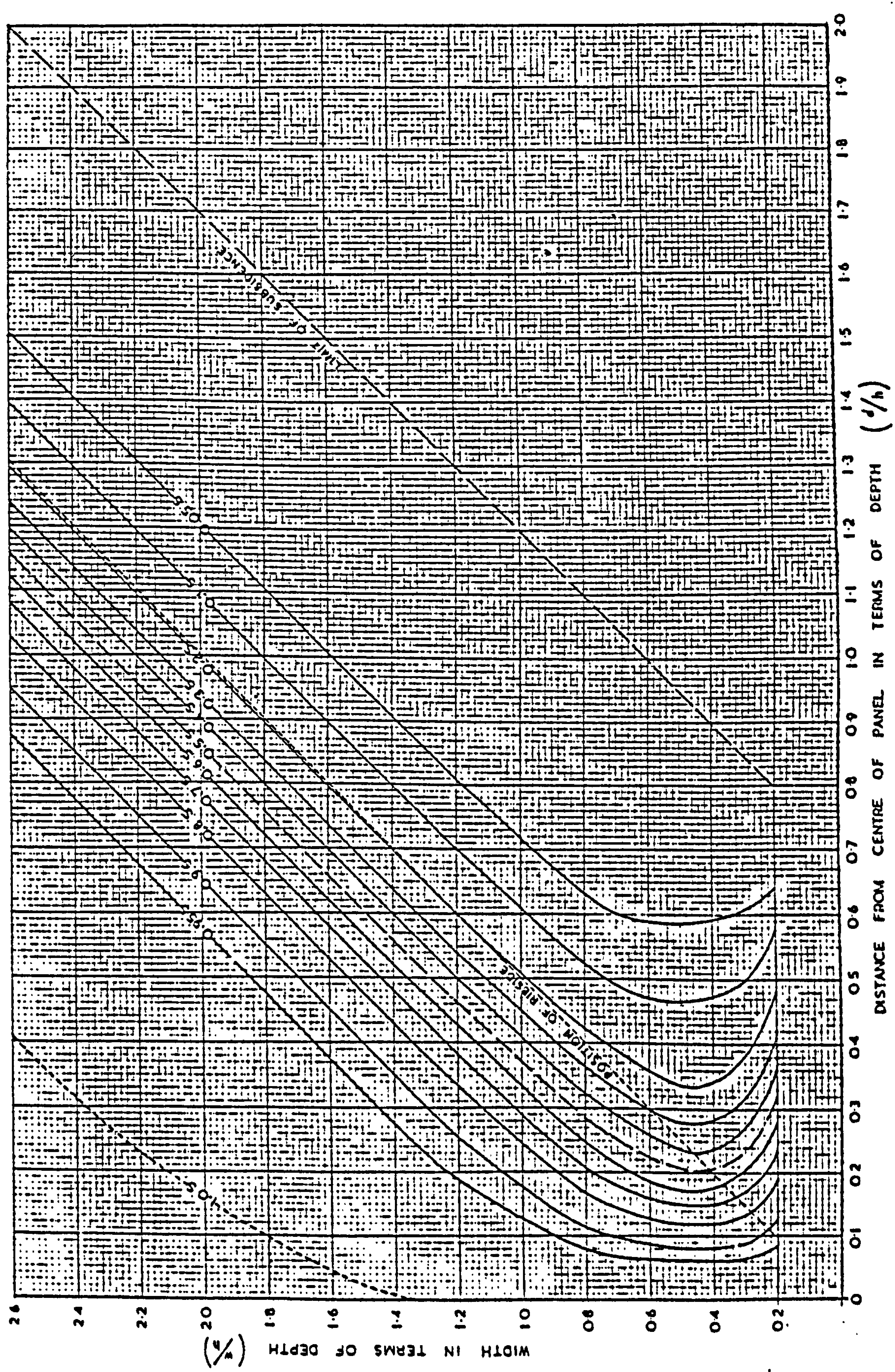


Figure A.2 Graph for the Prediction of Transverse Subsidence Profile Contours

(after Subsidence Engineers Handbook (3))

Values of s/S	0	0.05	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80	0.90	0.95	1.00
w/h RATIO OF PANEL	DISTANCES FROM PANEL CENTRE IN TERMS OF DEPTH												
2.6	2.00	1.51	1.39	1.29	1.24	1.19	1.16	1.12	1.08	1.03	0.95	0.87	0.41
2.4	1.90	1.41	1.29	1.19	1.14	1.10	1.06	1.02	0.98	0.93	0.85	0.77	0.31
2.2	1.80	1.31	1.19	1.09	1.04	1.00	0.96	0.92	0.88	0.83	0.75	0.67	0.23
2.0	1.70	1.21	1.09	0.99	0.94	0.90	0.86	0.82	0.78	0.73	0.65	0.57	0.16
1.8	1.60	1.11	1.00	0.90	0.84	0.80	0.76	0.72	0.68	0.63	0.55	0.47	0.10
1.6	1.50	1.01	0.90	0.80	0.74	0.70	0.66	0.62	0.58	0.53	0.45	0.37	0.05
1.4	1.40	0.91	0.80	0.70	0.64	0.60	0.56	0.52	0.48	0.43	0.35	0.27	0.01
1.3	1.35	0.86	0.75	0.65	0.59	0.55	0.51	0.47	0.43	0.38	0.30	0.23	0
1.2	1.30	0.81	0.70	0.60	0.54	0.50	0.46	0.42	0.38	0.33	0.25	0.19	0
1.1	1.25	0.77	0.65	0.55	0.50	0.45	0.42	0.38	0.34	0.29	0.21	0.16	0
1.00	1.20	0.72	0.61	0.51	0.45	0.41	0.37	0.33	0.29	0.24	0.18	0.13	0
0.98	1.19	0.71	0.60	0.50	0.44	0.40	0.36	0.33	0.28	0.24	0.17	0.12	0
0.96	1.18	0.70	0.59	0.49	0.43	0.39	0.35	0.32	0.27	0.23	0.16	0.11	0
0.94	1.17	0.69	0.58	0.48	0.42	0.38	0.34	0.31	0.26	0.22	0.16	0.11	0
0.92	1.16	0.68	0.57	0.47	0.41	0.37	0.33	0.30	0.26	0.21	0.15	0.10	0
0.90	1.15	0.68	0.57	0.46	0.40	0.36	0.32	0.29	0.25	0.20	0.14	0.10	0
0.88	1.14	0.67	0.56	0.45	0.40	0.36	0.32	0.28	0.24	0.20	0.13	0.10	0
0.86	1.13	0.66	0.55	0.45	0.39	0.35	0.31	0.27	0.23	0.19	0.13	0.09	0
0.84	1.12	0.65	0.54	0.44	0.38	0.34	0.30	0.26	0.22	0.18	0.12	0.09	0
0.82	1.11	0.64	0.53	0.43	0.37	0.33	0.29	0.26	0.21	0.17	0.12	0.08	0
0.80	1.10	0.63	0.52	0.42	0.36	0.32	0.28	0.25	0.21	0.17	0.11	0.08	0
0.78	1.09	0.63	0.52	0.42	0.36	0.32	0.28	0.24	0.20	0.16	0.11	0.08	0
0.76	1.08	0.62	0.51	0.41	0.35	0.31	0.27	0.23	0.20	0.16	0.11	0.07	0
0.74	1.07	0.61	0.50	0.40	0.34	0.30	0.26	0.23	0.19	0.15	0.10	0.07	0
0.72	1.06	0.61	0.50	0.39	0.34	0.30	0.26	0.22	0.18	0.15	0.10	0.07	0
0.70	1.05	0.60	0.49	0.39	0.33	0.29	0.25	0.21	0.18	0.14	0.10	0.07	0
0.68	1.04	0.60	0.49	0.38	0.32	0.28	0.24	0.21	0.17	0.14	0.10	0.07	0
0.66	1.03	0.60	0.48	0.38	0.32	0.28	0.24	0.20	0.17	0.14	0.09	0.06	0
0.64	1.02	0.59	0.48	0.37	0.31	0.27	0.23	0.20	0.17	0.13	0.09	0.06	0
0.62	1.01	0.59	0.47	0.37	0.31	0.27	0.23	0.19	0.16	0.13	0.09	0.06	0
0.60	1.00	0.59	0.47	0.36	0.30	0.26	0.22	0.19	0.16	0.13	0.09	0.06	0
0.58	0.99	0.59	0.47	0.35	0.30	0.25	0.22	0.18	0.16	0.13	0.09	0.06	0
0.56	0.98	0.59	0.47	0.35	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0
0.54	0.97	0.59	0.47	0.34	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0
0.52	0.96	0.59	0.47	0.34	0.28	0.24	0.21	0.17	0.15	0.12	0.08	0.06	0
0.50	0.95	0.59	0.47	0.34	0.28	0.24	0.21	0.17	0.15	0.12	0.08	0.06	0
0.48	0.94	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0
0.46	0.93	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0
0.44	0.92	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0
0.42	0.91	0.59	0.47	0.34	0.28	0.24	0.20	0.17	0.15	0.12	0.08	0.06	0
0.40	0.90	0.59	0.47	0.34	0.28	0.24	0.21	0.18	0.15	0.12	0.08	0.06	0
0.38	0.89	0.60	0.48	0.35	0.29	0.24	0.21	0.18	0.15	0.12	0.08	0.06	0
0.36	0.88	0.60	0.48	0.35	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0
0.34	0.87	0.60	0.49	0.36	0.30	0.25	0.22	0.19	0.16	0.12	0.08	0.06	0
0.32	0.86	0.60	0.49	0.37	0.31	0.26	0.22	0.20	0.16	0.13	0.09	0.06	0
0.30	0.85	0.61	0.50	0.38	0.32	0.27	0.23	0.20	0.17	0.13	0.09	0.06	0
0.28	0.84	0.61	0.51	0.39	0.33	0.28	0.24	0.21	0.18	0.14	0.09	0.07	0
0.26	0.83	0.62	0.52	0.41	0.35	0.30	0.26	0.22	0.19	0.15	0.10	0.07	0
0.24	0.82	0.62	0.53	0.43	0.36	0.32	0.28	0.24	0.20	0.16	0.11	0.07	0
0.22	0.81	0.63	0.55	0.46	0.39	0.34	0.30	0.26	0.21	0.17	0.11	0.08	0
0.20	0.80	0.64	0.57	0.48	0.41	0.37	0.32	0.28	0.23	0.19	0.13	0.08	0

Table A.1 The Relationship between W/H Ratio and the Location of s/S Points from Panel Centre Line in Terms of Depth (after Subsidence Engineers Handbook (3)) of distance from the face centre line in terms of depth are given in Table A.1.

Above a w/h ratio of 2.6, it was assumed that all the subsidence contours (Figure A.2) continued as straight lines and exhibited a linear relationship of the form $y = mx + c$. Values were then obtained for w/h ratios up to 14.0 using a Trend Line Analysis programme available on the authors calculator, a Texas TI-51-III. Quality control checks were frequently undertaken on the derived points using an additional Correlation programme. A series of points for each Contour were only accepted when the correlation coefficient exceeded 0.98 after a substantial amount of data from Table A.1 was included in the test.

Tabulated values for w/h ratios up to 14.0 are given in Appendix B, programme lines 208 - 315, pages 397 and 398.

A.2.2.2 Longitudinal Profile

The location of s/S subsidence points from the face line for the longitudinal profile were calculated from Figure A.3, which was obtained from the East Midlands Subsidence Unit (49).

Each of the s/S contours, Figure A.3, was digitised and the tabulated values can be found in Appendix B, programme lines 532 - 645, pages 403 and 404.

A.3.1 Maximum Tensile and Compressive Strain

Maximum tensile and compressive strain values are calculated by dividing maximum subsidence by the working seam depth and multiplying the answer by a parametric factor which is obtained from

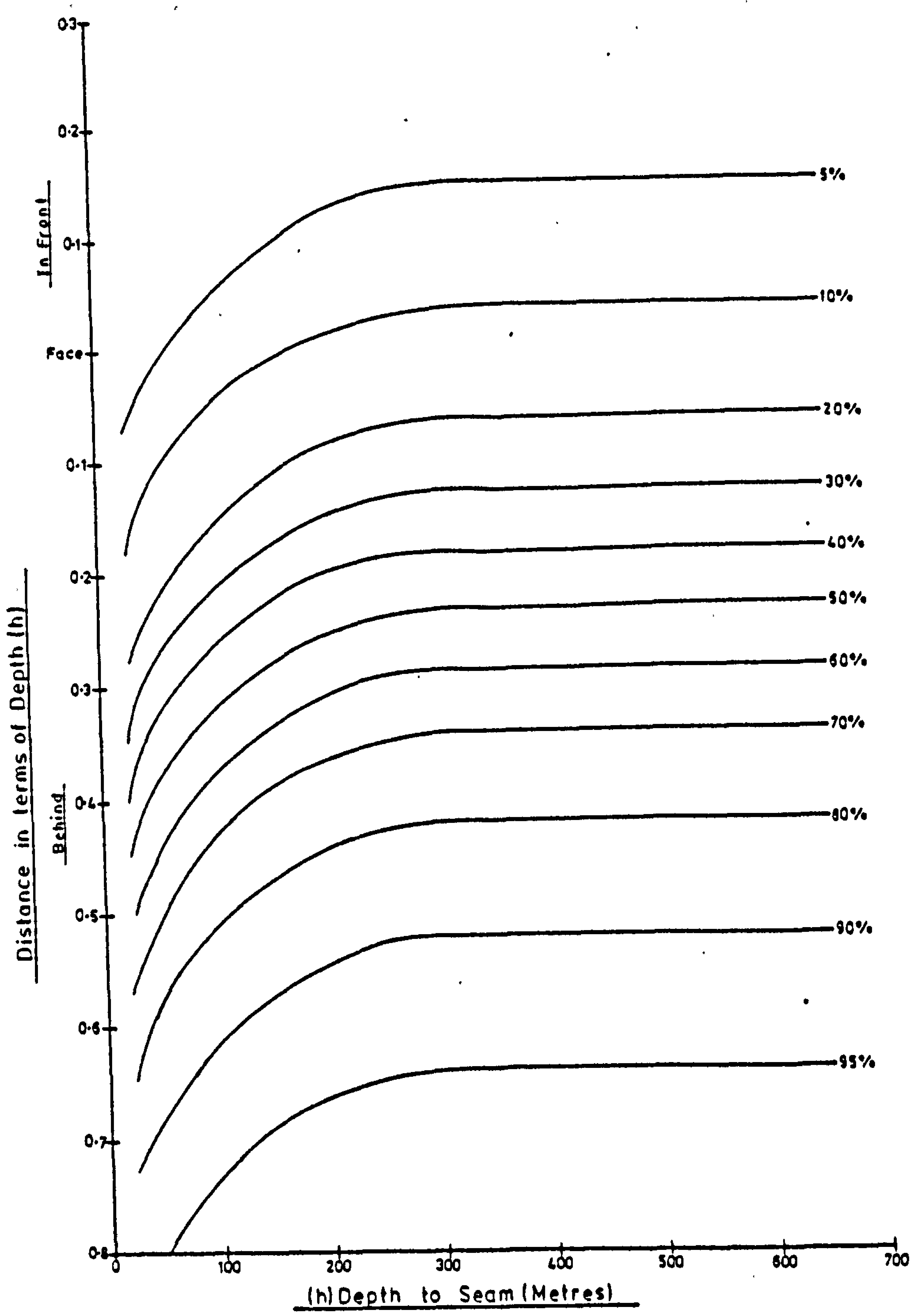


Figure A.3 Graph for the Prediction of Longitudinal Subsidence Contours
(after East Midlands Subsidence Unit (49))

Figure A.4. Beyond a w/h ratio of 1.4, it was assumed that both the tensile and compressive curves continued as straight lines. Tabulated tensile and compressive multiplier factors for w/h ratios up to 14.0 are given in Appendix B, programme lines 316 - 423, pages 399 and 400.

A.3.2 Distance of Strain Profile Points (e/E) from Face Line

A.3.2.1 Transverse Profile

The SEH graph for predicting the location of e/E strain points is given in Figure A.5, while the tabulated values of distance from the face centre line in terms of depth are given in Table A.3.

Above a w/h ratio of 3.0, it was assumed that all the strain contours continued as straight lines and exhibited a linear relationship of the form y = mx + c. Values were calculated for w/h ratios up to 14.0 using a Trend Line Analysis programme available on the authors calculator, a Texas TI-51-III. Quality control checks were frequently made on the derived points using an additional Correlation programme. A series of points for each contour were only accepted when the correlation coefficient exceeded 0.98 after a substantial portion of data from Table A.3 was included in the test.

Tabulated values for w/h ratios up to 14.0 are given in Appendix B, programme lines 424 - 531, pages 401 and 402.

