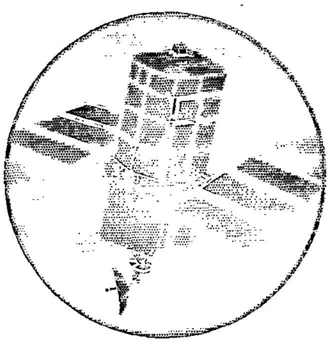
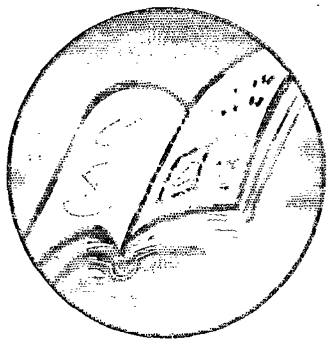
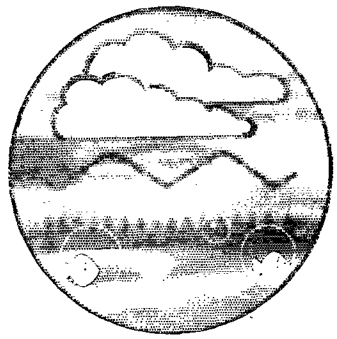
Predicting Mine Water Rebound

Research and Development
Technical Report
W179

NORTHUMBRIAN WATER

ENVIRONMENT AGENCY

Predicting Mine Water Rebound

R&D Technical Report W179

P L Younger and R Adams

Research Contractor:
University of Newcastle

**Further copies of this report are available from:
Environment Agency R&D Dissemination Centre, c/o
WRc, Frankland Road, Swindon, Wilts SN5 8YF**

tel: 01793-865000 fax: 01793-514562 e-mail: publications@wrcplc.co.uk

Publishing Organisation:
Environment Agency
Rio House
Waterside Drive
Aztec West
Almondsbury
Bristol BS32 4UD

Tel: 01454 624400 Fax: 01454 624409

ISBN:1 85705 050 9

This project was jointly funded by the Environment Agency and Northumbrian Water.

Dissemination status
Internal: Released to Regions
External: Released to the Public Domain

Statement of use
This report is a summary of the current "state of the art" on minewater rebound, which will provide hydrogeologists involved in the subject with much practical guidance.

Research contractor
This document was produced under R&D Project W6-614 by:

Department of Civil Engineering
Cassie Building
University of Newcastle
Newcastle upon Tyne
NE1 7RU

Tel: 0191 222 6000 Fax: 0191 222 6502

Environment Agency Project Leader
The Environment Agency's Project Leader for R&D Project W6-614 was:
Mr J Aldrick, Environment Agency, North East Region

FOREWORD

This report is one of the main outcomes of a three-year national R&D project co-funded by the Environment Agency, Northumbrian Water Ltd and the University of Newcastle Upon Tyne, under the title "Improved Modelling of Abandoned Coalfields". The Environment Agency National R&D Project Number for the project was 614. The project ran from January 1996 to January 1999, and was managed by a Project Steering Committee with the following membership:

Dr Alan Lowdon – Project Manager, Northumbrian Water Ltd
Dr R John Aldrick – Project Manager, Environment Agency North East Region

Sophie Barraclough, Environment Agency North East Region (who acted as Environment Agency Project Manager 1996-97)
Wayne Davies, Environment Agency, Welsh Region
Steve Dumpleton, British Geological Survey
Mike Eggboro, Environment Agency North West Region
Paul Hart, Environment Agency, Anglian Region
Bob Harris, Environment Agency National Groundwater Centre
Martin Kershaw, Environment Agency North East Region
Cliff Tubb, Environment Agency South West Region

The authors would like to take this opportunity to thank all of the Steering Committee members for their unstinting support over the years, without which the research would have been very much poorer.

The research reported here was undertaken primarily by Dr Paul L Younger and Mr Russell Adams of the University of Newcastle Upon Tyne, who are also the co-authors of this report. The University also allocated one of its EPSRC quota PhD studentships to support the project, and this was held by Dr Julia M Sherwood (now with the Environment Agency, South West Region), who undertook most of the work associated with the GRAM modelling approach described in Chapter 4.

The research drawn upon in writing this report took cognizance of previous work on related topics, as listed in the Reference list at the end of this report, and in the more comprehensive bibliography given in the Project Record. Although primarily conceived to address coal mine water problems, this project also benefited from insights obtained in Environment Agency projects concerning metals mines, particularly at Wheal Jane (Cornwall), South Crofty (Cornwall), Frazer's Grove (Durham) and the Nent Valley (Cumbria), in all of which Dr Younger formally participated with other Environment Agency funding. Some data used to test the water quality prediction protocol outlined in Chapter 6 was provided by the Scottish Environment Protection Agency (SEPA), and the assistance of Dr David Holloway is particularly acknowledged. Applications of GRAM to the Fife Coalfield (funded by one of SEPA's predecessor organisations) proved most useful in developing the model. Finally, the authors wish to acknowledge the debt to the following colleagues who have contributed so much to their appreciation of mine water problems: Dr Steven Banwart (Sheffield), Mr Sean Burke (Sheffield), Dr Bob Hedin (USA), Dr Bob Nairn (USA), Dr Peter Norton (Richmond), Prof Fernando Pendás Fernández (Spain) and Prof Rafael Fernández Rubio (Spain).

CONTENTS

Page No.

List of Figures

Page No.

List of Tables

Page No.

Glossary

Adit- an approximately horizontal mine tunnel (typically with a floor gradient of at least 1:500 towards the portal) which permits access from the surface into underground workings in hilly country. Most adits also perform(ed) a drainage function. Synonyms: **drift, level, horse level** (if used for haulage), **water drift, water level**).

Aerobic - condition in which where there is no shortage of atmospheric oxygen.

Anaerobic- condition in which there is a complete absence of atmospheric oxygen.

Aquifer- a body of rock which stores and transmits significant quantities of water.

Asymptotic - the condition under which a previously increasing or decreasing variable conforms to a more-or-less constant value.

Ferruginous- bearing large quantities of iron.

Longwall - a method mining whereby large rectangular areas of coal (or other mineral), known as "panels", are worked by means of repeated excavation (generally with drum shearers on tracks) of one of the shorter sides of the rectangle (the so-called longwall). Roadways on either side of the panel give access to the working longwall face from the main haulage roadway. Panels which are worked such that the working face gets ever further away from the main haulage roadway are called "advance longwall", whereas those which start away from the main haulage and work back towards it are "retreat faces". Parts of the panel from which the coal has already been extracted are left without roof support, so that the roof collapses into the worked void, forming a body of rubble known as "**goaf**".

Room & Pillar - a system of mining (mainly used before the 1960s) whereby interconnecting "rooms" (known as "**bords**" in the North East) are excavated on a rectilinear grid, with square or rectangular "pillars" of intact coal (or other mineral) left in place to support the roof. (Synonyms: **bord-and-pillar** (NE Coalfield); **stoop-and-room** (Scotland); **pillar-and-stall** (Yorkshire and E Midlands coalfields)).

Slaking- the process by which a previously solid rock (typically a shale) turns into a slurry upon wetting.