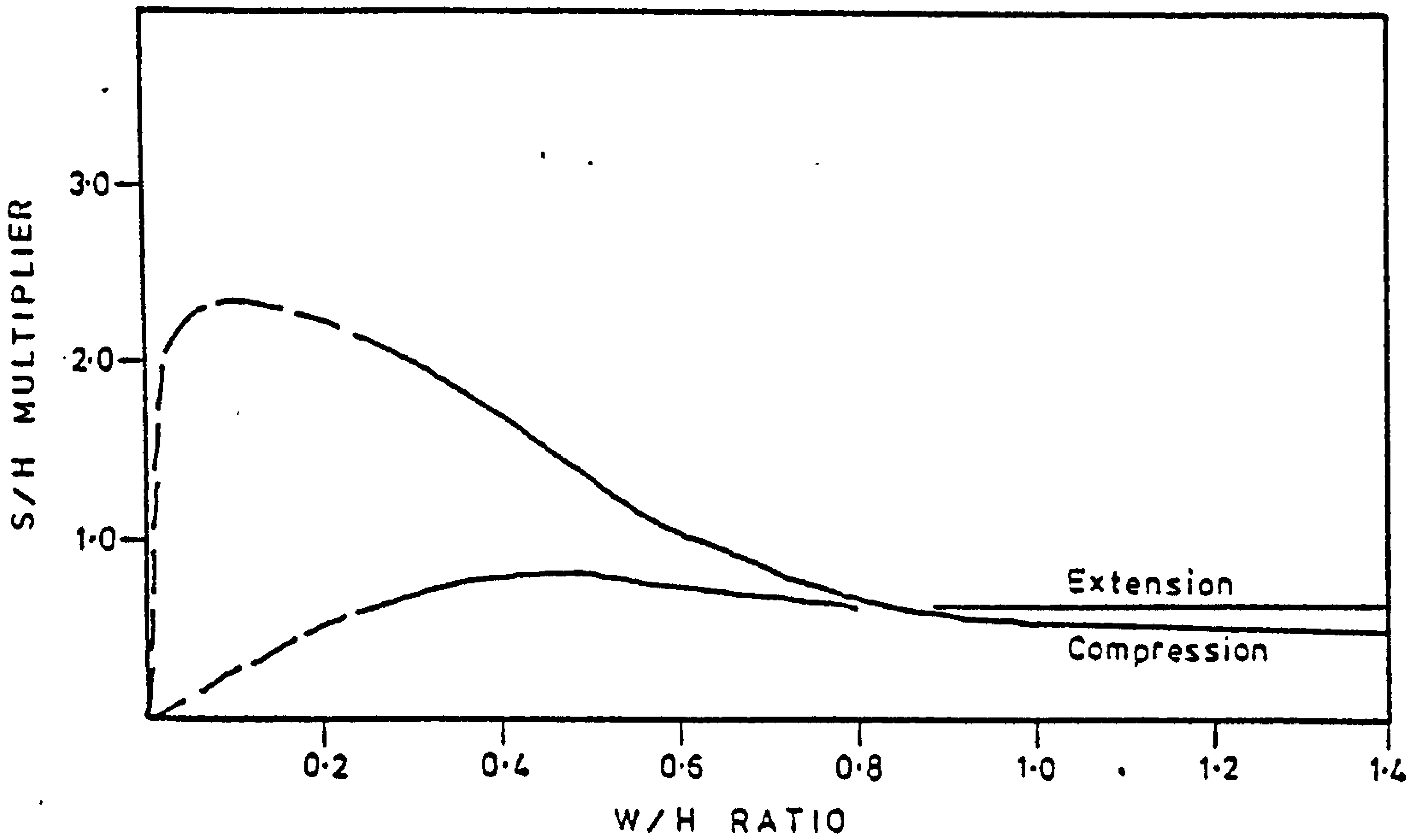


Figure A.4 NCB S/H Multiplier Values for the Prediction of Maximum Tensile and Compressive Strains
(after Subsidence Engineers Handbook (3))

W/H RATIO	TENSILE STRAIN	COMPRESSIVE STRAIN
0.2	0.55	2.25
0.4	0.80	1.70
0.6	0.80	1.05
0.8	0.65	0.70
1.0	0.65	0.55
1.2	0.65	0.50
1.4	0.65	0.50
1.6	0.65	0.50
1.8	0.65	0.50
2.0	0.65	0.50
3.0	0.65	0.50
4.0	0.65	0.50
5.0	0.65	0.50

Table A.2 NCB S/H Multiplier Values for the Prediction of Maximum Tensile and Compressive Strains
(after Subsidence Engineers Handbook (3))

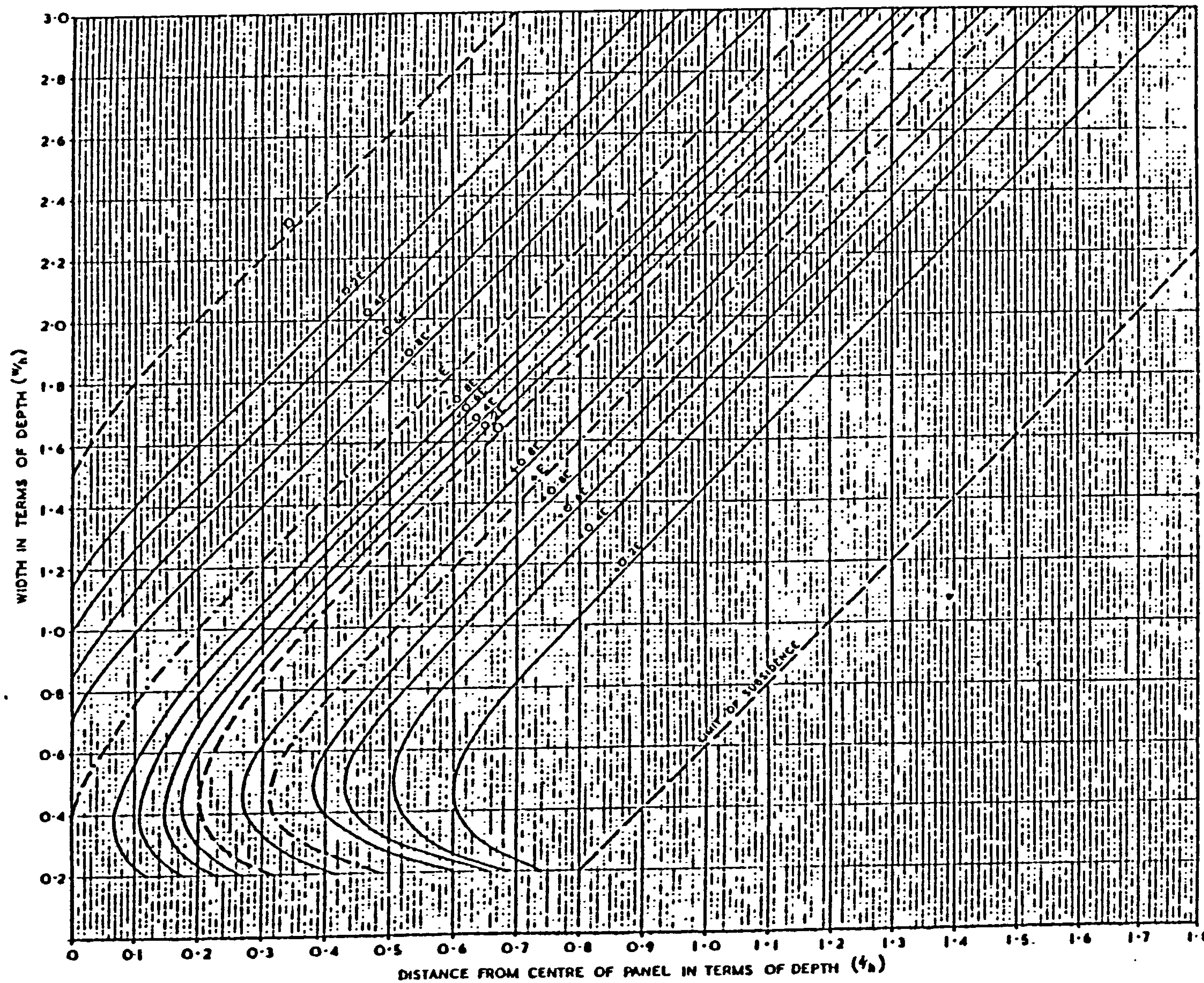


Figure A.5 Graph for the Prediction of Transverse Strain Profile Contours

(after Subsidence Engineers Handbook (3))

				Extension (+E)									Compression (—E)					
Values of e/E	0	0.20	0.40	0.60	0.80	1.00	0.80	0	0.20	0.40	0.60	0.80	1.00	0.80	0.60	0.40	0.20	0
w/h RATIO OF PANEL	DISTANCES FROM PANEL CENTRE IN TERMS OF DEPTH																	
3.0	2.2	1.78	1.67	1.61	1.56	1.50	1.46	1.36	1.34	1.31	1.28	1.25	1.19	1.10	1.03	0.96	0.90	0.70
2.6	2.0	1.58	1.47	1.41	1.36	1.30	1.26	1.16	1.14	1.11	1.08	1.05	0.99	0.90	0.83	0.77	0.70	0.50
2.2	1.8	1.38	1.27	1.21	1.16	1.10	1.06	0.96	0.94	0.91	0.88	0.85	0.79	0.70	0.63	0.67	0.60	0.30
2.0	1.7	1.28	1.17	1.11	1.05	1.00	0.96	0.86	0.84	0.81	0.78	0.75	0.69	0.60	0.53	0.47	0.40	0.20
1.8	1.6	1.17	1.07	1.01	0.95	0.90	0.86	0.76	0.73	0.71	0.68	0.65	0.59	0.50	0.43	0.37	0.30	0.10
1.6	1.5	1.08	0.97	0.91	0.85	0.80	0.76	0.66	0.63	0.61	0.58	0.55	0.49	0.40	0.33	0.27	0.20	0.03
1.4	1.4	0.98	0.87	0.81	0.75	0.70	0.66	0.56	0.53	0.51	0.48	0.45	0.39	0.30	0.23	0.17	0.10	0
1.3	1.35	0.93	0.82	0.76	0.70	0.65	0.61	0.51	0.49	0.46	0.43	0.40	0.34	0.25	0.18	0.12	0.05	0
1.2	1.3	0.88	0.77	0.71	0.66	0.61	0.56	0.46	0.44	0.41	0.38	0.35	0.29	0.20	0.13	0.07	0.02	0
1.1	1.25	0.83	0.72	0.66	0 61	0.56	0.62	0.42	0.39	0.37	0.33	0.31	0.24	0.15	0.09	0.03	0	0
1.0	1.2	0.79	0.68	0.62	0.57	0.51	0.47	0.37	0.35	0.32	0.29	0.26	0.20	0.10	0.05	0	0	0
0.98	1.19	0.78	0.67	0.61	0.56	0.50	0.46	0.36	0.34	0.31	0.28	0.25	0.19	0.09	0.04	0	0	0
0.96	1.18	0.77	0.66	0.60	0.55	0.49	0.45	0.35	0.33	0.30	0.27	0.24	0.18	0.09	0.04	0	0	0
0.94	1.17	0.76	0.65	0.59	0.54	0.48	0.44	0.35	0.32	0.30	0.26	0.23	0.17	0.08	0.03	0	0	0
0.92	1.16	0.75	0.64	0.58	0.53	0.47	0.43	0.34	0.31	0.29	0.25	0.22	0.16	0.07	0.02	0	0	0
0.90	1.15	0.74	0.63	0.57	0.52	0.46	0.42	0.33	0.30	0.28	0.24	0.21	0.16	0.06	0.02	0	0	0
0.88	1.14	0.73	0.62	0.56	0.51	0.46	0.41	0.32	0.29	0.27	0.24	0.21	0.15	0.05	0.01	0	0	0
0.86	1.13	0.72	0.61	0.55	0.50	0.45	0.40	0.31	0.29	0.26	0.23	0.20	0.14	0.05	0	0	0	0
0.84	1.12	0.71	0.60	0.54	0.49	0.44	0.39	0.30	0.28	0.25	0.22	0.19	0.13	0.04	0	0	0	0
0.82	1.11	0.70	0.59	0.53	0.48	0.43	0.38	0.29	0.27	0.25	0.21	0.18	0.12	0.03	0	0	0	0
0.80	1.10	0.69	0.58	0.53	0.48	0.42	0.37	0.29	0.26	0.24	0.20	0.17	0.11	0.02	0	0	0	0
0.78	1.09	0.68	0.57	0.52	0.47	0.41	0.36	0.28	0.26	0.23	0.20	0.17	0.11	0.02	0	0	0	0
0.76	1.08	0.67	0.57	0.51	0.46	0.40	0.36	0.27	0.25	0.22	0.19	0.16	0.10	0.01	0	0	0	0
0.74	1.07	0.67	0.56	0.50	0.45	0.39	0.35	0.26	0.24	0.22	0.18	0.15	0.09	0.01	0	0	0	0
.072	1.06	0.66	0.55	0.49	0.44	0.38	0.34	0.26	0.24	0.21	0.17	0.15	0.09	0	0	0	0	0
0.70	1.05	0.65	0.54	0.48	0.44	0.37	0.33	0.25	0.23	0.20	0.17	0.14	0.08	0	0	0	0	0
0.68	1.04	0.64	0.64	0.47	0.43	0.37	0.32	0.24	0.22	0.20	0.16	0.13	0.07	0	0	0	0	0
0.66	1.03	0.64	0.53	0.47	0.42	0.36	0.31	0.24	0.22	0.19	0.16	0.13	0.07	0	0	0	0	0
0.64	1.02	0.63	0.53	0.46	0.41	0.35	0.31	0.23	0.21	0.19	0.15	0.12	0.06	0	0	0	0	0
0.62	1.01	0.63	0.52	0.45	0.41	0.34	0.30	0.23	0.21	0.18	0.16	0.12	0.05	0	0	0	0	0
0.60	1.00	0.62	0.52	0.45	0.40	0.34	0.29	0.22	0.20	0.18	0.14	0.11	0.05	0	0	0	0	0
0.58	0.99	0.62	0.51	0.44	0.39	0.33	0.29	0.22	0.19	0.17	0.14	0.10	0.04	0	0	0	0	0
0.56	0.98	0.61	0.51	0.44	0.39	0.33	0.28	0.22	0.19	0.17	9.13	0.10	0.03	0	0	0	0	0
0.54	0.97	0.61	0.51	0.43	0.39	0.32	0.28	0.21	0.19	0.16	0.13	0.09	0.03	0	0	0	0	0
0.52	0.96	0.60	0.51	0.43	0.38	0.32	0.27	0.21	0.18	0.16	0.12	0.09	0.02	0	0	0	0	0
0.50	0.95	0.60	0.51	0.43	0.38	0.32	0.27	0.21	0.18	0.16	0.12	0.08	0.02	0	0	0	0	0
0.48	0.94	0.60	0.51	0.43	0.38	0.31	0.27	0.20	0.18	0.15	0.12	0.08	0.01	0	0	0	0	0
0.46	0.93	0.60	0.51	0.43	0.38	0.31	0.27	0.20	0.18	0.15	0.11	0.08	0.01	0	0	0	0	0
0.44	0.92	0.60	0.51	0.43	0.39	0.31	0.27	0.20	0.18	0.15	0.11	0.07	0.01	0	0	0	0	0
0.42	0.91	0.60	0.51	0.44	0.39	0.31	0.27	0.20	0.18	0.15	0.11	0.07	0	0	0	0	0	0
0.40	0.90	0.61	0.52	0.45	0.40	0.32	0.28	0.21	0.18	0.15	0.11	0.07	0	0	0	0	0	0
0.38	0.89	0.61	0.53	0.45	0.41	0.32	0.28	0.21	0.18	0.15	0.11	0.07	0	0	0	0	0	0
0.36	0.88	0.62	0.53	0.46	0.42	0.33	0.29	0.21	0.18	0.15	0.11	0.07	0	0	0	0	0	0
0.34	0.87	0.62	0.54	0.48	0.43	0.34	0.30	0.22	0.19	0.15	0.11	0.07	0	0	0	0	0	0
0.32	0.86	0.63	0.55	0.49	0.45	0.35	0.30	0.22	0.19	0.16	0.12	0.97	0	0	0	0	0	0
0.30	0.85	0.65	0.57	0.51	0.47	0.37	0.32	0.23	0.20	0.16	0.12	0.08	0	0	0	0	0	0
0.28	0.84	0.66	0.58	0.54	0.49	0.39	0.33	0.24	0.21	0.17	0.13	0.08	0	0	0	0	0	0
0.26	0.83	0.68	0.60	0.57	0.51	0.41	0.35	0.26	0.22	0.18	0.14	0.09	0	0	0	0	0	0
0.24	0.82	0.70	0.63	0.60	0.54	0.44	0.37	0.28	0.23	0.20	0.15	0.10	0	0	0	0	0	0
0.22	0.81	0.72	0.66	0.63	0.58	0.47	0.39	0.30	0.25	0.21	0.16	0.11	0	0	0	0	0	0
0.20	0.80	0.74	0.69	0.66	0.61	0.49	0.42	0.32	0.27	0.23	0.18	0.12	0	0	0	0	0	0

Table A.3 The Relationship between W/H ratio and the Location of e/E Points from Panel Centre Line in Terms of Depth (after Subsidence Engineers Handbook (3))

A.3.2.2 Longitudinal Profile

The location of e/E strain points from the face line for the longitudinal profile were calculated from Figure A.6, which was obtained from the East Midlands Subsidence Unit (49).

Each of the e/E contours, Figure A.6, was digitised and the tabulated values can be found in Appendix B, programme lines 646 - 759, pages 405 and 406.

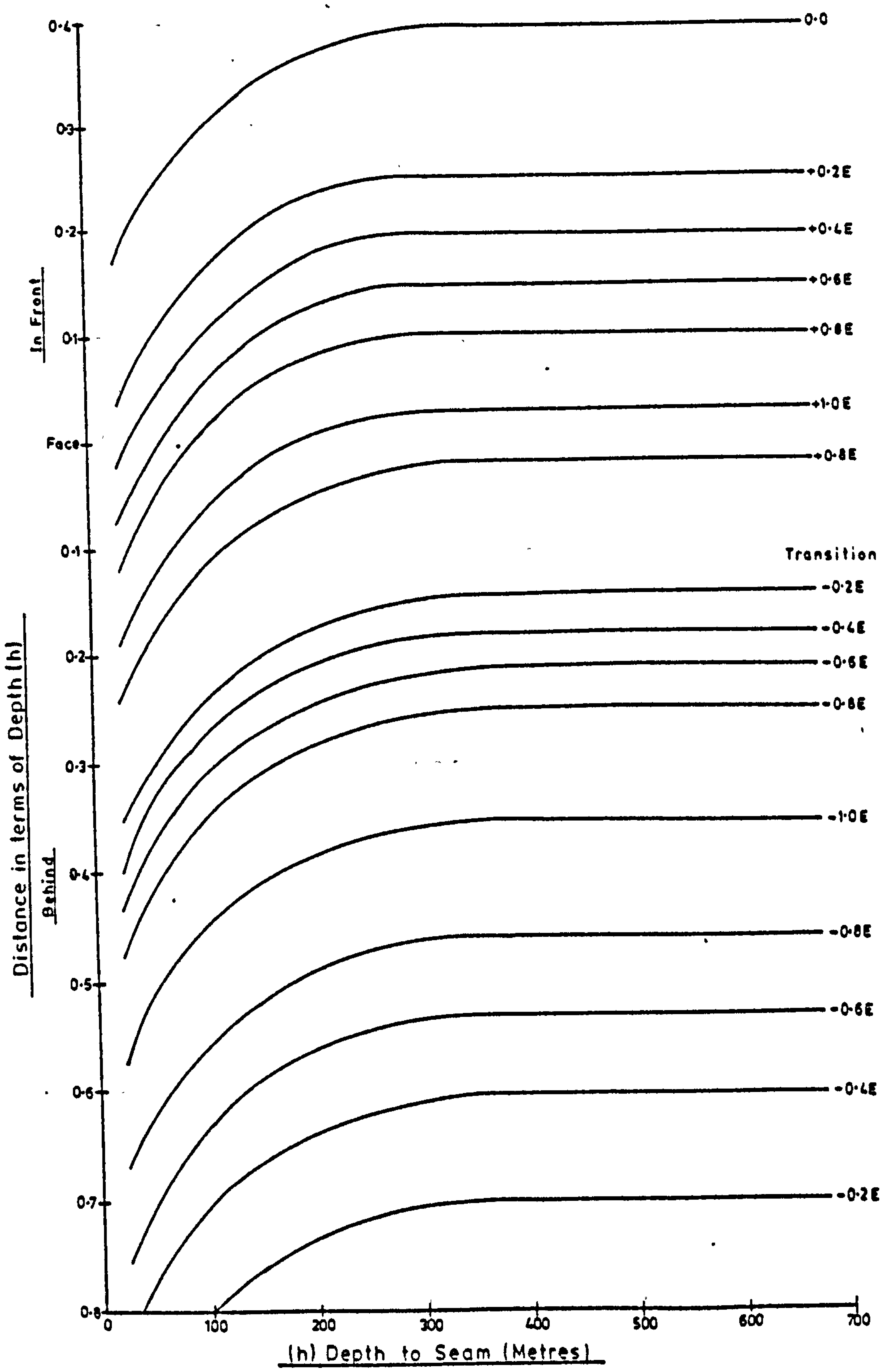


Figure A.6 Graph for the Prediction of Longitudinal Strain Contours
(after East Midlands Subsidence Unit (49))

APPENDIX B

LISTING OF SUBSIDENCE CALCULATION PROGRAM

INPUT NCB AREA
COLLIERY NAME
SEAM NAME
PANEL NAME
SEAM DEPTH NAME
PANEL WIDTH
EXTRACTED SEAM HEIGHT
PANEL LENGTH
SEAM DEPTH
W/H RATIO
S/M RATIO

↓

CALCULATE VALUE OF MAXIMUM SUBSIDENCE

↓

CALCULATE VALUES OF s/S AT POINTS ON TRANSVERSE SUBSIDENCE PROFILE

↓

CALCULATE NEW ABSOLUTE HEIGHT ABOVE SEAM OF TRANSVERSE SUBSIDENCE s/s POINTS

↓

CALCULATE DISTANCE OF s/S POINTS FROM FACE CENTRE LINE (TRANSVERSE PROFILE)

↓

CALCULATE S/H RATIO

↓

CALCULATE MAXIMUM TENSILE AND COMPRESSIVE STRAIN VALUES

↓

CALCULATE POSITION OF e/E VALUES ON STRAIN PROFILE (TRANSVERSE PROFILE)

↓

CALCULATE e/E STRAIN VALUES

↓

CALCULATE POSITION OF s/S VALUES ON THE LONGITUDINAL SUBSIDENCE PROFILE

↓

PRINT s/S SUBSIDENCE VALUES FOR EACH POSITION ON THE LONGITUDINAL PROFILE

↓

CALCULATE POSITION OF e/E VALUES ON THE LONGITUDINAL STRAIN PROFILE

↓

PRINT e/E STRAIN VALUES FOR EACH POSITION ON THE LONGITUDINAL PROFILE

↓

STOP

↓

END

Figure B.1 Step Procedure Chart for the Subsidence Calculation Program

```
UNIJOB IDEN :NUENR,ENXTAUEST1,BY C
OWNER FROM ENXTA-TIM
FORTRAN
       MASTER WESTOE1
       INTEGER T,V,X,Z,U,Y,XX,YY,ZZ,TT,VV,UU
       REAL M
       INTEGER   NCBAR(3),CONAM(3),SEAM(3),DSPAN(3),DEPTH(3)
       DIMENSION A(108,14),B(108,3),C(108,19),D(114,10),E(114,12),
      1R1(108,3)
       DATA NCBAR(1)/12H NORTH EAST /,CONAM(1)/12H  WESTOE     /,
      1SEAM(1)/12H  MAIN F2 - /,DSPAN(1)/12H   F 16     /,
      2DEPTH(1)/12H SEABED     /,
      3W/120.0/,M/2.40/,L/790/,H/220.0/,WH/ 0.56/,SH/0.40/
       WRITE(6,200)NCBAR,CONAM,SEAM,DSPAN,DEPTH
       WRITE(6,210)W,M,L,H
       WRITE(6,220)WH,SM
       SX=M*SH
       WRITE(6,230)SX
       SX1=0.00
       SX2=SX*0.05
       SX3=SX*0.10
       SX4=SX*0.20
       SX5=SX*0.30
       SX6=SX*0.40
       SX7=SX*0.50
       SX8=SX*0.60
       SX9=SX*0.70
       SX10=SX*0.80
       SX11=SX*0.90
       SX12=SX*0.95
       SX13=SX*1.00
       WRITE(6,240)
       WRITE(6,241)
       WRITE(6,250)
       WRITE(6,260)SX1,SX2,SX3,SX4,SX5,SX6,SX7,SX8,SX9,SX10,SX11,SX12,
      1SX13
       SXA1=H*1.0
       SXA2=H-SX2
       SXA3=H-SX3
       SXA4=H-SX4
       SXA5=H-SX5
       SXA6=H-SX6
       SXA7=H-SX7
       SXA8=H-SX8
       SXA9=H-SX9
       SXA10=H-SX10
       SXA11=H-SX11
       SXA12=H-SX12
       SXA13=H-SX13
       WRITE(6,275)SXA1,SXA2,SXA3,SXA4,SXA5,SXA6,SXA7,SXA8,SXA9,SXA10,
```

```
     1SXA11,SXA12,SXA13
      DO 9 I=1,108
      READ(5,130) (A(I,J),J=1,14)
      IF(ABS(WH-A(I,1)).GT.0.0001)GO TO 9
      WRITE(6,280) (A(I,K),K=1,14)
      DO 6 K=2,14
      A(I,K)=A(I,K)*H
    6 CONTINUE
      WRITE(6,280) (A(I,K),K=1,14)
      GO TO 13
    9 CONTINUE
   13 CONTINUE
      DO 310 IDUM=I+1,108
      READ(5,315)(A(I,J),J=1,14)
  310 CONTINUE
      SH=(SX*1000)/H
      WRITE(6,400)SH
      WRITE(6,401)
      WRITE(6,402)
      DO 20 T=1,108
      READ(5,300)(B(T,U),U=1,3)
      IF(ABS(WH-B(T,1)).GT.0.0001) GO TO 20
      WRITE(6,420)(B(T,V),V=1,3)
      DO 21 V=2,3
      B1(T,V)=B(T,V)*SH
   21 CONTINUE
      WRITE(6,430)(B1(T,V),V=1,3)
      GO TO 12
   20 CONTINUE
   12 CONTINUE
      DO 320 IDUM=T+1,108
      READ(5,325)(B(T,U),U=1,3)
  320 CONTINUE
      WRITE(6,242)
      DO 30 X=1,108
      READ(5,500) (C(X,Y),Y=1,19)
      IF(ABS(WH-C(X,1)).GT.0.0001) GO TO 30
      WRITE(6,600) (C(X,Z),Z=1,19)
      DO 31 Z=2,19
      C(X,Z)=C(X,Z)*H
   31 CONTINUE
      WRITE(6,610)(C(X,Z),Z=1,19)
      GO TO 14
   30 CONTINUE
   14 CONTINUE
      DO 330 IDUM=X+1,108
      READ(5,335) (C(X,Y),Y=1,19)
  330 CONTINUE
      F1=0.00
```

```
      E2=B1(T,2)*0.20
      E3=B1(T,2)*0.40
      E4=B1(T,2)*0.60
      E5=B1(T,2)*0.80
      E6=B1(T,2)*1.00
      E7=B1(T,2)*0.80
      E8=0.00
      E9=B1(T,3)*0.20
      E10=B1(T,3)*0.40
      E11=B1(T,3)*0.60
      E12=B1(T,3)*0.80
      E13=B1(T,3)*1.00
      E14=B1(T,3)*0.80
      E15=B1(T,3)*0.60
      E16=B1(T,3)*0.40
      E17=B1(T,3)*0.20
      E18=0.00
      WRITE(6,700)E1,E2,E3,E4,E5,E6,E7,E8,E9,E10,E11,E12,E13,E14,E15,
     1E16,E17,E18
      WRITE(6,510)
      WRITE(6,520)
      DO 50 TT=1,114
      READ(5,740) (E(TT,UU),UU=1,12)
      IF(ABS(H-E(TT,1)).GT.0.001) GO TO 50
      WRITE(6,745)(E(TT,VV),VV=1,12)
      DO 51 VV=2,12
      E(TT,VV)=E(TT,VV)*H
   51 CONTINUE
      WRITE(6,750) (E(TT,VV),VV=1,12)
      GO TO 16
   50 CONTINUE
   16 CONTINUE
      DO 350 IDUM=TT+1,114
      READ(5,355) (E(TT,UU),UU=1,12)
  350 CONTINUE
      WRITE(6,755)SX2,SX3,SX4,SX5,SX6,SX7,SX8,SX9,SX10,SX11,
     1SX12
      WRITE(6,757)SXA2,SXA3,SXA4,SXA5,SXA6,SXA7,SXA8,SXA9,SXA10,
     1SXA11,SXA12
      WRITE(6,530)
      DO 40 XX=1,114
      READ(5,710) (D(XX,YY),YY=1,16)
      IF (ABS(H-D(XX,1)).GT.0.0001) GO TO 40
      WRITE(6,720) (D(XX,ZZ),ZZ=1,16)
      DO 41 ZZ=2,16
      D(XX,ZZ)=D(XX,ZZ)*H
   41 CONTINUE
      WRITE(6,730) (D(XX,ZZ),ZZ=1,16)
      GO TO 15
   40 CONTINUE
```

```
   15 CONTINUE
      DO 360 IDUM=XX+1,114
      READ(5,345) (D(XX,YY),YY=1,16)
  340 CONTINUE
      WRITE(6,705)E1,F2,F3,E4,F5,E6,E7,E8,E9,E10,E11,E12,E13,
     1E14,F15
      STOP
  100 FORMAT(15A4)
  110 FORMAT(F6.1,F0.0,I3,F6.1)
  120 FORMAT(2F0.0)
  130 FORMAT(14F0.0)
  300 FORMAT(3F0.0)
  500 FORMAT(19F0.0)
  710 FORMAT(16F0.0)
  740 FORMAT(12F0.0)
  200 FORMAT(1H , ' NCB AREA : ',3A4,// ' COLLIERY : ',3A4,//
     1 ' SEAM : ',3A4,// ' PANEL : ',3A4,// ' HEIGHT TO  : ',3A4,/)
  210 FORMAT(1H ,' PANEL WIDTH (METRES) : ',F6.2,// ' SEAM THICKNESS
     1 (METRES) : ',F4.2,// ' PANEL LENGTH (METRES) : ',I3,//
     2 ' HEIGHT FROM SEAM (METRES) : ',F6.1,/)
  220 FORMAT(1H , ' W:D RATIO : ',F5.2,// ' S:M RATIO : ',F5.2,/)
  230 FORMAT(1H , ' MAX SUBSIDENCE (METRES) : ',F5.2,/)
  240 FORMAT(1H , ' CALC  OF  SUBSIDENCE/STRAIN  PROFILES  ',/)
  241 FORMAT(1H , ' TRANSVERSE SUBSIDENCE PROFILE (METRES) ',/)
  242 FORMAT(1H , ' TRANSVERSE STRAIN PROFILE (MM/M)  ',/)
  510 FORMAT(1H , ' CALC OF SUBSIDENCE TRAVELLING PROFILES ',/)
  520 FORMAT(1H , ' TRAVELLING SUBSIDENCE PROFILE (METRES) ',/)
  530 FORMAT(1H , ' TRAVELLING STRAIN PROFILE (MM/M) ',/)
  250 FORMAT(1H ,11X,'0.00',3X,'0.05',3X,'0.10',3X,'0.20',3X,'0.30',
     13X,'0.40',3X,'0.50',3X,'0.60',3X,'0.70',3X,'0.80',3X,'0.90',
     23X,'0.95',3X,'1.00',/)
  260 FORMAT(1H ,9X,13(F6.2,1X),/)
  275 FORMAT(1H ,9X,13(F6.2,1X),/)
  280 FORMAT(1H ,2X,14(F6.2,1X),/)
  400 FORMAT(1H , '  S:H RATIO : ',F5.2,/)
  401 FORMAT(1H , ' STRAIN-SUBSIDENCE-DEPTH RATIO ',/)
  402 FORMAT(1H , '   W:D    EXT    COMP ',/)
  420 FORMAT(1H ,2X,3(F6.2,1X),/)
  430 FORMAT(1H ,2X,3(F6.2,1X),/)
  600 FORMAT(1H ,2X,19(F6.2,1X),/)
  610 FORMAT(1H ,2X,19(F6.2,1X),/)
  700 FORMAT(1H ,9X,18(F5.2,2X),/)
  720 FORMAT(1H , 1X,F6.2,1X,15(F5.3,2X),/)
  730 FORMAT(1H , 1X,16(F6.2,1X),/)
  705 FORMAT(1H ,9X,15(F5.2,2X),/)
  745 FORMAT(1H , 1X,F6.2,2X,11(F6.3,1X),/)
  750 FORMAT(1H , 1X,12(F6.2,1X),/)
  755 FORMAT(1H ,10X,11(F4.2,3X),/)
  757 FORMAT(1H ,10X,11(F6.2,1X),/)
  315 FORMAT(14F0.0)
  325 FORMAT(3F0.0)
  335 FORMAT(19F0.0)
  345 FORMAT(16F0.0)
  355 FORMAT(12F0.0)
      STOP
      END
****
```

```
207 DATA
208 14.00 7.70 7.21 7.09 6.99 6.94 6.89 6.86 6.82 6.78 6.73 6.65 6.57 6.11
209 13.80 7.60 7.11 6.99 6.89 6.84 6.79 6.76 6.72 6.68 6.63 6.55 6.47 6.01
210 13.60 7.50 7.01 6.89 6.79 6.74 6.69 6.66 6.62 6.58 6.53 6.45 6.37 5.91
211 13.40 7.40 6.91 6.79 6.69 6.64 6.59 6.56 6.52 6.48 6.43 6.35 6.27 5.81
212 13.20 7.30 6.81 6.69 6.59 6.54 6.49 6.46 6.42 6.38 6.33 6.25 6.17 5.71
213 13.00 7.20 6.71 6.59 6.49 6.44 6.39 6.36 6.32 6.28 6.23 6.15 6.07 5.61
214 12.80 7.10 6.61 6.49 6.39 6.34 6.29 6.26 6.22 6.18 6.13 6.05 5.97 5.51
215 12.60 7.00 6.51 6.39 6.29 6.24 6.19 6.16 6.12 6.08 6.03 5.95 5.87 5.41
216 12.40 6.90 6.41 6.29 6.19 6.14 6.09 6.06 6.02 5.98 5.93 5.85 5.77 5.31
217 12.20 6.80 6.31 6.19 6.09 6.04 5.99 5.96 5.92 5.88 5.83 5.75 5.67 5.21
218 12.00 6.70 6.21 6.09 5.99 5.94 5.89 5.86 5.82 5.78 5.73 5.65 5.57 5.11
219 11.80 6.60 6.11 5.99 5.89 5.84 5.79 5.76 5.72 5.68 5.63 5.55 5.47 5.01
220 11.60 6.50 6.01 5.89 5.79 5.74 5.69 5.66 5.62 5.58 5.53 5.45 5.37 4.91
221 11.40 6.40 5.91 5.79 5.69 5.64 5.59 5.56 5.52 5.48 5.43 5.35 5.27 4.81
222 11.20 6.30 5.81 5.69 5.59 5.54 5.49 5.46 5.42 5.38 5.33 5.25 5.17 4.71
223 11.00 6.20 5.71 5.59 5.49 5.44 5.39 5.36 5.32 5.28 5.23 5.15 5.07 4.61
224 10.80 6.10 5.61 5.49 5.39 5.34 5.29 5.26 5.22 5.18 5.13 5.05 4.97 4.51
225 10.60 6.00 5.51 5.39 5.29 5.24 5.19 5.16 5.12 5.08 5.03 4.95 4.87 4.41
226 10.40 5.90 5.41 5.29 5.19 5.14 5.09 5.06 5.02 4.98 4.93 4.85 4.77 4.31
227 10.20 5.80 5.31 5.19 5.09 5.04 4.99 4.96 4.92 4.88 4.83 4.75 4.67 4.21
228 10.00 5.70 5.21 5.09 4.99 4.94 4.89 4.86 4.82 4.78 4.73 4.65 4.57 4.11
229 9.80 5.60 5.11 4.99 4.89 4.84 4.79 4.76 4.72 4.68 4.63 4.55 4.47 4.01
230 9.60 5.50 5.01 4.89 4.79 4.74 4.69 4.66 4.62 4.58 4.53 4.45 4.37 3.91
231 9.40 5.40 4.91 4.79 4.69 4.64 4.59 4.56 4.52 4.48 4.43 4.35 4.27 3.81
232 9.20 5.30 4.81 4.69 4.59 4.54 4.49 4.46 4.43 4.38 4.33 4.25 4.17 3.71
233 9.00 5.20 4.71 4.59 4.49 4.44 4.39 4.36 4.33 4.28 4.23 4.15 4.07 3.61
234 8.80 5.10 4.61 4.49 4.39 4.34 4.29 4.26 4.23 4.18 4.13 4.05 3.97 3.51
235 8.60 5.00 4.51 4.39 4.29 4.24 4.19 4.16 4.13 4.08 4.03 3.95 3.87 3.41
236 8.40 4.90 4.41 4.29 4.19 4.14 4.09 4.06 4.03 3.98 3.93 3.85 3.77 3.31
237 8.20 4.80 4.31 4.19 4.09 4.04 3.99 3.96 3.93 3.88 3.83 3.75 3.67 3.21
238 8.00 4.70 4.21 4.09 3.99 3.94 3.89 3.86 3.82 3.78 3.73 3.65 3.57 3.11
239 7.80 4.60 4.11 3.99 3.89 3.84 3.79 3.76 3.72 3.68 3.63 3.55 3.47 3.01
240 7.60 4.50 4.01 3.89 3.79 3.74 3.69 3.66 3.62 3.58 3.53 3.45 3.37 2.91
241 7.40 4.40 3.91 3.79 3.69 3.64 3.59 3.56 3.52 3.48 3.43 3.35 3.27 2.81
242 7.20 4.30 3.81 3.69 3.59 3.54 3.49 3.46 3.42 3.38 3.33 3.25 3.17 2.71
243 7.00 4.20 3.71 3.59 3.49 3.44 3.39 3.36 3.32 3.28 3.23 3.15 3.07 2.61
244 6.80 4.10 3.61 3.49 3.39 3.34 3.29 3.26 3.22 3.18 3.13 3.05 2.97 2.51
245 6.60 4.00 3.51 3.39 3.29 3.24 3.19 3.16 3.12 3.08 3.03 2.95 2.87 2.41
246 6.40 3.90 3.41 3.29 3.19 3.14 3.09 3.06 3.02 2.98 2.93 2.85 2.77 2.31
247 6.20 3.80 3.31 3.19 3.09 3.04 2.99 2.96 2.92 2.88 2.83 2.75 2.67 2.21
248 6.00 3.70 3.21 3.09 2.99 2.94 2.89 2.86 2.82 2.78 2.73 2.65 2.57 2.11
249 5.80 3.60 3.11 2.99 2.89 2.84 2.79 2.76 2.72 2.68 2.63 2.55 2.47 2.01
250 5.60 3.50 3.01 2.89 2.79 2.74 2.69 2.66 2.62 2.58 2.53 2.45 2.37 1.91
251 5.40 3.40 2.91 2.79 2.69 2.64 2.59 2.56 2.52 2.48 2.43 2.35 2.27 1.81
252 5.20 3.30 2.81 2.69 2.59 2.54 2.49 2.46 2.42 2.38 2.33 2.25 2.17 1.71
253 5.00 3.20 2.71 2.59 2.49 2.44 2.39 2.36 2.32 2.28 2.23 2.15 2.07 1.61
254 4.80 3.10 2.61 2.49 2.39 2.34 2.29 2.26 2.22 2.18 2.13 2.05 1.97 1.51
255 4.60 3.00 2.51 2.39 2.29 2.24 2.19 2.16 2.12 2.08 2.03 1.95 1.87 1.41
256 4.40 2.90 2.41 2.29 2.19 2.14 2.09 2.06 2.02 1.98 1.93 1.85 1.77 1.31
257 4.20 2.80 2.31 2.19 2.09 2.04 1.99 1.96 1.92 1.88 1.83 1.75 1.67 1.21
258 4.00 2.70 2.21 2.09 1.99 1.94 1.89 1.86 1.82 1.78 1.73 1.65 1.57 1.11
259 3.80 2.60 2.11 1.99 1.89 1.84 1.79 1.76 1.72 1.68 1.63 1.55 1.47 1.01
260 3.60 2.50 2.01 1.89 1.79 1.74 1.69 1.66 1.62 1.58 1.53 1.45 1.37 0.91
261 3.40 2.40 1.91 1.79 1.69 1.64 1.59 1.56 1.52 1.48 1.43 1.35 1.27 0.81
```