Stope- a vertical or sub-vertical mine void from which mineral (typically in the form of a vein) has been (or is being) extracted. If the extraction proceeds upwards from the main haulage roadway (the most common case) it is termed "**overhand stoping**"; if it proceeds downwards from the main haulage it is termed "**underhand stoping**".

Stythe- oxygen deficient air, often encountered in abandoned or poorly-ventilated coal mine workings, and sometimes in other types of mine. Synonym: **blackdamp**.

Executive Summary

"Mine water rebound" is the process whereby abandoned deep mine workings flood with water after the cessation of dewatering, often leading to surface discharges of polluted mine waters and/or the flow of polluted mine waters into adjoining aquifers. The environmental impacts of abandoned mine discharges can be severe. Consequently a rational policy for managing mine abandonment is desirable. Central to developing management strategies is an ability to assess whether a particular mine closure will lead to problems after rebound is complete. If so, development of cost-beneficial remedial responses demands an estimate of the time-scales over which problems may develop, in other words, the time-scale over which rebound will occur.

Methods for predicting the time needed for rebound in a particular case vary from simple manual calculations to fully 3-D physically-based modelling. Selection and correct application of the appropriate technique demands an appreciation of the "non-standard" hydrogeology of deep-mined systems. The flow of water through abandoned mine systems is strongly controlled by the presence and layout of mine workings. Room-and-pillar workings, longwall workings and sub-vertical stopes all behave in distinctive manners. Furthermore, adjustments of surrounding strata to the removal of rock in mine voids leads to changes in the permeability and storage properties beyond the mine boundaries.

The simplest methods of analysis are manual "rate of fill" calculations. Depending on the available data, these may be succeeded by construction of frequency curves of void space versus depth, and by the fitting of exponential rebound curves to early recovery data. Where large systems are analysed, it is often possible to identify distinct "ponds" in deep mine workings, which are extensively interconnected internally, but which only meet other ponds at distinct overflow points. In these circumstances, a well-tested semi-distributed computer code (GRAM – Groundwater Rebound in Abandoned Mineworkings) may be used to predict rebound, with explicit analysis of uncertainty by means of Monte Carlo simulations. At the highest available level of analysis, a newly-developed code (VSS-NET), which is part of the SHETRAN family of software, can be used to model variably saturated flow through mined systems and surrounding strata, including the modelling of turbulent flow in large voids such as roadways.

The prediction of the water quality of future discharges is feasible, at least in terms of total iron concentration, by reference to the sulphur content of the worked seams (or, as a proxy, their stratigraphic proximity to marine beds in the sequence) and the distance of future discharge points from the outcrop of the shallowest worked seam. It is possible to predict peak iron concentrations post-rebound by reference to sulphur content alone. Long-term iron concentrations are best predicted on the basis of a combination of proximity to outcrop and sulphur content. Although the predicted concentrations fall only in broad bands, they do at least allow distinction between severely polluted discharges (with iron in the range 50 – 150 mg/l) and discharges with intermediate (4 – 20 mg/l iron) or low (< 4 mg/l iron) levels of pollution.

Key words: acidity, coal, discharge, environment, groundwater, hydrogeology, iron, mine, minewater, modelling, pollution, prediction, rebound, sulphur, water, water quality.

1. THE CURRENT AND FUTURE STATUS OF MINE WATER POLLUTION PROBLEMS IN BRITAIN

1.1. Britain's Mining History

Britain has one of the longest mining histories in the world. Underground flint mines dating from the Stone Age remain accessible at Grimes Graves in Norfolk (Temple, 1972), and Bronze Age mines for tin and copper are known in Cornwall (Buckley, 1992) and Wales respectively. Indeed, Cornwall was apparently known to the ancient Greeks as "the Cassiterides" on account of the abundance of its tin ('cassiterite' being the name of the most common tin mineral to this day). The Romans certainly worked gold in Wales, and also smelted argentiferous galena from the Pennines to obtain silver (Raistrick and Jennings, 1965). The environmental impacts of these early mining activities were probably very localised.

In the Middle Ages, however, large-scale mining began to be pursued with vigour, not only in Wales, Cornwall and the Pennines, but also in the main coalfields. Deep-mining of coal (as opposed to the casual grubbing of outcrops) is recorded from the 12th Century, when it was often organised by monasteries. Environmental impacts began to become more widespread thereafter. Indeed, old coal workings southwest of Dalkeith in Scotland, which were excavated in the 13th Century under the auspices of Newbattle Abbey, continue to discharge acidic, ferruginous waters into the River Esk.

In the high Pennines, a primitive form of hydraulic mining known as "hushing" was used to explore for, and to work, lead veins. Hushing involved impoundment of upland runoff behind earth embankments, with sudden releases of large volumes of water onto the lower hill-slopes to wash away soil and weathered overburden. The narrow valleys ("hushes") thus created persist as major landscape features, and represent permanent alterations of the natural drainage of many Pennine hills.

Below the same hills, long adits were driven for exploration and drainage. These frequently attained great lengths. For instance, the County Adit in west Cornwall, which was initiated in 1748, had attained a total drivage length of some 64 km by 1880 (Buckley, 1992). Many of these old adits continue to flow at high rates, and represent major, permanent modifications of the hydrology of their host catchments (Younger, 1998a). The County Adit, for instance, transmits around 60% of the effective precipitation falling on the overlying surface catchment, with only 40% forming direct surface runoff (Knight Piésold, 1995). Table 1.1 lists some major examples of such adits in metalliferous mining regions of England. Few adits of this type were constructed in the English coalfields, because of the generally subdued topographic relief in these largely lowland areas. One notable exception to this is the Bridgewater Canal near Wigan, which drains a large area of shallow coal workings by means of several adits. A similarly extensive adit system in Fife, Scotland, drains a large area of old workings to a single point near Fordell Castle, Inverkeithing.

Table 1.1: Average flow (Ml/d) and chemistry data (all "total in mg/l" except for pH and conductivity (μS/cm)) for selected major adits draining abandoned UK mines (after Younger, 1998a)