262	3.20	2.30	1.81	1.60	1.59	1.54	1.49	1.46	1.42	1.38	1.33	1.25	1.17	0.71
263	3.00	2.20	1.71	1.50	1.49	1.44	1.39	1.36	1.32	1.28	1.23	1.15	1.07	0.61
264	2.80	2.10	1.61	1.40	1.39	1.34	1.29	1.26	1.22	1.18	1.13	1.05	0.97	0.51
265	2.60	2.00	1.51	1.30	1.29	1.24	1.19	1.16	1.12	1.08	1.03	0.95	0.87	0.41
266	2.40	1.90	1.41	1.20	1.19	1.14	1.10	1.06	1.02	0.98	0.93	0.85	0.77	0.31
267	2.20	1.80	1.31	1.10	1.09	1.04	1.00	0.96	0.92	0.88	0.83	0.75	0.67	0.23
268	2.00	1.70	1.21	1.00	0.99	0.94	0.90	0.86	0.82	0.78	0.73	0.65	0.57	0.16
269	1.80	1.60	1.11	1.00	0.90	0.84	0.80	0.76	0.72	0.68	0.63	0.55	0.47	0.10
270	1.60	1.50	1.01	0.90	0.80	0.74	0.70	0.66	0.62	0.58	0.53	0.45	0.37	0.05
271	1.40	1.40	0.91	0.80	0.70	0.64	0.60	0.56	0.52	0.48	0.43	0.35	0.27	0.01
272	1.30	1.35	0.86	0.75	0.65	0.59	0.55	0.51	0.47	0.43	0.38	0.30	0.23	0.00
273	1.20	1.30	0.81	0.70	0.60	0.54	0.50	0.46	0.42	0.38	0.33	0.25	0.19	0.00
274	1.10	1.25	0.77	0.66	0.55	0.50	0.45	0.42	0.38	0.34	0.28	0.21	0.16	0.00
275	1.00	1.20	0.72	0.61	0.51	0.45	0.41	0.37	0.33	0.29	0.24	0.18	0.13	0.00
276	0.98	1.19	0.71	0.60	0.50	0.44	0.40	0.36	0.33	0.28	0.24	0.17	0.12	0.00
277	0.96	1.18	0.70	0.59	0.49	0.43	0.39	0.35	0.32	0.27	0.23	0.16	0.11	0.00
278	0.94	1.17	0.69	0.58	0.48	0.42	0.38	0.34	0.31	0.26	0.22	0.16	0.11	0.00
279	0.92	1.16	0.68	0.57	0.47	0.41	0.37	0.35	0.30	0.26	0.21	0.15	0.10	0.00
280	0.90	1.15	0.68	0.57	0.46	0.40	0.36	0.32	0.29	0.25	0.20	0.14	0.10	0.00
281	0.88	1.14	0.67	0.56	0.45	0.40	0.36	0.32	0.28	0.24	0.20	0.13	0.10	0.00
282	0.86	1.13	0.66	0.55	0.45	0.39	0.35	0.31	0.27	0.23	0.19	0.13	0.09	0.00
283	0.84	1.12	0.65	0.54	0.44	0.38	0.34	0.30	0.26	0.22	0.18	0.12	0.09	0.00
284	0.82	1.11	0.64	0.53	0.43	0.37	0.33	0.29	0.26	0.21	0.17	0.12	0.08	0.00
285	0.80	1.10	0.63	0.52	0.42	0.36	0.32	0.28	0.25	0.21	0.17	0.11	0.08	0.00
286	0.78	1.09	0.63	0.52	0.42	0.36	0.32	0.28	0.24	0.20	0.16	0.11	0.08	0.00
287	0.76	1.08	0.62	0.51	0.41	0.35	0.31	0.27	0.23	0.20	0.16	0.11	0.07	0.00
288	0.74	1.07	0.61	0.50	0.40	0.34	0.30	0.26	0.23	0.19	0.15	0.10	0.07	0.00
289	0.72	1.06	0.61	0.50	0.39	0.34	0.30	0.26	0.22	0.18	0.15	0.10	0.07	0.00
290	0.70	1.05	0.60	0.49	0.39	0.33	0.29	0.25	0.21	0.18	0.14	0.10	0.07	0.00
291	0.68	1.04	0.60	0.49	0.38	0.32	0.28	0.24	0.21	0.17	0.14	0.10	0.07	0.00
292	0.66	1.03	0.60	0.48	0.38	0.32	0.28	0.24	0.20	0.17	0.14	0.09	0.06	0.00
293	0.64	1.02	0.59	0.48	0.37	0.31	0.27	0.23	0.20	0.17	0.13	0.09	0.06	0.00
294	0.62	1.01	0.59	0.47	0.37	0.31	0.27	0.23	0.19	0.16	0.13	0.09	0.06	0.00
295	0.60	1.00	0.59	0.47	0.36	0.30	0.26	0.22	0.19	0.16	0.13	0.09	0.06	0.00
296	0.58	0.99	0.59	0.47	0.35	0.30	0.25	0.22	0.18	0.16	0.13	0.09	0.06	0.00
297	0.56	0.98	0.59	0.47	0.35	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0.00
298	0.54	0.97	0.59	0.47	0.34	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0.00
299	0.52	0.96	0.59	0.47	0.34	0.28	0.24	0.21	0.17	0.15	0.12	0.08	0.06	0.00
300	0.50	0.95	0.59	0.47	0.34	0.28	0.24	0.21	0.17	0.15	0.12	0.08	0.06	0.00
301	0.48	0.94	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0.00
302	0.46	0.93	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0.00
303	0.44	0.92	0.59	0.47	0.33	0.28	0.23	0.20	0.17	0.15	0.12	0.08	0.06	0.00
304	0.42	0.91	0.59	0.47	0.34	0.28	0.24	0.20	0.17	0.15	0.12	0.08	0.06	0.00
305	0.40	0.90	0.59	0.47	0.34	0.28	0.24	0.21	0.18	0.15	0.12	0.08	0.06	0.00
306	0.38	0.89	0.60	0.48	0.35	0.29	0.24	0.21	0.18	0.15	0.12	0.08	0.06	0.00
307	0.36	0.88	0.60	0.48	0.35	0.29	0.25	0.21	0.18	0.15	0.12	0.08	0.06	0.00
308	0.34	0.87	0.60	0.49	0.36	0.30	0.25	0.22	0.19	0.16	0.12	0.08	0.06	0.00
309	0.32	0.86	0.60	0.49	0.37	0.31	0.26	0.22	0.20	0.16	0.13	0.09	0.06	0.00
310	0.30	0.85	0.61	0.50	0.38	0.32	0.27	0.23	0.20	0.17	0.13	0.09	0.06	0.00
311	0.28	0.84	0.61	0.51	0.39	0.33	0.28	0.24	0.21	0.18	0.14	0.09	0.07	0.00
312	0.26	0.83	0.62	0.52	0.41	0.35	0.30	0.26	0.22	0.19	0.15	0.10	0.07	0.00
313	0.24	0.82	0.62	0.53	0.43	0.36	0.32	0.28	0.24	0.20	0.16	0.11	0.07	0.00
314	0.22	0.81	0.63	0.53	0.46	0.39	0.34	0.30	0.26	0.21	0.17	0.11	0.08	0.00
315	0.20	0.80	0.64	0.57	0.48	0.41	0.37	0.32	0.28	0.23	0.19	0.13	0.08	0.00

316	14.00	0.65	0.50
317	13.80	0.65	0.50
318	13.60	0.65	0.50
319	13.40	0.65	0.50
320	13.20	0.65	0.50
321	13.00	0.65	0.50
322	12.80	0.65	0.50
323	12.60	0.65	0.50
324	12.40	0.65	0.50
325	12.20	0.65	0.50
326	12.00	0.65	0.50
327	11.80	0.65	0.50
328	11.60	0.65	0.50
329	11.40	0.65	0.50
330	11.20	0.65	0.50
331	11.00	0.65	0.50
332	10.80	0.65	0.50
333	10.60	0.65	0.50
334	10.40	0.65	0.50
335	10.20	0.65	0.50
336	10.00	0.65	0.50
337	9.80	0.65	0.50
338	9.60	0.65	0.50
339	9.40	0.65	0.50
340	9.20	0.65	0.50
341	9.00	0.65	0.50
342	8.80	0.65	0.50
343	8.60	0.65	0.50
344	8.40	0.65	0.50
345	8.20	0.65	0.50
346	8.00	0.65	0.50
347	7.80	0.65	0.50
348	7.60	0.65	0.50
349	7.40	0.65	0.50
350	7.20	0.65	0.50
351	7.00	0.65	0.50
352	6.80	0.65	0.50
353	6.60	0.65	0.50
354	6.40	0.65	0.50
355	6.20	0.65	0.50
356	6.00	0.65	0.50
357	5.80	0.65	0.50
358	5.60	0.65	0.50
359	5.40	0.65	0.50
360	5.20	0.65	0.50
361	5.00	0.65	0.50
362	4.80	0.65	0.50
363	4.60	0.65	0.50
364	4.40	0.65	0.50
365	4.20	0.65	0.50
366	4.00	0.65	0.50
367	3.80	0.65	0.50
368	3.60	0.65	0.50
369	3.40	0.65	0.50