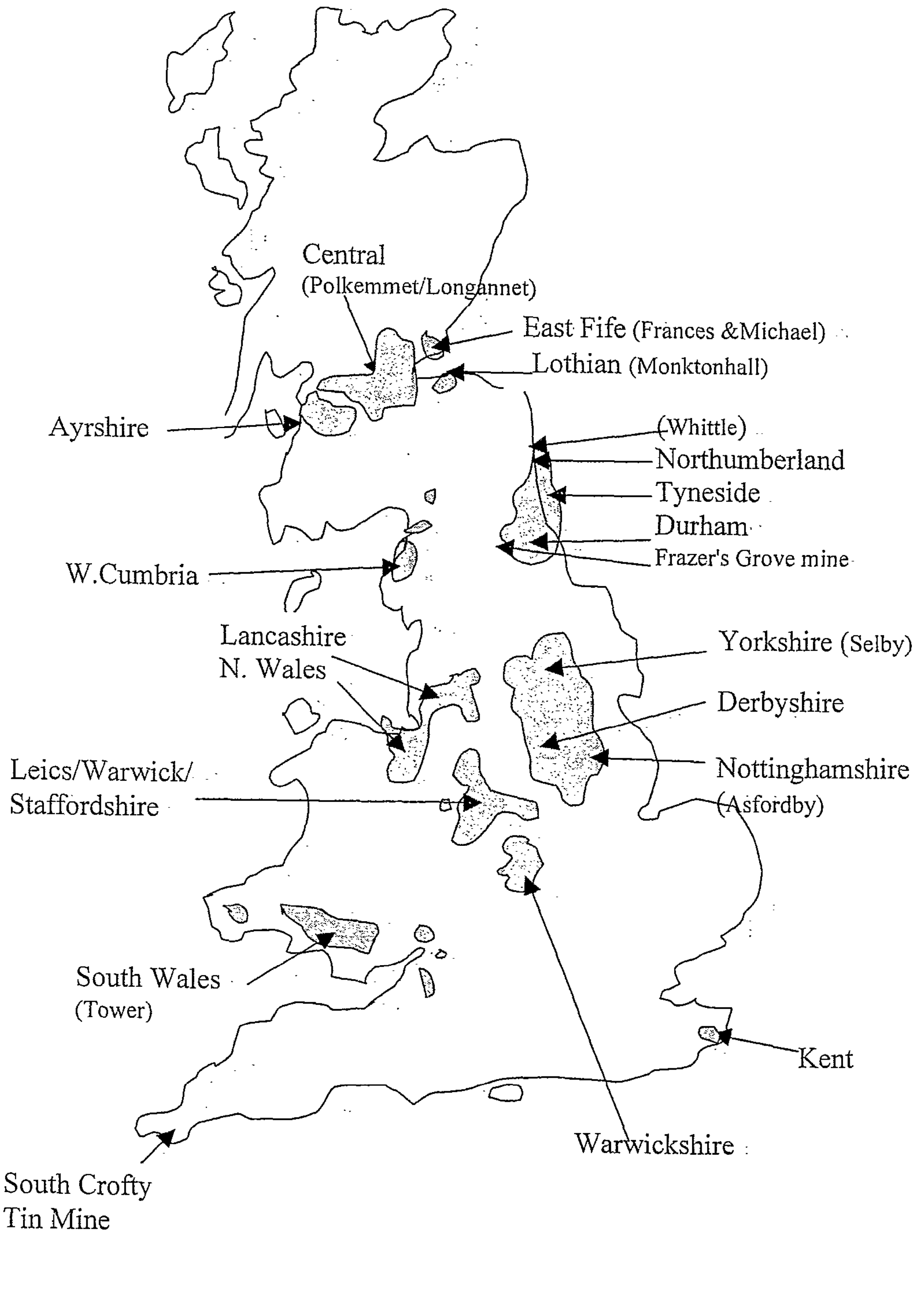
Adit	County	Flow	Cond	pH	Fe	Zn	Cu	SO_4	HCO_3	Cl
County Adit	Cornwall	34	- -	4.0	8	4.5	1.2	- -	0.0	- -
Blackett Level	Northumberland	10	425	7.6	0.03	0.08	0.015	50	200	28
NentForce Level	Cumbria	2	600	7.5	0.12	3.1	0.002	181	220	12
Eagle Level	North Yorkshire	5	400	7.6	0.09	0.09	--	12.7	193	22
Dolcoath Deep Adit	Cornwall	7	- -	7.0	0.5	0.9	0.1	87	- -	48

Discharges of polluted water from old adits has long been a problem in the upland mining districts of England. Most of the USA experience with polluted mine waters relates to similar circumstances. For instance, innumerable old coal mines in the Appalachian hills are predominantly accessed by adits, as are the metals mines of the Rocky Mountains.

It is only in recent decades that abandoned mine water discharges have begun to emerge from the major, regional coalfields of the British lowlands (Henton, 1979, 1981). The major coalfields of Britain are shown on Figure 1.1. Most of these underwent their greatest period of development in the mid-19th Century, and they were (in a few cases, still are) predominantly worked by deep shafts. These have given rise to regionally-interconnected systems of deep workings, usually on several levels corresponding to the different seams. The scale of some of these mine systems is truly exceptional, at least by the standards of the coalfields in the eastern USA which have been the focus of most mining hydrological research to date. Close analogues for the regionally-extensive lowland coalfields of Britain exist in Europe, Australia, and South Africa, though in most of these cases, the coal industry is not so far advanced towards total closure as is the case in the UK. Consequently, little hydrological research of direct applicability to the main UK coalfields has yet been reported in the international literature. This has meant that an indigenous response to problems of large-scale coalfield closure has had to be developed more-or-less "from scratch" in the UK. This report is the first attempt to synthesise the findings of the last 6 years of research aimed at formulating this response. Details of the research are to be found elsewhere (Younger and Adams, 1999); the emphasis here is on providing practical guidance for hydrogeologists concerned with predicting and understanding the consequences of deep mine abandonment[1]. As such, it is further hoped that this report may in due course be found useful in the other countries with deep-mined lowland coalfields.

[1] This report does *not* deal with the particular problems posed by surface mines (opencast), nor with runoff from waste rock piles (spoil heaps).

Figure 1.1. The Major Coalfields of Great Britain, showing also the locations of Whittle Colliery, Frazer's Grove Fluorspar Mine and South Crofty Tin Mine (individual mines named in brackets).



1.2. A New Awareness of Environmental Impacts

The public and regulatory appreciation of the possible consequences of the cessation of dewatering in extensive systems of interconnected mine workings was swiftly elevated onto a new plane in January 1992, ironically by a somewhat atypical case, involving the Wheal Jane tin mine in Cornwall (see Section 1.3.2). The image of a vast orange plume in the Fal estuary is ingrained in the memories of most scientists who followed the contemporaneous press coverage.

These events were still fresh in the memories of many in October 1992, when the President of the Board of Trade, Mr Michael Heseltine, announced the UK government's intention to close more than half of the remaining coal mines in Britain. The proposed closures were not spread evenly around the country. Consequently, for some major mining districts (such as Durham, Lancashire, Leicestershire, South Wales, Midlothian and Fife) this closure programme effectively meant that entire coalfields would be finally abandoned. In all cases, British Coal signalled its intention to promptly cease dewatering all such coalfields as soon as the last mine was closed. A political furore followed the initial closure announcement. In this furore, concerns about possible mine water pollution problems after closure were raised (see Younger, 1993). Eventually the government was forced to make some concessions, which eventually led to the survival of some mines (in Midlothian and South Wales) under new ownership, and to the retention of regional dewatering in the Durham Coalfield on environmental grounds (see Younger, 1993; Younger and Sherwood, 1993; Younger and Harbourne, 1995).

In many other areas, mines were reprieved in 1992, but were eventually closed on "geological grounds" by the mid-1990s (Milne, 1994). In the Spring of 1993, a heavily contaminated discharge began to flow into the Neath Canal at Ynysarwed (South Wales), following flooding of Blaenant Colliery, South Wales. Dewatering had ceased little more than a year earlier (see Section 1.3.2). As a result, it became impossible for further closures to take place without a prior environmental appraisal. These new conditions became formalised in 1995, when the newly-formed Coal Authority and some of the major private mine operators (particularly RJB Mining plc) signed memoranda of understanding with the Environment Agency, committing them to give the Environment Agency six months' notice of a forthcoming abandonment. Subsequently, the Conservative government amended the Water Resources Act 1991 so that mine waters, which might emerge from any mine closed after 31-12-1999 would be subject to the same laws as other polluting discharges. In March 1998, the government published regulations under Section 91B of the Environment Act 1995 requiring the operators of mines to provide the Environment Agency with information on dewatering rates, water quality, mine layout and other factors relevant to mine abandonment assessment, a full six months before abandonment. The Environment Agency is thus far better equipped to evaluate future mine closure plans now than it was at the start of the 1990s. Amongst other things, therefore, this report aims to demonstrate how Environment Agency staff might proceed with the information which must now be made available to them in order to prepare a response to abandonment proposals.