370	3.20	0″.65	0″.50
371	3.00	0″.65	0″.50
372	2.80	0″.65	0″.50
373	2.60	0″.65	0″.50
374	2.40	0″.65	0″.50
375	2.20	0″.65	0″.50
376	2.00	0″.65	0″.50
377	1.80	0″.65	0″.50
378	1.60	0″.65	0″.50
379	1.40	0″.65	0″.50
380	1.30	0″.65	0″.50
381	1.20	0″.65	0″.50
382	1.10	0″.65	0″.55
383	1.00	0″.65	0″.55
384	0.98	0″.65	0″.60
385	0.96	0″.65	0″.60
386	0.94	0″.65	0″.60
387	0.92	0″.65	0″.60
388	0.90	0″.65	0″.60
389	0.88	0″.65	0″.60
390	0.86	0″.65	0″.60
391	0.84	0″.65	0″.65
392	0.82	0″.65	0″.70
393	0.80	0″.65	0″.70
394	0.78	0″.65	0″.70
395	0.76	0″.65	0″.75
396	0.74	0″.70	0″.80
397	0.72	0″.70	0″.80
398	0.70	0″.70	0″.85
399	0.68	0″.70	0″.90
400	0.66	0″.70	0″.90
401	0.64	0″.75	0″.95
402	0.62	0″.75	1″.00
403	0.60	0″.80	1″.05
404	0.58	0″.80	1″.10
405	0.56	0″.80	1″.15
406	0.54	0″.80	1″.20
407	0.52	0″.80	1″.30
408	0.50	0″.80	1″.40
409	0.48	0″.80	1″.45
410	0.46	0″.75	1″.50
411	0.44	0″.80	1″.60
412	0.42	0″.80	1″.65
413	0.40	0″.80	1″.70
414	0.38	0″.80	1″.80
415	0.36	0″.80	1″.85
416	0.34	0″.75	1″.90
417	0.32	0″.70	1″.95
418	0.30	0″.70	2″.00
419	0.28	0″.65	2″.05
420	0.26	0″.65	2″.10
421	0.24	0″.60	2″.20
422	0.22	0″.60	2″.20
423	0.20	0″.55	2″.25

424	14.0	7.7	7.3	7.2	7.1	7.0	7.0	6.9	6.8	6.8	6.7	6.7	6.7	6.7	6.6	6.5	6.4	6.4	6.2
425	13.8	7.6	7.2	7.1	7.0	6.9	6.9	6.8	6.7	6.7	6.6	6.6	6.6	6.6	6.5	6.4	6.3	6.2	6.1
426	13.6	7.5	7.1	7.0	6.9	6.8	6.8	6.7	6.6	6.6	6.5	6.5	6.5	6.5	6.4	6.3	6.2	6.1	6.0
427	13.4	7.4	7.0	6.9	6.8	6.7	6.6	6.6	6.5	6.5	6.4	6.4	6.4	6.4	6.3	6.2	6.1	6.0	5.9
428	13.2	7.3	6.9	6.7	6.7	6.6	6.5	6.5	6.4	6.4	6.4	6.3	6.3	6.3	6.2	6.1	6.0	6.0	5.8
429	13.0	7.2	6.8	6.6	6.6	6.5	6.4	6.4	6.3	6.3	6.3	6.2	6.2	6.2	6.1	6.0	5.9	5.9	5.7
430	12.8	7.1	6.7	6.5	6.5	6.4	6.3	6.3	6.2	6.2	6.2	6.1	6.1	6.1	6.0	5.9	5.8	5.8	5.6
431	12.6	7.0	6.6	6.4	6.4	6.3	6.2	6.2	6.1	6.1	6.1	6.0	6.0	6.0	5.9	5.8	5.7	5.7	5.5
432	12.4	6.9	6.5	6.3	6.3	6.2	6.1	6.1	6.0	6.0	6.0	5.9	5.9	5.9	5.8	5.7	5.6	5.6	5.4
433	12.2	6.8	6.4	6.2	6.2	6.1	6.1	6.0	5.9	5.9	5.9	5.8	5.8	5.8	5.7	5.6	5.5	5.5	5.3
434	12.0	6.7	6.3	6.1	6.1	6.0	6.0	5.9	5.8	5.8	5.8	5.7	5.7	5.7	5.6	5.5	5.4	5.4	5.2
435	11.8	6.6	6.2	6.0	6.0	5.9	5.9	5.8	5.7	5.7	5.7	5.6	5.6	5.6	5.5	5.4	5.3	5.3	5.1
436	11.6	6.5	6.1	5.9	5.9	5.8	5.8	5.7	5.6	5.6	5.6	5.5	5.5	5.5	5.4	5.3	5.2	5.2	5.0
437	11.4	6.4	6.0	5.8	5.8	5.7	5.7	5.6	5.5	5.5	5.5	5.4	5.4	5.4	5.3	5.2	5.1	5.1	4.9
438	11.2	6.3	5.9	5.7	5.7	5.6	5.6	5.5	5.4	5.4	5.4	5.3	5.3	5.3	5.2	5.1	5.0	5.0	4.8
439	11.0	6.2	5.8	5.6	5.6	5.5	5.5	5.4	5.3	5.3	5.3	5.2	5.2	5.2	5.1	5.0	4.9	4.9	4.7
440	10.8	6.1	5.7	5.5	5.5	5.4	5.4	5.3	5.2	5.2	5.2	5.1	5.1	5.1	5.0	4.9	4.8	4.8	4.6
441	10.6	6.0	5.6	5.4	5.4	5.3	5.3	5.2	5.1	5.1	5.1	5.0	5.0	5.0	4.9	4.8	4.7	4.7	4.5
442	10.4	5.9	5.5	5.3	5.3	5.2	5.2	5.1	5.0	5.0	5.0	4.9	4.9	4.9	4.8	4.7	4.6	4.6	4.4
443	10.2	5.8	5.4	5.2	5.2	5.1	5.0	5.0	4.9	4.9	4.9	4.8	4.8	4.8	4.7	4.6	4.5	4.5	4.3
444	10.0	5.7	5.3	5.2	5.1	5.0	4.9	4.9	4.8	4.8	4.8	4.7	4.7	4.7	4.6	4.5	4.4	4.4	4.2
445	9.8	5.6	5.2	5.1	5.0	4.9	4.8	4.8	4.7	4.7	4.7	4.6	4.6	4.6	4.5	4.4	4.3	4.3	4.1
446	9.6	5.5	5.1	5.0	4.9	4.8	4.7	4.7	4.6	4.6	4.6	4.5	4.5	4.5	4.4	4.3	4.2	4.2	4.0
447	9.4	5.4	5.0	4.9	4.8	4.7	4.6	4.6	4.5	4.5	4.5	4.4	4.4	4.4	4.3	4.2	4.1	4.1	3.9
448	9.2	5.3	4.9	4.8	4.7	4.6	4.6	4.5	4.4	4.4	4.4	4.3	4.3	4.3	4.2	4.1	4.0	4.0	3.8
449	9.0	5.2	4.8	4.7	4.6	4.5	4.5	4.4	4.3	4.3	4.3	4.2	4.2	4.2	4.1	4.0	3.9	3.9	3.7
450	8.8	5.1	4.7	4.6	4.5	4.4	4.4	4.3	4.2	4.2	4.2	4.1	4.1	4.1	4.0	3.9	3.8	3.8	3.6
451	8.6	5.0	4.6	4.5	4.4	4.3	4.3	4.2	4.1	4.1	4.1	4.0	4.0	4.0	3.9	3.8	3.7	3.7	3.5
452	8.4	4.9	4.5	4.4	4.3	4.2	4.2	4.1	4.0	4.0	4.0	3.9	3.9	3.9	3.8	3.7	3.6	3.6	3.4
453	8.2	4.8	4.4	4.3	4.2	4.1	4.1	4.0	3.9	3.9	3.9	3.8	3.8	3.8	3.7	3.6	3.5	3.5	3.3
454	8.0	4.7	4.3	4.2	4.1	4.0	4.0	3.9	3.8	3.8	3.8	3.8	3.7	3.7	3.6	3.5	3.4	3.4	3.2
455	7.8	4.6	4.2	4.1	4.0	3.9	3.9	3.8	3.7	3.7	3.7	3.7	3.6	3.6	3.5	3.4	3.3	3.3	3.1
456	7.6	4.5	4.1	4.0	3.9	3.8	3.8	3.7	3.6	3.6	3.6	3.6	3.5	3.5	3.4	3.3	3.2	3.2	3.0
457	7.4	4.4	4.0	3.9	3.8	3.7	3.7	3.6	3.5	3.5	3.5	3.5	3.4	3.4	3.3	3.2	3.1	3.1	2.9
458	7.2	4.3	3.9	3.8	3.7	3.6	3.6	3.5	3.4	3.4	3.4	3.4	3.3	3.3	3.2	3.1	3.0	3.0	2.8
459	7.0	4.2	3.8	3.7	3.6	3.5	3.5	3.4	3.3	3.3	3.3	3.3	3.2	3.2	3.1	3.0	2.9	2.9	2.7
460	6.8	4.1	3.7	3.6	3.5	3.4	3.4	3.3	3.2	3.2	3.2	3.2	3.1	3.1	3.0	2.9	2.8	2.8	2.6
461	6.6	4.0	3.6	3.5	3.4	3.3	3.3	3.2	3.1	3.1	3.1	3.1	3.0	3.0	2.9	2.8	2.7	2.7	2.5
462	6.4	3.9	3.5	3.4	3.3	3.2	3.2	3.1	3.0	3.0	3.0	3.0	2.9	2.9	2.8	2.7	2.6	2.6	2.4
463	6.2	3.8	3.4	3.3	3.2	3.1	3.1	3.0	2.9	2.9	2.9	2.9	2.8	2.8	2.7	2.6	2.5	2.5	2.3
464	6.0	3.7	3.3	3.2	3.1	3.0	3.0	2.9	2.8	2.8	2.8	2.8	2.7	2.7	2.6	2.5	2.4	2.4	2.2
465	5.8	3.6	3.2	3.1	3.0	2.9	2.9	2.8	2.7	2.7	2.7	2.7	2.6	2.6	2.5	2.4	2.3	2.3	2.1
466	5.6	3.5	3.1	3.0	2.9	2.8	2.8	2.7	2.6	2.6	2.6	2.6	2.5	2.5	2.4	2.3	2.2	2.2	2.0
467	5.4	3.4	3.0	2.9	2.8	2.7	2.7	2.6	2.5	2.5	2.5	2.5	2.4	2.4	2.3	2.2	2.1	2.1	1.9
468	5.2	3.3	2.9	2.8	2.7	2.6	2.6	2.5	2.4	2.4	2.4	2.4	2.3	2.3	2.2	2.1	2.0	2.0	1.8
469	5.0	3.2	2.8	2.7	2.6	2.5	2.5	2.4	2.3	2.3	2.3	2.3	2.2	2.2	2.1	2.0	1.9	1.9	1.7
470	4.8	3.1	2.7	2.6	2.5	2.4	2.4	2.3	2.2	2.2	2.2	2.2	2.1	2.1	2.0	1.9	1.8	1.8	1.6
471	4.6	3.0	2.6	2.5	2.4	2.3	2.3	2.2	2.1	2.1	2.1	2.1	2.0	2.0	1.9	1.8	1.7	1.7	1.5
472	4.4	2.9	2.5	2.4	2.3	2.2	2.2	2.1	2.0	2.0	2.0	2.0	1.9	1.9	1.8	1.7	1.6	1.6	1.4
473	4.2	2.8	2.4	2.3	2.2	2.1	2.1	2.0	1.9	1.9	1.9	1.8	1.8	1.8	1.7	1.6	1.5	1.5	1.3
474	4.0	2.7	2.3	2.2	2.1	2.0	2.0	1.9	1.8	1.8	1.8	1.7	1.7	1.7	1.6	1.5	1.5	1.4	1.2
475	3.8	2.6	2.2	2.1	2.0	1.9	1.9	1.8	1.7	1.7	1.7	1.6	1.6	1.6	1.5	1.4	1.4	1.3	1.1
476	3.6	2.5	2.1	2.0	1.9	1.8	1.8	1.7	1.6	1.6	1.6	1.5	1.5	1.5	1.4	1.3	1.3	1.2	1.0
477	3.4	2.4	2.0	1.9	1.8	1.7	1.7	1.6	1.5	1.5	1.5	1.4	1.4	1.4	1.3	1.2	1.2	1.1	0.9