1.3. Status of Mine Water Pollution and Related Problems at the Start of the New Millennium

1.3.1. Possible impacts of mine water rebound

The principal hydrological consequences of ceasing dewatering in large inter-connected mined systems are associated with the process of "mine water rebound", in which formerly dewatered voids gradually fill with water until a surface overflow point is encountered. In descending order of frequency and environmental significance, the principal consequences of this process may be summarised as follows:

A. *Surface water pollution (most common)*, after voids have eventually flooded up to ground level, leading to acidic and/or ferruginous discharges (which may also contain elevated levels of other ecotoxic metals, most notably aluminium and zinc) into previously "clean" streams. The consequences of such pollution can extend to abandonment of public water supply intakes, fish deaths and impoverishment of aquatic flora and fauna, poisoning of land animals which drink the water, and removal of a critical food source for birds and mammals which prey on freshwater fish.

B. *Localised flooding* of agricultural, industrial or residential areas, particularly where structures have been inadvertently constructed over former mine entries.

C. *Temporary loss of dilution* for other pollutants in surface waters as former pumped discharges cease to augment flows.

D. *Over-loading and clogging of sewers.*

E. *Pollution of over-lying aquifers* by upward migration of mine water.

F. *Temporarily accelerated mine gas emissions*, driven ahead of the rising water table. While media attention is usually paid to the risk of flammable methane emissions, methane is such a light gas that it will have normally vented to surface anyway if an upward pathway exists. A more worrying scenario relates to oxygen-deficient air (widely termed "stythe", following north-eastern usage). Stythe is a strong asphyxiant, which acts rapidly and often fatally, particularly in confined spaces. The principal component of stythe is carbon dioxide, which is more dense than air, and will thus lurk above the water table until hydraulically forced upwards. It is worth noting that hydrogeological investigations in and around abandoned mines may be subject to hazards arising from these gases. Methane pockets trapped in antiformal structures in old workings may become highly pressurised by the underlying water, making drilling hazardous. Furthermore, during periods of low atmospheric pressure, stythe emissions from old workings can render fieldwork in the vicinity of old shafts and drifts dangerous.

G. *The risk of subsidence* as rising waters weaken previously dry, open, shallow old workings. The primary mechanism is considered to be slaking of seat-earths underlying pillars of coal left to provide roof support. However, recent (as yet unpublished) research by seismologists of the British Geological Survey suggests that reactivation of fault movement may be responsible for some tremors and differential subsidence in north eastern and north western England.

H. Impingement on landfills, leading to increased rates of leachate and gas emissions.

Over the last three decades, consequences A to G have all been observed in the UK, and some examples of these are outlined below (Section 1.3.2). No examples of consequence H have yet come to light, though some are feared in northern England.

1.3.2. Impacts observed to the end of the 20th Century

A. *Surface water pollution*: The single most famous instance is that of Wheal Jane, a tin mine in Cornwall which was abandoned in 1991. In January 1992, some 50 Ml of acidic, highly metalliferous water unexpectedly burst out of an old adit, some 10 months after dewatering had ceased in voids some 400m below ground. The event has been widely described in print (for a full appreciation, the reader is referred to NRA, 1994; Hamilton *et al*, 1994;; Banks *et al*, 1997; Bowen *et al*, 1998; and Younger, 1998a). Since 1992, the mine water has been pumped and treated by the Environment Agency to minimise any further impacts (Knight Piésold, 1995). Less famous but in many ways more damaging than Wheal Jane (which released its acid waters into an already-dead river) is the discharge from the Ynysarwed coal mine adit in the Neath Valley, South Wales, which began to flow in the Spring of 1993, little more than a year after the closure of the associated deep mine (Younger, 1994; Younger, 1997a; Ranson and Edwards, 1997; Ranson *et al*, 1998). This discharge killed all aquatic fauna over a 12 km reach of the Neath Canal, and deprived BP Chemicals' Port Talbot Works of its source of process water. Numerous similar (if less extreme) discharges occur in Wales, Yorkshire (NRA, 1994), Cleveland (Younger, 1995b), Durham (Younger and Bradley, 1994; Younger, 1995a; Jarvis and Younger, 1997), Northumberland and the Midland Valley of Scotland (Henton, 1981; Robins, 1990; Younger, 1999). The Scottish Environment Protection Agency estimate that some 480 km of streams in Scotland are degraded by abandoned coal mine water (Younger, 1999). Current estimates for England identify some 520 km affected by abandoned mines (NRA, 1994), with a little under half of this total being attributed to abandoned metal mines. Indeed, in the Forth catchment (Scotland), discharges from abandoned coal mines are now the single greatest cause of freshwater pollution. On the other hand, it should be noted that closures of some major mines have yielded surface discharges of relatively good quality, due to the low pyrite content of the ore; the best recent example of this genre is the Geevor Tin Mine in Cornwall, in which dewatering ceased on 6th June 1991. After completion of rebound, a discharge into coastal waters commenced on January 8th 1995, but the water was of generally good quality (excepting a marginal exceedance of Environmental Quality Standards with respect to copper).

B. *Localised flooding* has affected residential, agricultural and industrial land in at least four UK coalfields. For instance, at Allanton, North Lanarkshire, Scotland, mine water rebound following the closure of the Kingshill No 1 Colliery around 1970 culminated in surface discharges commencing around 1985. While the bulk of post-rebound discharge emerged from the Allanton shafts into pre-existing lagoons, a substantial amount of ferruginous water emerged in the yards of nearby houses that had been constructed while the mine was still dewatered. This ferruginous water has caused clogging of local surface water drains, resulting in the occasional backing-up of storm water during wet periods, causing flooding of properties (C Schmolke, personal communication, 1998).

   Around midnight on the 24th of June 1998, water suddenly began to surcharge the main storm sewer in the main street of the sleepy coastal village of Spittal, Northumberland.

The flow of water was violent, flipping heavy covers of access chambers and quickly flooding the street to a depth of around one metre. Of course the water did not restrict itself to the street, but burst also into nineteen homes. Apart from the normal water damage associated with flooding, the unfortunate residents soon discovered that their properties were being stained by the deposition of iron hydroxide. Chemical analysis of a sample obtained in the early hours of 25th June revealed the flood water to have a pH of 5.9, with 1.4 mg/l each of iron and manganese, 920 mg/l of SO_4; and a specific electrical conductance of 1690 µS/cm. The water also carried much silt, and fragments of coal, sandstone and shale. The flow continued as a torrent for about 17 hours, then declined over a period of only 30 minutes to leave a trickle in the bottom of the storm sewer by 17:30 h on 25th June. The total volume of water released in the 17 hour period is estimated to have been at least 60 Ml. Subsequent CCTV inspection of the storm sewers in the town revealed a connection to an old, stone-arched adit portal, buried beneath made ground. A manuscript in local archives records a "coal drift" in this position, which had been abandoned in 1820. All of the evidence suggests that a substantial head of mine water stored behind a barrier of roof-fall debris finally exceeded the strength of the barrier some 178 years after mine closure, delivering a frightening lesson in local history to the unsuspecting residents.

Flooding of agricultural and industrial land is best documented from southwest County Durham, where the cessation of dewatering in the Coal Measures south of the Butterknowle Fault resulted in minewater rebound in the St Helens Auckland area (Younger, 1997c). Eventually, a major uncontrolled discharge emerged from the St Helens Engine Shaft, which had been abandoned during the General Strike in 1926. By 1979, when the discharge commenced, a light engineering workshop had been constructed over the shaft cap, and the outburst of minewater so damaged the building that it had to be demolished. The minewater was then diverted into the nearest culverted water-course, whence it flooded adjoining agricultural land, resulting in the loss of a hay pasture by waterlogging, and in the submergence of an access track. A herd of cattle which were in the pasture at the time developed severe scour, which was fatal for many beasts. The local vet considered the scour to be attributable to the cattle drinking the sulphate-rich minewater. Similar cases are known around the UK, but have not been systematically documented.

C. *Temporary loss of dilution* of sewage effluents was experienced in the vicinity of Worsborough Reservoir, near Barnsley, following the cessation of pumping in the Strafford Colliery Shaft in 1993 (Banks *et al*, 1996).

D. *Over-loading and clogging of sewers.* Perhaps the most significant examples of this genre are associated with the Cleveland iron orefield. Waters from two separate mines (Eston and New Marske) near Middlesborough cause repeated clogging of surface water sewers (Ross, 1998). Before the problem was fully appreciated, several homes had been subjected to repeated flooding as the sewers surcharged. Since Northumbrian Water Ltd adopted the sewers in 1997, more than £10,000 has been spent on sewer rehabilitation to prevent further flooding, while a more permanent solution is sought (Ross, 1998).

E. *Pollution of over-lying aquifers* by upward migration of minewater. The only case documented so far concerns the same southwestern part of the Durham Coalfield (ie south of the Butterknowle Fault) mentioned under B above. The eastern part of this area of coalfield underlies the Permian Magnesian Limestone aquifer, which is widely used for public supply abstractions in County Durham. One mine close to the edge of the Permian outcrop (Mainsforth Colliery) caused propagation of subsidence fractures from the coal workings into the overlying aquifer, resulting in a high, sustained inflow of Magnesian Limestone water (Frost, 1979). After pumping ceased, mine water rebound in the Mainsforth area led to a 10m rise in the water table in the Magnesian Limestone, which was accompanied by a deterioration in water quality as sulphate rich mine waters entered the aquifer (Younger, 1995c).

F. *Temporarily accelerated mine gas emissions* associated with mine water rebound are a cause for concern in many exposed coalfield areas. However, as mine gas emissions vary rapidly in response to fluctuations in barometric pressure (highest release rates corresponding to periods of low barometric pressure), it has proved very difficult to establish a deterministic relationship between mine water rebound and accelerated gas release. However, one case widely believed to exemplify the phenomenon occurred at Widdrington, Northumberland, in 1995. In this fatal case, a man working in a small factory was overcome by carbon dioxide when he entered a well-frequented basement area, which happened to adjoin an old drift portal (Burrell and Friel, 1996). The hitherto unprecedented carbon dioxide accumulation in this area has been ascribed to a local rise in the water table in the old workings.

G. *The risk of subsidence* as rising waters weaken support pillars. The only published example of this phenomenon in the UK to date occurred in 1994 in the Leicestershire Coalfield (Smith and Colls, 1996). As rising mine water levels reached the position of an old barrier of intact coal at a depth of about 140m below ground level, a surface subsidence scar developed. The scar lies in the precise location predicted by subsidence mechanics theory from the mapped edge of the underground pillar (Smith and Colls, 1996). Similar problems in North Tyneside have been noted by Reeves (personal communication, 1994) and attributed to slaking of seat-earths in shallow bord-and-pillar workings which are flooding for the first time in centuries.

Pumping schemes to prevent pollution and other impacts are currently being operated by the Coal Authority in Staffordshire, West Yorkshire (Laine, 1997) and County Durham (Younger, 1993). Similar schemes are being installed in the Dysart-Leven Coalfield, Fife (Younger *et al*, 1995; Younger, 1995b), and for the Central Coalfield around Polkemmet Colliery, West Lothian (Chen *et al*, 1997; Sadler and Rees, 1998). Design and long-term operation of such schemes is costly; for instance, the Durham scheme costs in excess of £1M per annum. This is in itself a disincentive to using this approach, though in particular circumstances, cost-benefit analyses can show that the long-term pumping option is less costly than dealing with the consequences of complete abandonment, at least over discounting periods of a century or more (Younger and Harbourne, 1995). Technical problems associated with developing and maintaining pumping schemes range from the poor predictably of mine water flow paths in the subsurface (complicated by the fact that roof falls in old roadways can alter head distributions even while pumping rates remain stable), to the requirement for submersible pumps in shafts to be adequately shrouded to prevent over-heating.

1.4. Areas Where Future Problems Might Arise

Table 1.2 lists the main areas in the UK where mine water rebound is either underway, or may commence in future if dewatering ends. Some indications of the likely occurrence and severity of the possible impacts A - H identified in Section 1.3.1 above are also given in Table 1.2. The most pressing rebound problems demanding hydrogeological analysis are in the major coalfields of South Yorkshire (Burke, 1997), Northumberland (Younger, 1995b), South Tyneside (Younger, 1995b), Leicestershire (Smith and Colls, 1996), Warwickshire, Asfordby (Vale of Belvoir), north Staffordshire, north Derbyshire (Walker, 1998), Lancashire (Thewsey, 1998), and Kent. Analysis and monitoring already underway in a number of districts will require sustained regulatory vigilance. The main districts in this category are west Cornwall (South Crofty), west Durham (Frazer's Grove), mid-Northumberland (Whittle), Midlothian (Monktonhall), and East Fife (Frances and Michael collieries). In the future, further analyses will also be required for the major coalfields of Durham, west Fife, Nottinghamshire, and south Derbyshire if / when current dewatering arrangements are terminated. For all of these coalfields, there is an increasingly urgent need to consider what can be done to minimise impacts A through H.

Treatment of existing and future polluting discharges could in theory be accomplished by "active treatment" using established alkali dosing and sedimentation technology. Refined treatment methods on these lines are being applied at Wheal Jane, at Bullhouse, West Yorkshire, and at Polkemmet, Scotland. In many cases, the most cost-effective long term strategy will be best identified on hydrogeological grounds, taking into account the evidence for long-term piezometric stability and natural attenuation of pollution in the subsurface. It will often be necessary to use active treatment only during the first few decades after mine water emergence, when the quality is usually at its worst (Younger, 1997a). Subsequently, long-term treatment can be provided by passive means (Younger, 1997b). Such an approach is being pioneered at Ynysarwed, South Wales, with active treatment being implemented in the first instance, with a view to discontinuing its use a decade or so later, when asymptotic mine water quality is expected to be encountered, after which an aerobic wetland will be used in the long term (Ranson and Edwards, 1997; Ranson *et al*, 1998).