478	3.2	2.3	1.9	1.8	1.7	1.7	1.6	1.5	1.5	1.4	1.4	1.4	1.3	1.3	1.2	1.1	1.1	1.0	0.8
479	3.0	2.2	1.8	1.7	1.6	1.6	1.5	1.5	1.4	1.3	1.3	1.3	1.2	1.2	1.1	1.0	1.0	0.9	0.7
480	2.8	2.1	1.7	1.6	1.5	1.5	1.4	1.4	1.3	1.2	1.2	1.2	1.1	1.1	1.0	.95	.88	0.8	0.6
481	2.0	2.0	1.6	1.5	1.4	1.4	1.3	1.3	1.2	1.1	1.1	1.1	1.0	[illegible]	.9	.83	.77	.7	.65
482	2.4	1.9	1.5	1.4	1.3	1.3	1.2	1.2	1.1	1.0	1.0	1.0	0.9	.9	.8	.73	.67	.6	.54
483	2.2	1.8	1.4	1.3	1.2	1.2	1.1	1.1	1.0	0.9	0.9	0.9	0.8	.8	.7	.63	.57	.5	.43
484	2.0	1.7	1.3	1.2	1.1	1.1	1.0	1.0	0.9	0.8	0.8	0.8	0.7	.7	.6	.50	.47	.4	.32
485	1.8	1.6	1.2	1.1	1.0	.95	.90	.86	.76	.73	.71	.68	.65	.59	.5	.4	.4	.3	0.1
486	1.6	1.5	1.1	1.0	.91	.85	.80	.76	.66	.63	.61	.58	.55	.49	.4	.3	.3	.2	0.0
487	1.4	1.4	.98	.87	.81	.75	.70	.66	.56	.53	.51	.48	.45	.39	.3	.2	.2	.1	0.0
488	1.3	1.35	.93	.82	.76	.7	.65	.61	.51	.49	.46	.43	.4	.34	.25	.18	.12	.05	0.0
489	1.2	1.3	.88	.77	.71	.66	.61	.56	.46	.44	.41	.38	.35	.29	.2	.13	.07	.02	0.0
490	1.1	1.25	.83	.72	.66	.61	.56	.52	.42	.39	.37	.33	.31	.24	.12	.09	.03	0.0	0.0
491	1.0	1.2	.79	.68	.62	.57	.51	.47	.37	.35	.32	.29	.26	.2	.1	.05	0.0	0.0	0.0
492	0.98	1.19	.78	.67	.61	.56	.5	.46	.36	.34	.31	.28	.25	.19	.09	.04	0.0	0.0	0.0
493	0.96	1.18	.77	.66	.6	.55	.49	.45	.35	.33	.3	.27	.24	.18	.09	.04	0.0	0.0	0.0
494	0.94	1.17	.76	.65	.59	.54	.48	.44	.35	.32	.3	.26	.23	.17	.08	.03	0.0	0.0	0.0
495	0.92	1.16	.75	.64	.58	.53	.47	.43	.34	.31	.29	.25	.22	.16	.07	.02	0.0	0.0	0.0
496	0.90	1.15	.74	.63	.57	.52	.46	.42	.33	.30	.28	.24	.21	.15	.06	.02	0.0	0.0	0.0
497	0.88	1.14	.73	.62	.56	.51	.46	.41	.32	.29	.27	.24	.21	.15	.05	.01	0.0	0.0	0.0
498	0.86	1.13	.72	.61	.55	.50	.45	.40	.31	.29	.26	.23	.20	.14	.05	0.0	0.0	0.0	0.0
499	0.84	1.12	.71	.60	.54	.49	.44	.39	.30	.28	.25	.22	.19	.13	.04	0.0	0.0	0.0	0.0
500	0.82	1.11	.70	.59	.53	.48	.43	.38	.29	.27	.25	.21	.18	.12	.03	0.0	0.0	0.0	0.0
501	0.80	1.10	.69	.58	.53	.48	.42	.37	.29	.26	.24	.20	.17	.11	.02	0.0	0.0	0.0	0.0
502	0.78	1.09	.68	.57	.52	.47	.41	.36	.28	.26	.23	.20	.17	.11	.02	0.0	0.0	0.0	0.0
503	0.76	1.08	.67	.57	.51	.46	.40	.36	.27	.25	.22	.19	.16	.10	.01	0.0	0.0	0.0	0.0
504	0.74	1.07	.67	.56	.50	.45	.39	.35	.26	.24	.22	.18	.15	.09	.01	0.0	0.0	0.0	0.0
505	0.72	1.06	.66	.55	.49	.44	.38	.34	.26	.24	.21	.17	.15	.09	0.0	0.0	0.0	0.0	0.0
506	0.70	1.05	.65	.54	.48	.44	.37	.33	.25	.23	.20	.17	.14	.08	0.0	0.0	0.0	0.0	0.0
507	0.68	1.04	.64	.54	.47	.43	.37	.32	.24	.22	.20	.16	.13	.07	0.0	0.0	0.0	0.0	0.0
508	0.66	1.03	.64	.53	.47	.42	.36	.31	.24	.22	.19	.16	.13	.07	0.0	0.0	0.0	0.0	0.0
509	0.64	1.02	.63	.53	.46	.41	.35	.31	.23	.21	.19	.15	.12	.06	0.0	0.0	0.0	0.0	0.0
510	0.62	1.01	.63	.52	.45	.41	.34	.30	.23	.21	.18	.15	.12	.05	0.0	0.0	0.0	0.0	0.0
511	0.60	1.00	.62	.52	.45	.40	.34	.29	.22	.20	.18	.14	.11	.05	0.0	0.0	0.0	0.0	0.0
512	0.58	0.99	.62	.51	.44	.39	.33	.29	.22	.19	.17	.14	.11	.04	0.0	0.0	0.0	0.0	0.0
513	0.56	0.98	.61	.51	.44	.39	.33	.28	.22	.19	.17	.13	.10	.03	0.0	0.0	0.0	0.0	0.0
514	0.54	0.97	.61	.51	.43	.39	.32	.28	.21	.19	.16	.13	.09	.03	0.0	0.0	0.0	0.0	0.0
515	0.52	0.96	.60	.51	.43	.38	.32	.27	.21	.18	.16	.12	.09	.02	0.0	0.0	0.0	0.0	0.0
516	0.50	0.95	.60	.51	.43	.38	.32	.27	.21	.18	.16	.12	.08	.02	0.0	0.0	0.0	0.0	0.0
517	0.48	0.94	.60	.51	.43	.38	.31	.27	.20	.18	.15	.12	.08	.01	0.0	0.0	0.0	0.0	0.0
518	0.46	0.93	.60	.51	.43	.38	.31	.27	.20	.18	.15	.11	.08	.01	0.0	0.0	0.0	0.0	0.0
519	0.44	0.92	.60	.51	.43	.39	.31	.27	.20	.18	.15	.11	.07	.01	0.0	0.0	0.0	0.0	0.0
520	0.42	0.91	.60	.51	.44	.39	.31	.27	.20	.18	.15	.11	.07	0.0	0.0	0.0	0.0	0.0	0.0
521	0.40	0.90	.61	.52	.45	.40	.32	.28	.21	.18	.15	.11	.07	0.0	0.0	0.0	0.0	0.0	0.0
522	0.38	0.89	.61	.53	.45	.41	.32	.28	.21	.18	.15	.11	.07	0.0	0.0	0.0	0.0	0.0	0.0
523	0.36	0.88	.62	.53	.46	.42	.33	.29	.21	.18	.15	.11	.07	0.0	0.0	0.0	0.0	0.0	0.0
524	0.34	0.87	.62	.54	.48	.43	.34	.30	.22	.19	.15	.11	.07	0.0	0.0	0.0	0.0	0.0	0.0
525	0.32	0.86	.63	.55	.49	.45	.35	.30	.22	.19	.16	.12	.07	0.0	0.0	0.0	0.0	0.0	0.0
526	0.30	0.85	.65	.57	.51	.47	.37	.32	.23	.20	.16	.12	.08	0.0	0.0	0.0	0.0	0.0	0.0
527	0.28	0.84	.66	.58	.54	.49	.39	.33	.24	.21	.17	.13	.08	0.0	0.0	0.0	0.0	0.0	0.0
528	0.26	0.83	.68	.60	.57	.51	.41	.35	.26	.22	.18	.14	.09	0.0	0.0	0.0	0.0	0.0	0.0
529	0.24	0.82	.70	.63	.60	.54	.44	.37	.28	.23	.20	.15	.10	0.0	0.0	0.0	0.0	0.0	0.0
530	0.22	0.81	.72	.66	.63	.58	.47	.39	.30	.23	.21	.16	.11	0.0	0.0	0.0	0.0	0.0	0.0
531	0.20	0.80	.74	.69	.66	.61	.49	.42	.32	.27	.23	.18	.12	0.0	0.0	0.0	0.0	0.0	0.0

532	10.0	.140	.255	.355	.430	.490	.540	.590	.680	.755	.785	.970
533	11.0	.134	.249	.349	.423	.483	.532	.583	.672	.747	.781	.965
534	12.0	.128	.243	.343	.416	.476	.524	.576	.664	.739	.777	.960
535	13.0	.122	.237	.337	.409	.469	.516	.569	.656	.731	.773	.955
536	14.0	.116	.231	.331	.402	.462	.508	.562	.648	.723	.769	.950
537	15.0	.110	.225	.325	.395	.455	.500	.555	.640	.715	.765	.945
538	16.0	.105	.220	.320	.389	.449	.494	.550	.633	.708	.761	.940
539	17.0	.100	.215	.315	.383	.443	.488	.545	.626	.701	.757	.935
540	18.0	.095	.210	.310	.377	.437	.482	.540	.619	.694	.753	.930
541	19.0	.090	.205	.305	.371	.431	.476	.535	.612	.687	.749	.925
542	20.0	.085	.200	.300	.365	.425	.470	.530	.605	.680	.745	.920
543	21.0	.082	.195	.296	.360	.420	.465	.525	.599	.674	.742	.916
544	22.0	.079	.190	.292	.355	.415	.460	.522	.593	.668	.739	.912
545	23.0	.076	.185	.288	.350	.410	.455	.518	.587	.662	.736	.908
546	24.0	.073	.180	.284	.345	.405	.450	.514	.581	.656	.733	.904
547	25.0	.070	.175	.280	.340	.400	.445	.510	.575	.650	.730	.900
548	26.0	.067	.172	.278	.337	.396	.442	.507	.571	.646	.728	.895
549	27.0	.064	.169	.276	.334	.392	.439	.504	.567	.642	.726	.890
550	28.0	.061	.166	.274	.331	.388	.436	.501	.563	.638	.724	.885
551	29.0	.058	.163	.272	.328	.384	.433	.498	.559	.634	.722	.880
552	30.0	.055	.160	.270	.325	.380	.430	.495	.555	.630	.720	.875
553	31.0	.053	.157	.267	.322	.377	.427	.492	.552	.627	.718	.870
554	32.0	.051	.154	.264	.319	.375	.424	.489	.549	.624	.716	.865
555	33.0	.049	.151	.261	.316	.372	.421	.486	.546	.621	.714	.860
556	34.0	.047	.148	.258	.313	.369	.418	.483	.543	.618	.712	.855
557	35.0	.045	.145	.255	.310	.365	.415	.480	.540	.615	.710	.850
558	36.0	.043	.143	.253	.308	.362	.413	.478	.538	.612	.708	.846
559	37.0	.041	.141	.251	.306	.359	.411	.476	.536	.609	.706	.842
560	38.0	.039	.139	.249	.304	.356	.409	.474	.534	.606	.704	.838
561	39.0	.037	.137	.247	.302	.353	.407	.472	.532	.603	.702	.834
562	40.0	.035	.135	.245	.300	.350	.405	.470	.530	.600	.700	.830
563	41.0	.033	.133	.243	.298	.348	.403	.468	.527	.598	.698	.827
564	42.0	.031	.131	.241	.296	.346	.401	.466	.524	.596	.696	.824
565	43.0	.029	.129	.239	.294	.344	.399	.464	.521	.594	.694	.821
566	44.0	.027	.127	.237	.292	.342	.397	.462	.518	.592	.692	.818
567	45.0	.025	.125	.235	.290	.340	.395	.460	.515	.590	.690	.815
568	46.0	.024	.123	.233	.288	.338	.393	.458	.513	.587	.688	.813
569	47.0	.023	.121	.231	.286	.336	.391	.456	.511	.584	.686	.811
570	48.0	.022	.119	.229	.284	.334	.389	.454	.509	.581	.684	.809
571	49.0	.021	.117	.227	.282	.332	.387	.452	.507	.577	.682	.807
572	50.0	.020	.115	.225	.280	.330	.385	.450	.505	.575	.680	.805
573	51.0	.018	.113	.223	.278	.329	.383	.448	.503	.574	.670	.803
574	52.0	.016	.111	.221	.276	.328	.381	.446	.501	.573	.673	.801
575	53.0	.014	.109	.219	.274	.327	.379	.444	.499	.572	.677	.799
576	54.0	.012	.107	.217	.272	.326	.377	.442	.497	.571	.676	.797
577	55.0	.010	.105	.215	.270	.325	.375	.440	.495	.570	.675	.795
578	56.0	.008	.103	.214	.268	.323	.374	.438	.494	.568	.673	.793
579	57.0	.006	.101	.213	.266	.321	.373	.436	.493	.566	.671	.791
580	58.0	.004	.099	.212	.264	.319	.372	.434	.492	.564	.600	.789
581	59.0	.002	.097	.211	.262	.317	.371	.432	.491	.562	.607	.787
582	60.0	.000	.095	.210	.260	.315	.370	.430	.490	.560	.605	.785
583	61.0	.001	.094	.209	.259	.313	.369	.428	.488	.559	.604	.783
584	62.0	.002	.093	.208	.258	.311	.368	.426	.486	.558	.603	.781
585	63.0	.003	.092	.204	.257	.309	.367	.424	.484	.557	.602	.779
586	64.0	.004	.091	.202	.256	.307	.366	.422	.482	.556	.601	.777
587	65.0	.005	.090	.200	.255	.305	.365	.420	.480	.555	.600	.775
588	66.0	.007	.088	.199	.254	.304	.363	.419	.478	.554	.650	.774

589	67.0	.009	.086	.198	.253	.303	.361	.418	.476	.553	.658	.773
590	68.0	.011	.084	.197	.252	.302	.359	.417	.474	.552	.657	.772
591	69.0	.013	.082	.196	.251	.301	.357	.416	.472	.551	.656	.771
592	70.0	.015	.080	.195	.250	.300	.355	.415	.470	.550	.655	.770
593	71.0	.016	.079	.193	.248	.298	.354	.414	.469	.548	.653	.768
594	72.0	.017	.078	.191	.246	.296	.353	.413	.468	.546	.651	.766
595	73.0	.018	.077	.189	.244	.294	.352	.412	.467	.544	.649	.764
596	74.0	.019	.076	.187	.242	.292	.351	.411	.466	.542	.647	.762
597	75.0	.020	.075	.185	.240	.290	.350	.410	.465	.540	.645	.760
598	80.0	.025	.070	.180	.235	.285	.345	.400	.455	.535	.640	.755
599	85.0	.030	.060	.175	.230	.280	.335	.395	.450	.530	.635	.750
600	90.0	.040	.055	.165	.225	.275	.330	.390	.445	.525	.630	.745
601	95.0	.045	.050	.160	.220	.270	.325	.385	.440	.520	.625	.740
602	100.0	.050	.050	.155	.215	.265	.320	.380	.435	.515	.620	.735
603	110.0	.060	.040	.145	.205	.255	.310	.370	.425	.505	.610	.725
604	120.0	.070	.030	.140	.195	.250	.305	.360	.415	.495	.600	.715
605	130.0	.080	.025	.130	.190	.240	.295	.350	.405	.485	.590	.705
606	140.0	.085	.015	.125	.185	.235	.290	.340	.400	.450	.580	.695
607	150.0	.095	.010	.115	.175	.225	.285	.335	.390	.470	.575	.690
608	160.0	.100	.005	.110	.170	.220	.275	.330	.385	.465	.565	.680
609	170.0	.110	.000	.100	.165	.215	.270	.325	.380	.460	.560	.675
610	180.0	.115	.005	.095	.160	.210	.265	.320	.375	.455	.555	.670
611	190.0	.120	.010	.090	.155	.205	.260	.315	.370	.450	.550	.665
612	200.0	.125	.015	.085	.150	.205	.255	.310	.370	.445	.545	.660
613	210.0	.130	.020	.080	.145	.200	.250	.305	.365	.440	.540	.655
614	220.0	.135	.020	.075	.140	.195	.250	.300	.360	.440	.535	.650
615	230.0	.135	.025	.070	.140	.190	.245	.300	.355	.435	.535	.650
616	240.0	.140	.030	.070	.135	.190	.240	.295	.355	.430	.530	.645
617	250.0	.145	.030	.065	.135	.185	.240	.290	.350	.430	.525	.645
618	260.0	.145	.035	.060	.130	.185	.235	.290	.345	.425	.525	.645
619	270.0	.145	.035	.060	.130	.180	.235	.290	.345	.425	.525	.640
620	280.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
621	290.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
622	300.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
623	310.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
624	320.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
625	330.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
626	340.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
627	350.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
628	360.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
629	380.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
630	400.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
631	420.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
632	440.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
633	460.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
634	480.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
635	500.0	.150	.040	.030	.125	.180	.230	.285	.340	.420	.525	.640
636	525.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
637	550.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
638	575.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
639	600.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
640	625.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
641	650.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
642	675.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
643	700.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
644	725.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640
645	750.0	.150	.040	.060	.125	.180	.230	.285	.340	.420	.525	.640