Table 1.2. Areas of the UK where future mine water rebound problems may arise.

Area / mine[a]	EA Region	Current status[b]	Possible impacts without prevention scheme[c]	Anticipated severity of impacts[d]	System understanding[e]	Prevention scheme?[f]
South Crofty	South West	R	A	1	4	Y, EA/MC
Kent	South East	R	E	3	1	N
Tower Colliery	Wales	D	None currently anticipated	Not applicable	2	NA
Small drift mines	Wales	D	A	3	1	N
Warwickshire	Midlands	R	A, E	4	1	N
Leicestershire	Midlands	R	A, B	4, 3	1	N
Staffordshire	Midlands	R / D	A, B, E, F	3, 1, 2, 2	1	N
Point of Ayr	Wales	R	None anticipated	Not applicable	3	NA
Asfordby	Midlands	R	E	2	1	N
Nottinghamshire	Midlands	D	A, C, E	2, 3	3	N
Derbyshire	Midlands	R	A, E	2, 3	3	N
South Yorkshire	North East	R	A, B, E, F, G	3, 1, 3, 3, 1	3	N
West Yorkshire	North East	D	A	4	4	Y, CA
Doncaster district	North East	D	A, B, E, F, G	3, 1, 3, 3, 1	1	N
Lancashire	North West	R	A, B, E	3, 1, 3	4	N
Selby complex	North East	D	E	2	2	N
West Cumbria	North West	R	A, E	3, 1	1	N
Durham Coalfield	North East	D	A, B, C, E, F, G, H	4, 1, 3, 3, 2, 2	4	Y, CA
Frazer's Grove	North East	D	None anticipated	Not applicable	4	NA
South Tyneside	North East	R	A, B, E, F	3, 2, 1, 2	1	N
Blenkinsopp Drift	North East	D	A	4	1	N
S. Northumberland	North East	R	A, B, F	3, 2, 2	1	N
Ellington	North East	D	A, B, F	3, 2, 2	1	N
Whittle-Shilbottle	North East	R	A, B, F	5, 2, 4	4	Y, CA / EA
Monktonhall	SEPA	R	A, B, F	4, 1, 1	3	Y, CA
Polkemmet	SEPA	D	A, B, F	4, 1, 1	4	Y, CA
Longannet	SEPA	D	A, B, F	3, 1, 1	1	N
Frances & Michael	SEPA	R	A, B, F	4, 1, 1	4	Y, CA

[a] locations on Figure 1.1. [b] at 31-12-1998: D = pumped dewatering, R = rebound underway. [c] A – H as defined in Section 1.3.1. [d] respectively for each impact listed, on a scale of 1 – 5 (5 worst). [e] i.e. level of understanding of hydrogeology on a scale of 1 – 5 (5 best). [f] Is there an agreed scheme to prevent post rebound impacts (Y/N/NA)? If "Y", the responsible party is identified: CA = Coal Authority, EA = Environment Agency, MC = mining company.

1.5. The Rationale For Predicting Mine Water Rebound

Why is it found to be worth the expenditure of resources on predicting mine water rebound? Recent experiences around the UK illustrate several reasons:

(i) The decision over whether to allow rebound to continue unabated, or to implement a preventative pump-and-treat scheme (or other solution), needs to be informed by a thorough understanding of hydrogeological risks (eg Younger *et al*, 1995) which can then be evaluated within a framework of social and environmental cost-benefit analysis (eg Younger and Harbourne, 1995; Knight Piésold, 1997).

(ii) While rebound predictions for coal mines may be undertaken by mining companies and/or the Coal Authority, the Environment Agency will often be called upon to evaluate these predictions. The ability to run independent checks on the work of external organisations may be valuable in the occasional cases where the interpretation is contentious. Furthermore, as the remit of the Coal Authority extends only to coal mines, it will often fall to the Agency itself to undertake rebound modelling for other types of mines.

(iii) There have been some cases where a *presumption* that the closure of a particular mine would lead to serious pollution has been shown to be incorrect (eg Younger, 1998b). Had action been based on the initial presumption alone, vast resources would have been unnecessarily squandered.

(iv) There are related issues of public concern, generally beyond the remit of the Environment Agency (and hence not considered in detail in this report), which nevertheless benefit from an appraisal of the status and likely evolution of mine water rebound. Chief examples of this are problems related to stythe gas and subsidence / tectonic activity triggered by rebound (Section 1.3.1). Both of these issues generally fall within the remit of the Coal Authority, although neither is free from technical controversy and/or problems of legal definition.

(v) In all cases to date, the costs of undertaking a hydrogeological investigation of a possible rebound scenario have been on the order of 1 – 2% of the *capital* costs of implementing a remediation scheme. When the *operating* costs of such schemes are taken into account, the provision of a sound hydrogeological basis for system design becomes vanishingly inexpensive.

The remainder of this report is concerned with providing practical guidance for hydrogeologists who may need to evaluate problems associated with mine water rebound. It is assumed that the reader has a basic familiarity with standard hydrogeological terminology and concepts, but has no previous knowledge of mining hydrogeology. For those hydrogeologists already familiar with mining, some of the earlier sections of the report may be found tedious. However, in the later sections of the report we present state-of-the-art insights and methods for mine water analysis that have not previously appeared in print; it is anticipated that these will be found interesting and innovative by even the most experienced.

The order of presentation is meant to reproduce the order in which an actual investigation will ideally proceed:

(i) we begin with concepts of mine system behaviour (Chapter 2)

(ii) we next apply simple manual methods of analysis (Chapter 3), which may be sufficient on their own or the necessary precursor for computer-based analyses

(iii) if necessary / feasible, we move on to computer-assisted analysis, using either the simple semi-distributed modelling approach enshrined in the GRAM code (Chapter 4) or the full-blown, 3-D, variably saturated, turbulent and laminar flow code VSS-NET (Chapter 5).

(iv) Finally, as a post-processing activity after the flow system has been analysed, we use empirical relationships derived from existing mine water discharges to make some predictions of the short- and long-term quality of mine water discharges to surface waters and overlying aquifers (Chapter 6).

2. THE HYDROGEOLOGY OF MINED SYSTEMS

2.1. Are Mined Systems Aquifers?

An aquifer is a water-bearing body of rock which stores and transmits water. The disturbance of the subsurface caused by mining creates substantial systems of inter-connected voids which, once flooded, typically display the storage and transmission functions characteristic of aquifers. For instance, in County Durham, the mined Coal Measures yield more water to nine dewatering stations than is currently yielded by the nearby Magnesian Limestone aquifer (which has a similar total outcrop area) to a similar number of public supply wells.

So mined systems are aquifers. However, because of the size and engineered connectivity of mined voids, the aquifer behaviour of flooded, abandoned, mine voids often has little in common with the Darcian, laminar-flow aquifers which account for most groundwater resources. Undoubtedly, mined systems do display marked behavioural similarities to karstified limestones (which are also widely used as water resources) and evaporites, although they are also distinguished from these closest natural relatives by the particular geometries of voids present and the lateral scale of inter-connection.

The styles of voids produced by mining are summarised below. In terms of the lateral scale of inter-connection, it is not uncommon for natural karst systems to have many hundreds of kilometres of interconnected passageways, particularly in the case of labyrinth caves. However, the largest labyrinth cave systems usually underlie a surface zone with an area of only a few tens of km^2 (Middleton and Waltham, 1986). For instance, the determined visitor to Mammoth Caves in Kentucky may walk through up to 530 km of underground passageways, only to emerge a mere 6 km (as the crow flies) from the point of entry. By contrast, regionally-mined coalfields in Europe often underlie areas of many hundreds of km^2, and are frequently inter-connected over many tens of kilometres as the crow flies. It is a truism that, until the late 1960s when closures commenced in earnest, one could have gone underground in the south-west corner of the Durham coalfield, and finally returned to the surface in South Shields, a full 50 km to the north-east (Harrison *et al*, 1989).

So, we may conclude that regionally-mined systems resemble karst aquifers to some degree, save that they are often much more laterally extensive than single karst drainage systems. Mined systems may perhaps be classified as "man-made mega-karst". This brings us cold comfort, however, since few hydrogeologists are ever fully at ease when analysing karstic flow systems. A common response to modelling karst is to brush over the fact that the system does not compare very well with a sand filter (the device for which Darcy's Law was, after all, devised), and apply a standard Darcian flow-code (e.g. Cullen and LaFleur, 1984), such as MODFLOW (McDonald and Harbaugh, 1984). Where the modeller seeks no more than a regional water-balance, this approach is unlikely to disappoint, though it remains no more defensible as an algorithm than the simplest of "black box" calculations. As soon as some detail is required (e.g. reproduction of the details of a spring hydrograph, or a piezometric record at a given point) then problems are typically encountered. Numerous attempts have been made to devise more appropriate models for karst aquifers at this level of detail. For instance, Ford and Williams (1989) describe "grey box" models, in which flow calculations are kept as simple as possible, but available information on cave geometry and void distribution (e.g. the presence of epikarst) is taken into account when formulating transfer coefficients. Two-continuum models of regional karst aquifers have been developed by Teutsch and Sauter (1990) by superimposing two MODFLOW grids, one representing relatively slow flow through small fractures, the other representing (by means of extremely high T values) flow through the major conduits. More recently, a model named CAVE (Carbonate Aquifer Void Evolution) has been developed in which a single-layer MODFLOW model has a 2-D pipe-network routed through it, representing caves and other large cylindrical conduits, in which the flow can be represented as either laminar or turbulent according to ambient conditions (Younger *et al*, 1997). This model also allows for simulation of the development of caves by calcite dissolution in the pipe-network. Hanna and Rajaram (1998) have recently described a similar model in which the "fast flow" network is represented by a "parallel plate" fracture model using the Poiseuille equation. To date, no karst model has been developed which allows the direct representation of fully-3D conduit networks routed through variably-saturated porous media, though this is precisely the kind of formulation that will be required if physically-based simulation of real systems is ever to be realised.

Turning to mined systems, we find a similar reluctance to eschew simple, laminar- flow, Darcian models in favour of more realistic representations. Where the model is applied only to conditions in the strata surrounding the mine voids, MODFLOW may be used successfully. For instance Toran and Bradbury (1988) used MODFLOW to simulate piezometric drawdown (during mining) and subsequent recovery (after mine closure) for strata surrounding a lead-zinc mine. Despite their efforts being considered a qualified success, it proved impractical to apply the 3-D capabilities of MODFLOW to this system. Problems were encountered with achieving sufficient discretisation, both in space and in time. Paulino *et al* (1997) overcame most of these difficulties in a collaborative geotechnical-hydrogeological investigation of changes in permeability arising from active subsidence above longwall workings in northern Spain. However, where the model must simulate flow in flooded voids, rather than (or as well as) in the surrounding strata, MODFLOW and similar laminar-flow models appear to be less applicable. For instance, when Lancaster (1995) and Sherwood (1997) applied MODFLOW to two multi-seam lowland coalfields in the UK, they found it was difficult to avoid non-convergence if all worked seams were modelled explicitly. Problems are particularly acute in faulted sequences, and where dips are steep. Definition of boundary conditions also proved particularly problematic for a case involving active rebound.

In the light of the problems encountered in applying MODFLOW and similar codes to rebound problems, new approaches to the prediction of rebound have been developed. These approaches are described in detail in Chapters 4 & 5 of this report. In order to understand the new techniques presented in those chapters, it is necessary first to appreciate the nature of mine voids and the hydrological behaviour they exhibit. The remainder of this Chapter is dedicated to this task.

2.2. Mining Techniques and the Geometry of Mine Voids

To understand the geometry of underground voids, it is necessary to have an outline appreciation of the activities involved in mining. In essence, the working of all underground mines involves the following activities:

1. Development and maintenance of access (for humans, machinery, ancillary services and ventilation).

2. Extraction of as much valuable mineral product as possible.

3. Handling of waste rock.

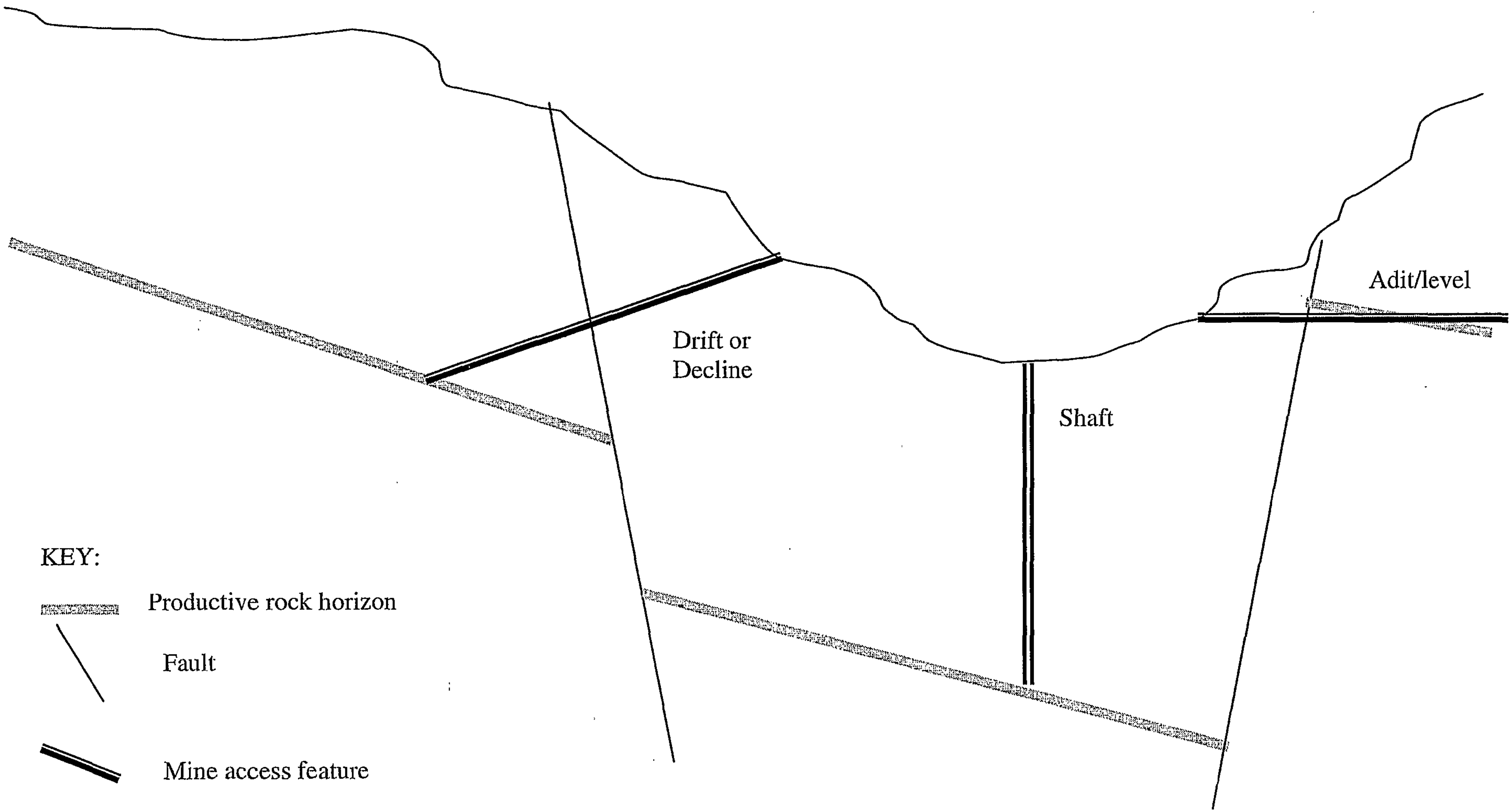
The variations in each of these activities produce distinctive patterns of underground voids, with implications for the short- and long-term hydrological behaviour of the mined system.