646	10.0	.120	.025	.075	.140	.185	.255	.300	.0	.415	.470	.490	.545	.66	.74	.84
647	11.0	.124	.020	.071	.135	.180	.250	.295	.0	.410	.465	.485	.540	.65	.73	.83
648	12.0	.128	.015	.067	.130	.175	.245	.290	.0	.405	.460	.480	.535	.65	.73	.83
649	13.0	.132	.010	.063	.125	.170	.240	.285	.0	.400	.455	.475	.530	.64	.72	.82
650	14.0	.136	.005	.059	.120	.165	.235	.280	.0	.395	.450	.470	.525	.64	.72	.81
651	15.0	.140	.000	.055	.115	.160	.230	.275	.0	.390	.445	.465	.520	.63	.71	.80
652	16.0	.143	.004	.051	.111	.156	.226	.272	.0	.386	.441	.462	.515	.63	.71	.80
653	17.0	.146	.008	.047	.107	.152	.222	.269	.0	.382	.437	.459	.510	.62	.70	.79
654	18.0	.149	.012	.043	.103	.148	.218	.266	.0	.378	.433	.456	.505	.61	.70	.78
655	19.0	.152	.016	.039	.099	.144	.214	.263	.0	.374	.429	.453	.500	.60	.69	.78
656	20.0	.155	.020	.035	.095	.140	.210	.260	.0	.370	.425	.450	.495	.59	.68	.77
657	21.0	.158	.023	.032	.091	.134	.206	.256	.0	.367	.420	.446	.492	.59	.68	.77
658	22.0	.161	.026	.029	.087	.128	.202	.252	.0	.364	.415	.442	.489	.58	.68	.76
659	23.0	.164	.029	.026	.083	.122	.198	.248	.0	.361	.410	.438	.486	.58	.68	.76
660	24.0	.167	.032	.023	.079	.116	.194	.244	.0	.358	.405	.434	.483	.58	.67	.75
661	25.0	.170	.035	.020	.075	.110	.190	.240	.0	.355	.400	.430	.480	.57	.67	.75
662	26.0	.173	.037	.018	.072	.108	.188	.238	.0	.353	.397	.428	.477	.57	.67	.75
663	27.0	.176	.039	.016	.069	.106	.186	.236	.0	.351	.394	.426	.474	.57	.67	.75
664	28.0	.178	.041	.014	.066	.104	.184	.234	.0	.349	.391	.424	.471	.57	.67	.75
665	29.0	.181	.043	.012	.063	.102	.182	.232	.0	.347	.388	.422	.468	.56	.66	.74
666	30.0	.183	.045	.010	.060	.100	.180	.230	.0	.345	.385	.420	.465	.56	.66	.74
667	31.0	.185	.048	.008	.058	.098	.178	.228	.0	.343	.382	.418	.463	.56	.66	.74
668	32.0	.187	.051	.006	.056	.096	.176	.226	.0	.341	.379	.416	.461	.56	.66	.74
669	33.0	.189	.053	.004	.054	.094	.174	.224	.0	.339	.376	.414	.459	.56	.66	.74
670	34.0	.191	.055	.002	.052	.092	.172	.222	.0	.337	.373	.412	.457	.55	.65	.73
671	35.0	.193	.057	.000	.050	.090	.170	.220	.0	.335	.370	.410	.455	.55	.65	.73
672	36.0	.195	.060	.002	.048	.088	.168	.218	.0	.333	.367	.408	.453	.55	.65	.73
673	37.0	.197	.063	.004	.046	.086	.166	.216	.0	.331	.364	.406	.451	.55	.65	.73
674	38.0	.199	.066	.006	.044	.084	.164	.214	.0	.329	.361	.404	.449	.54	.64	.72
675	39.0	.201	.068	.008	.042	.082	.162	.212	.0	.327	.358	.402	.447	.54	.64	.72
676	40.0	.203	.070	.010	.040	.080	.160	.210	.0	.325	.355	.400	.445	.54	.64	.72
677	41.0	.206	.072	.012	.038	.078	.158	.208	.0	.323	.354	.398	.443	.54	.64	.72
678	42.0	.209	.074	.014	.036	.076	.156	.206	.0	.321	.353	.396	.441	.54	.64	.72
679	43.0	.211	.076	.016	.034	.074	.154	.204	.0	.319	.352	.394	.439	.53	.63	.71
680	44.0	.213	.078	.018	.032	.072	.152	.202	.0	.317	.351	.392	.437	.53	.63	.71
681	45.0	.215	.080	.020	.030	.070	.150	.200	.0	.315	.350	.390	.435	.53	.63	.71
682	46.0	.217	.082	.022	.029	.069	.148	.198	.0	.314	.348	.389	.433	.53	.63	.71
683	47.0	.219	.084	.024	.028	.068	.146	.196	.0	.313	.346	.388	.431	.53	.63	.71
684	48.0	.221	.085	.025	.027	.067	.144	.194	.0	.312	.344	.387	.429	.52	.62	.70
685	49.0	.223	.086	.026	.026	.066	.142	.192	.0	.311	.342	.386	.427	.52	.62	.70
686	50.0	.225	.087	.027	.025	.065	.140	.190	.0	.310	.340	.385	.425	.52	.62	.70
687	51.0	.227	.089	.029	.023	.064	.138	.188	.0	.308	.339	.383	.423	.52	.62	.70
688	52.0	.229	.091	.031	.021	.063	.136	.186	.0	.306	.338	.381	.421	.52	.62	.70
689	53.0	.230	.092	.033	.019	.062	.134	.184	.0	.304	.337	.379	.419	.51	.61	.70
690	54.0	.231	.093	.035	.017	.061	.132	.182	.0	.302	.336	.377	.417	.51	.61	.70
691	55.0	.232	.095	.037	.015	.060	.130	.180	.0	.300	.335	.375	.415	.51	.61	.70
692	56.0	.234	.097	.039	.013	.058	.129	.179	.0	.299	.333	.373	.414	.51	.61	.70
693	57.0	.236	.099	.041	.011	.056	.128	.178	.0	.298	.331	.371	.413	.51	.61	.70
694	58.0	.237	.100	.042	.009	.054	.127	.177	.0	.297	.329	.369	.412	.51	.61	.70
695	59.0	.239	.102	.044	.007	.052	.126	.176	.0	.296	.327	.367	.411	.50	.61	.69
696	60.0	.240	.103	.045	.005	.050	.125	.175	.0	.295	.325	.365	.410	.50	.61	.69
697	61.0	.242	.105	.046	.005	.049	.124	.174	.0	.294	.323	.364	.408	.50	.61	.69
698	62.0	.244	.107	.047	.004	.048	.123	.173	.0	.293	.321	.363	.406	.50	.60	.69
699	63.0	.245	.108	.048	.004	.047	.122	.172	.0	.292	.319	.362	.404	.50	.60	.68
700	64.0	.246	.109	.049	.003	.046	.121	.171	.0	.291	.317	.361	.402	.49	.60	.68
701	65.0	.247	.110	.050	.003	.045	.120	.170	.0	.290	.315	.360	.400	.49	.60	.68
702	66.0	.249	.111	.052	.001	.044	.118	.168	.0	.288	.314	.358	.399	.49	.60	.68
703	67.0	.251	.112	.053	.001	.043	.116	.166	.0	.286	.313	.356	.398	.49	.60	.68
704	68.0	.252	.113	.054	.003	.042	.114	.164	.0	.284	.312	.354	.397	.49	.60	.67

69.0	.253	.114	.055	.006	.041	.112	.162	.0	.282	.311	.352	.396	.40	.60	.67
70.0	.253	.115	.056	.009	.040	.110	.160	.0	.280	.310	.350	.395	.40	.60	.67
71.0	.255	.117	.058	.011	.035	.109	.159	.0	.278	.309	.348	.393	.49	.60	.67
72.0	.256	.118	.060	.012	.036	.108	.158	.0	.276	.308	.346	.391	.40	.60	.67
73.0	.257	.119	.061	.013	.034	.107	.157	.0	.274	.307	.344	.389	.40	.60	.67
74.0	.258	.121	.062	.014	.032	.106	.156	.0	.272	.306	.342	.387	.40	.50	.67
75.0	.260	.123	.063	.015	.030	.105	.155	.0	.270	.305	.340	.385	.40	.50	.67
80.0	.268	.130	.070	.023	.025	.100	.150	.0	.265	.300	.335	.380	.47	.58	.66
85.0	.273	.140	.075	.030	.020	.095	.140	.0	.260	.295	.330	.375	.47	.58	.65
90.0	.280	.144	.083	.035	.010	.090	.135	.0	.255	.290	.325	.365	.46	.57	.64
95.0	.285	.147	.090	.040	.000	.080	.130	.0	.250	.285	.320	.360	.46	.57	.64
100.0	.290	.154	.095	.050	.000	.075	.125	.0	.250	.280	.315	.355	.45	.56	.63
110.0	.300	.165	.105	.060	.010	.065	.115	.0	.235	.270	.305	.345	.44	.55	.62
120.0	.310	.175	.115	.070	.020	.055	.105	.0	.225	.260	.295	.340	.43	.54	.61
130.0	.320	.185	.122	.075	.035	.045	.095	.0	.220	.250	.290	.330	.42	.53	.61
140.0	.328	.192	.131	.085	.040	.035	.085	.0	.210	.240	.280	.320	.42	.53	.60
150.0	.335	.200	.140	.095	.050	.030	.080	.0	.205	.235	.275	.315	.41	.52	.59
160.0	.342	.206	.147	.100	.060	.020	.075	.0	.200	.230	.270	.305	.40	.51	.58
170.0	.350	.214	.154	.110	.065	.015	.070	.0	.190	.220	.260	.300	.40	.50	.58
180.0	.355	.219	.160	.115	.070	.010	.065	.0	.190	.220	.255	.295	.39	.50	.57
190.0	.360	.225	.165	.120	.075	.000	.060	.0	.180	.210	.250	.290	.39	.50	.56
200.0	.365	.230	.170	.125	.080	.010	.055	.0	.180	.210	.250	.285	.38	.49	.56
210.0	.370	.235	.175	.130	.080	.010	.050	.0	.175	.205	.240	.280	.38	.49	.56
220.0	.374	.239	.180	.135	.085	.015	.045	.0	.170	.200	.240	.275	.38	.48	.55
230.0	.377	.243	.184	.140	.090	.015	.040	.0	.165	.200	.235	.275	.37	.48	.55
240.0	.380	.245	.186	.140	.090	.020	.035	.0	.160	.195	.230	.270	.37	.47	.55
250.0	.384	.246	.190	.140	.095	.020	.035	.0	.160	.195	.230	.270	.37	.47	.54
260.0	.385	.249	.192	.145	.095	.025	.030	.0	.155	.190	.225	.265	.36	.47	.54
270.0	.388	.250	.194	.145	.100	.025	.030	.0	.150	.190	.220	.260	.36	.47	.54
280.0	.390	.250	.195	.150	.100	.030	.025	.0	.150	.185	.220	.260	.36	.46	.53
290.0	.391	.250	.195	.150	.100	.030	.025	.0	.150	.183	.215	.255	.36	.46	.53
300.0	.393	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
310.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
320.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
330.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
340.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
350.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
360.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
380.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
400.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
420.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
440.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
460.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
480.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
500.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
525.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
550.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
575.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
600.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
625.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
650.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
675.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
700.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
725.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53
750.0	.395	.250	.195	.150	.100	.030	.020	.0	.150	.180	.210	.250	.36	.46	.53

****

UNIEND

APPENDIX C

LABORATORY SCALE SLUG TESTING

APPENDIX C

LABORATORY SCALE 'SLUG' TESTING

C.1 Introduction

'Slug' testing can be used to determine aquifer parameters by either the removal or injection of a water 'slug' in a test well or borehole, Bouwer (6). The time taken for the water level in the borehole to return to its original rest level, is then monitored.

C.2 Method of Analysis

The equation for residual head in an instantaneous vertical line sink can be written, equation C1,

$$S_h = \frac{Qe^{-\frac{r^2S}{4Tt}}}{4\pi Tt} \qquad \text{..........} \quad C1$$

where S_h = Residual Head after injection of a water slug.

r = Distance from the test well to an observation well.

t = Time since slug was injected measured from the average of the times marking the start and finish of the injection.

S = Storage Coefficient

T = Transmissivity

Q = Volume of Slug.

Equation C1 is quoted from a report by Deeney (112), but was originally derived by Theis (34).

If a small volume of water is added to a test well, the aquifer reaction to the injected slug is not normally measurable beyond the immediate well vicinity. Water level measurements are therefore only made in the injection well: the distance, r, then becomes the radius of the well, r_w. Since both r and r_w are small, and S is also small, the exponent of e in equation C1 approaches zero as t becomes large and the value of the exponential term approaches unity. If Q is expressed in m^3, T in m^2/day, t in minutes and S_h in metres, equation C1 can be re-written,

$$T = \frac{1440\ Q}{4\pi S_h t} \qquad \text{..........} \quad C2$$

or

$$T = \frac{114.6\ Q\ (1/t)}{S_h} \qquad \text{..........} \quad C3$$

A plot of S_h against 1/t on arithmetic graph paper results in a straight line through the origin. T is calculated from the co-ordinates of any point on the straight line. However, in practice, actual well test data usually defines as exponential curve and a straight line can therefore only usually be drawn through the later data values. Early data is seldom plotted, which means that the scales can be expanded and a more accurate plot produced.

C.3 Apparatus and Test Procedure

The test apparatus is shown in Figure C.1. It consists of a core sample 60 mm in diameter and 120 mm long into the centre of which has been drilled a 10 mm diameter hole to a depth of 80 mm.

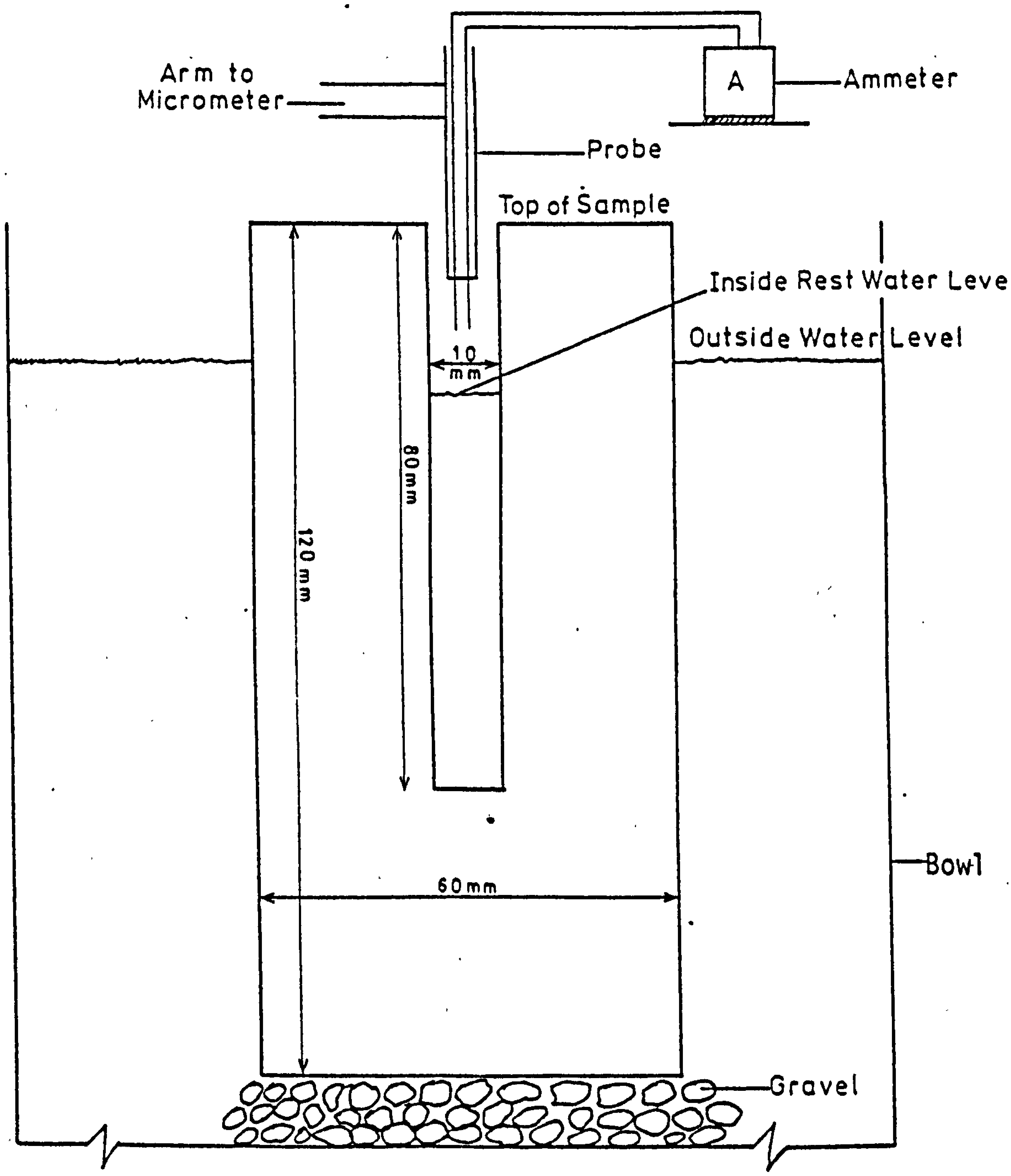


Figure C.1 Apparatus for Determining Permeability by Slug Testing Techniques

Once a sample has been prepared, the drill hole is thoroughly washed with water, although acid can be used. The sample is then immersed in a large bowl (plastic washing-up bowl) which is filled with water. A gravel base provides a complete hydraulic connection.

The sample is left for several days until water in the drill hole reaches a rest level. Once equilibrium is established, the hole can then be emptied using a pipette and the volume of water measured. Readings are then taken at regular intervals, to monitor the rise in water back to its original level. The resulting data is analysed using equation C3.

The water level is measured using two wire electrodes attached to a screw micrometer. The electrodes are lowered from a base reading, until contact is made with the water. An electrical circuit is then completed which registers as a deflection on a sensitive ammeter. Using this type of probe, the electrodes do not create artifically high water levels within the drill hole, due to capillary or surface tension effects.

C.4 Applicability of the Technique

When well test data is analysed, certain assumptions are made concerning the conditions which exist within the aquifer medium and test well. These can be listed as seven main points, Kruseman and de Ridder (32):

1) the aquifer is apparently of infinite areal extent.
2) the aquifer is homogeneous, isotropic and of uniform thickness.
3) prior to pumping, the piezometric surface and/or phreatic surface are (nearly) horizontal.

4) the discharge rate is constant.

5) the aquifer is fully penetrated.

In addition, for unsteady methods only,

6) storage in the well can be neglected.

7) water removed from storage is discharged instantaneously with decline in head.

Although aquifers of infinite lateral extent do not exist, many are so wide that for all practical purposes they can be considered to be. The assumption that aquifers are isotropic and homogeneous is probably never met in practice. All aquifers contain lithological variations which effect the permeability. Similarly, the necessity for aquifers to be of constant thickness is not necessary, since the development of a cone of depression will seldom vary much with aquifer thickness. Therefore, in many actual situations, no serious errors result if not all the assumptions are fulfilled.

The laboratory testing procedure is considered to fulfil many of the theoretical criteria mentioned by Kruseman and de Ridder (32).

C.4.1 Advantages of the Technique

1) If a test sample is chosen carefully, its small size will correspond more readily to an isotropic and homogeneous medium.

2) The bowl of water in which the sample rests can be assumed to be of infinite areal extent, since the volume of water removed during testing is very small when compared with the total volume in the system.

3) The drill hole diameter is small compared to the sample diameter.

4) A cone of depression develops within the sample which is equivalent to that developed around large scale test wells.

5) Radial flow and hence Darcian conditions should be developed around the drill hole.

6) The laboratory technique resembles more closely the theoretical assumptions for deriving aquifer parameters, than the actual field techniques.

C.4.2 Disadvantages of the Technique

1) The technique involves working on a very small scale, with water volumes of several cm^3 and a rate of rise in the drill hole of mm/hour.

2) Technical difficulties are posed by measuring the very small changes in water level which occur in the narrow diameter drill hole. The probe must be sufficiently small so as not to create artifically high water levels due to capillary effects.

3) The test assumes the sample and water to constitute a single isotropic, homogeneous aquifer medium, whereas in reality a boundary condition may exist.

4) The test hole and sample should be thoroughly clean, otherwise anomalous results might occur due to clogged pores.

5) Water level in the bowl should remain constant, although significant evaporation losses can occur unless suitable precautionary measures are taken.

6) Composition of the water, can result in permeability changes due to chemical, physical or bacterialogical action on the sample. Ideally, distilled water should be used during experiments although this seldom resembles the water encountered under field conditions.

C.5 Analysis of Test Results

Tables C1 and C2 show test data collected from two samples of Darley Dale sandstone. Sample A is a normal specimen, while Sample F has been subjected to intact failure under uniaxial compression. Table C3 shows recovery levels for the two samples, A and F, and Figure C.2 a graph of Residual Head (S_h) against the Reciprocal of Time (1/t) for both specimens.

The experimental work was undertaken by the author, although the data was only analysed after consultation with Black (113).

Values taken from Figure C.2 for substitution in equation C3 are:

	S_h	$1/t$
A_1	1.41×10^{-2}	6.75×10^{-4}
A_2	0.92×10^{-2}	5.50×10^{-4}
F_1	2.16×10^{-2}	6.75×10^{-4}
F_2	1.70×10^{-2}	5.25×10^{-4}

where the volume of water removed is

Specimen A = $4.56\ cm^3 = 4.56 \times 10^{-6}\ m^3$

Specimen F = $4.10\ cm^3 = 4.10 \times 10^{-6}\ m^3$

Sample A

	22/6 (9.36 am)	24/6 (10.03 am)	26/6 (11.50 am)
Top of Sample	81.79	81.91	82.0
Outside Water Level	55.62	56.13	55.88
Inside Rest Water Level	56.26	-	-

Actual Volume of Water Removed 4.56 cm^3

Theoretical Volume of Water Removed - 4.59 cm^3

Time since start of Recovery (mins)	Micrometer Reading (mm)	Amount of Recovery (mm)
0	23.24	0.00
30	24.23	0.99
59	25.96	2.71
94	27.68	4.44
121	28.72	5.48
150	29.87	6.63
191	31.60	8.35
210	32.39	9.14
246	33.78	10.54
286	35.41	12.61
344	37.39	14.14
405	39.57	16.33
1505	65.66	42.42
1534	66.37	43.13
1569	66.98	43.74
1603	67.43	44.20
1650	68.20	44.96
1677	68.60	45.36
1709	68.98	45.74
1737	69.29	46.05
1776	69.64	46.40
1801	70.00	46.76
1837	70.51	47.27
1876	70.56	47.32
2907	77.14	53.90
2998	77.39	54.15
3147	77.90	54.66
3442	78.36	55.11
4265	79.63	56.39
5894	80.69	57.45

Table C.1 Test Data - Sample A

Sample F

	22/6 (9.21 am)	24/6 (9.55 am)	26/6 (11.51 am)
Top of Sample	84.07	84.33	84.33
Outside Water Level	56.21	55.88	56.08
Inside Rest Water Level	59.46	-	-

Actual Volume of Water Removed - 4.10 cm^3

Theoretical Volume of Water Removed - 4.15 cm^3

Time since start of Recovery (mins)	Micrometer Reading (mm)	Amount of Recovery (mm)
0	6.63	0.00
24	6.73	0.10
39	7.19	0.56
56	7.56	0.94
69	8.13	1.50
113	9.73	3.10
132	10.62	3.98
169	11.86	5.23
204	13.10	6.47
230	14.35	7.72
259	15.36	8.73
306	17.02	10.39
353	18.67	12.04
439	20.95	14.32
1514	44.78	38.15
1552	45.31	38.68
1581	45.69	39.06
1621	46.25	39.62
1662	46.71	40.08
1695	46.94	40.31
1721	47.29	40.66
1756	47.50	40.87
1787	47.83	41.20
1819	48.08	41.45
1850	48.41	41.78
1894	48.64	42.01
2914	54.00	47.34
3015	54.05	47.42
3159	54.25	47.62
3461	54.63	48.00
4279	55.50	48.87
5910	56.21	49.58

Table C.2 Test Data - Sample F

Sample A (Normal)			Sample F (Stressed)		
t	1/t	S_h	t	1/t	S_h
(mins)	$\times 10^{-4}$	(mm)	(mins)	$\times 10^{-4}$	(mm)
0	-	56.26	0	-	59.46
30	333.0	55.27	24	417.0	59.36
59	169.0	53.54	39	256.0	58.90
94	106.0	51.82	56	179.0	58.52
121	82.6	50.77	69	145.0	57.96
150	66.7	49.63	113	88.5	56.36
191	52.3	47.90	132	75.8	55.47
210	47.6	47.11	169	59.2	54.23
246	40.7	45.72	204	49.0	52.98
286	35.0	44.09	230	43.5	51.74
344	29.1	42.11	259	38.6	50.72
405	24.7	39.93	306	32.7	49.07
1505	6.64	13.84	353	28.3	47.42
1534	6.52	13.13	439	22.8	45.13
1569	6.37	12.52	1514	6.61	21.31
1603	6.24	12.06	1552	6.44	20.77
1650	6.06	11.30	1581	6.33	20.39
1677	5.96	10.89	1621	6.17	19.83
1709	5.85	10.51	1662	6.02	19.38
1737	5.76	10.21	1695	5.90	19.15
1776	5.63	9.85	1721	5.81	18.80
1801	5.55	9.50	1756	5.69	18.59
1837	5.44	9.00	1787	5.60	18.26
1876	5.33	8.94	1819	5.50	18.00
2907	3.44	2.36	1850	5.41	17.67
2998	3.33	2.11	1894	5.28	17.45
3147	3.18	1.60	2914	3.43	12.11
3442	2.91	1.14	3015	3.32	12.04
4265	2.34	-0.13	3159	3.17	11.83
5894	1.70	-1.19	3461	2.89	11.45
-	-	-	4279	2.34	10.59
-	-	-	5910	1.69	9.88

Table C.3 Time and Recovery Levels within the Normal and Stressed Samples

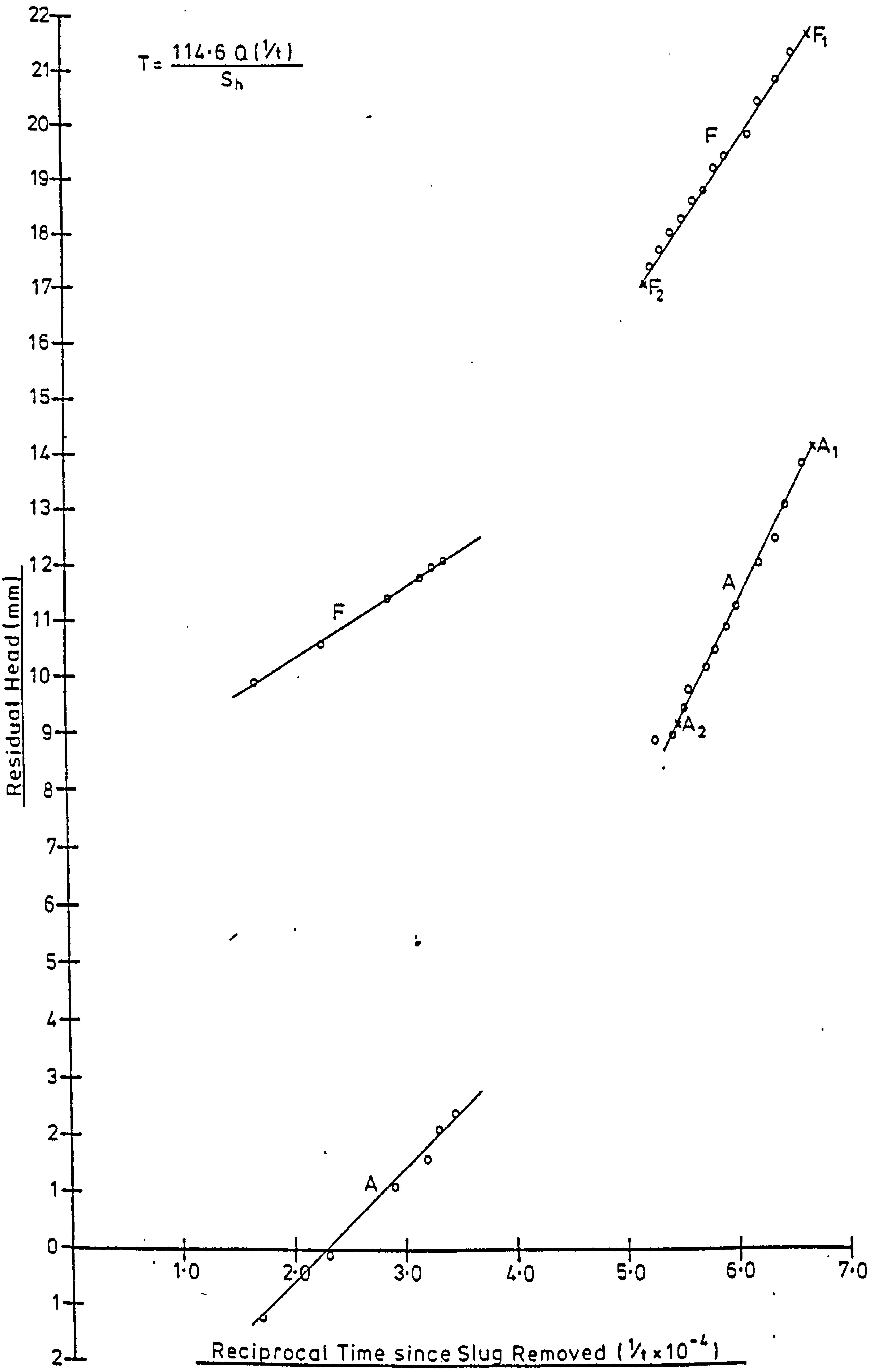


Figure C.2 Residual Head (S_h) against the Reciprocal of Time (1/t) for Samples A and F

Therefore using $T = \frac{114.6\ Q\ (1/t)}{S_h}$ C3

	Transmissivity (T)	Permeability (k)
A_1	2.50×10^{-5}	3.12×10^{-4}
A_2	3.12×10^{-5}	3.90×10^{-4}
F_1	1.47×10^{-5}	1.84×10^{-4}
F_2	1.45×10^{-5}	1.81×10^{-4}

where k is calculated using equation 2.11, Chapter 2 or Bouwer (6),

$$T = kD \qquad \text{.......... 2.11}$$

where T - Transmissivity

k - Permeability

D - Aquifer Thickness

D in this case is taken as the penetration depth of the drill hole - 8.0×10^{-2} m.

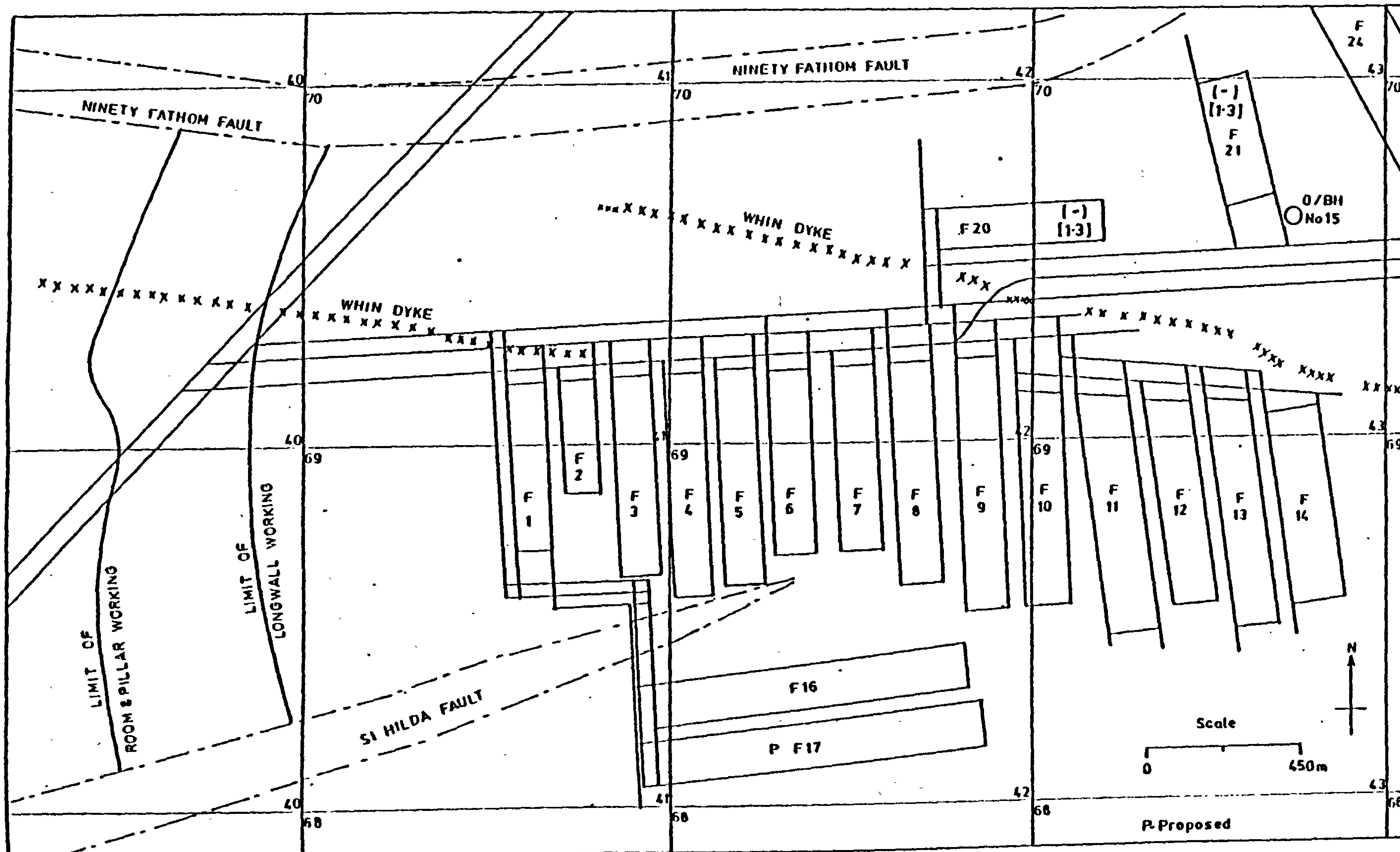
The results show that a decrease in permeability has occurred within the stressed sample. This can be associated with crushing of the pore spaces and a reduction in intergranular permeability. A significant change from micro to macro fissuring within the stressed sample does not appear to have occurred and is probably due to the intact nature of the specimen after failure.

C.6 Conclusion

The technique offers a cheap, simple and effective method for determining the permeability of standard core samples, while keeping the amount of sample preparation to a minimum. Using standard uniaxial testing procedures, it is possible to determine whether a relationship exists between permeability and the induced strain on a specimen. Similarly, with the advent of Stiff Testing Machines it should be possible to monitor the post-failure permeability characteristics of a sample.

Using a series of control samples, it should be possible to vary the drill hole characteristics in order to simulate different well types and aquifer conditions. A hole drilled completely through the sample could be sealed at one end using resin or glue and used to simulate a fully penetrating well system. Similarly, the effect of variations in hydrochemistry and bacterial growth on or within a specimen could also be assessed. If large scale triaxial cells were available, the technique could also be extended to monitor the permeability changes associated with loading under triaxial conditions.

Finally, the technique offers almost unlimited potential for the testing of most rock types under a variety of loading conditions. However, rock types which disintegrate when in contact with water, such as mudstones, could not be used. The method could therefore prove useful for solving many of the hydrogeological problems encountered in both coal and metalliferous mining. Locally, it could have been used to help solve a variety of hydrogeological problems which have been encountered in the underground ironstone workings of Lincolnshire, Aston and Smith (114).



( ) MAXIMUM WATER YIELD m³/min [ ] RESIDUAL WATER YIELD m³/min

Figure 8.13 Central and Southern Areas, Main (F1) Seam, Westoe Colliery