Access to deep mines is achieved by tunnels varying in inclination from horizontal to vertical. Although examples of every conceivable angle could be cited, the most common forms of deep mine access are (Figure 2.1):

- shaft - a vertical or sub-vertical (> 70°) tunnel connecting the workings to the ground surface.
- adit - an essentially horizontal tunnel from the ground surface to adjoining workings, particularly common in hilly areas. (In coal-mining districts, "drift" is often used synonymously with "adit", though "drift" is also used to mean other underground tunnels).
- decline - an inclined tunnel from ground surface to the workings. The inclined tunnel may also be termed an "incline", an "inclined drift", a "slope entry" or (in the north Pennines) a "dib".

Shafts are constructed for a variety of purposes, and thus in a corresponding variety of plan forms and diameters. The type of shaft which most commonly springs to mind is the archetypal "main shaft" used for "man-riding", and usually also for transport of materials, for drawing of the run-of-mine product to bank, and as a conduit for the downcast ventilation current (see Glossary). A man-riding shaft will typically be of large diameter (commonly in the range 5m to 8m) and may be circular or rectangular in plan form. In most countries, it is a legal requirement for every deep mine to have at least two means of egress from the deep workings, to ensure escape in case of unforeseen equipment failure or roof-falls etc. The maintenance of a reliable ventilation system is also greatly facilitated by the presence of two deep mine access routes; otherwise the downcast and upcast streams must be carried in the same shaft / adit, separated by brattices. It is therefore common to find two major shafts in close proximity (often within 50m of each other) at many mine sites. Because the long-term viability of the mine will depend on the unhindered use of the main shafts, they are generally engineered to an extremely high-quality specification, and their immediate surroundings (for a radius of 100m or more) will often be left unworked (save for penetration by main access roadways), so that collapse of old workings cannot threaten the structural integrity of the shaft. The radial zone of unworked strata around a main shaft is termed a "shaft pillar". For these reasons, main shafts can be confidently expected to remain open indefinitely, with little or no maintenance.

Figure 2.1. Means of access to deep mine workings



Besides "main shafts", there are several other types of shaft, which are frequently smaller in diameter and (often) not engineered to quite so high a specification. The most common shafts are ***"air shafts"***, which are found at regular intervals along the line of most adits. As the name suggests, these are constructed to allow a through-current of ventilation in the mine. Because there was no requirement for frequent access of men or materials, these air shafts are typically as narrow in diameter as was economically feasible, given the available extraction technology. For instance, diameters of as little as 1.5m to 2m are common amongst the 18th and 19th Century hand-dug air-shafts in Europe. Some of these old air shafts were deliberately constructed to allow ingress of water, which was allowed to fall down the shaft in a narrow (< 100mm diameter) pipe, trapping air in the process and thus producing a blast of compressed air at the foot of the shaft (Sopwith, 1833). In more recent times, air shafts have often been replaced altogether by large-diameter boreholes (250 - 750mm). A further type of shaft which can cause confusion when interpreting old mine site plans is the ***"counter-weight shaft"***. These were constructed to allow the descent of a weight on the end of a cable, as an essential part of the operation of certain 18th and 19th Century winding mechanisms. As with air shafts, they were generally as narrow as possible, with diameters frequently in the range 2m to 4m.

In the early history of mining in most parts of the world, the first shafts are shallow, informally constructed ***"bell-pits"***, so named because of their shape in vertical profile. These are instinctively constructed under primitive mining conditions: when a valuable deposit is followed in from outcrop in a simple adit, it is soon discovered that there are limits to the span width of an unsupported roof before collapse occurs. Extending down-dip from the first line of collapses, a shallow shaft will be sunk to the productive bed, and working continued radially outwards until the limit of roof stability is reached once more. Ancient bell-pits are common in the UK (albeit almost always unmapped), and their environmental significance lies in their capacity to provide the final subsurface flow-paths for mine waters emerging from more recent workings abandoned down-dip.

Not all shafts reach daylight. A significant number of shafts are constructed entirely underground, to provide access to the more remote zones of a dipping ore-body. An underground shaft is usually termed a ***"staple"***[1] or "staple shaft" in coal mines, and a "sub-vertical shaft" ("sub" indicating "subsurface" in this case) in hard rock mines. Alternative terms include "hopper" or "ore chute" (for a staple down which mined rock is tipped to main haulage roadways), "rise" (where a narrow shaft is driven upwards to access higher working zones), "sump" (in lead mining circles) and "winze" (in Cornish copper and tin mining).

Shafts have four principal drawbacks as mine access features:

(i) the high cost, which is often more than half of the total capital cost of a deep mine.

(ii) in flat-lying stratiform deposits, little or no payable rock will be extracted during shaft sinking.

(iii) the full weight of the mined rock must be taken by the winding gear cables. This necessitates the use of very high-specification winch plant, which is expensive to purchase and maintain. At the present level of technology, the deepest single shaft in the world (South Shaft, Western Deep Levels gold mine, South Africa) is limited to 2700m by the technical limitations of winding equipment. The deepest coal mine shaft ever sunk in the UK was at Bickershaw Colliery, South Lancashire, which accessed workings at over 1300m total depth. Metal mine shafts were rarely very deep, with the recently-abandoned

[1] "Staple" is pronounced to rhyme with "apple"

shafts at South Crofty (770m depth) being very deep by Cornish standards. The deepest shaft currently working in the UK is 1100m deep, at the Boulby potash mine, Cleveland.

(iv) water in the workings must always be pumped out of the mine, for it cannot flow out by gravity.

Where their installation is feasible, ***adits*** have the potential to mitigate all of these drawbacks: in stratiform deposits, they can often be driven in the productive horizon, so that revenue generated during construction can defray the cost of excavation. Moreover, in horizontal or gently-inclined adits, much of the weight of the mined rock can be borne by the strata (via rails in most cases), making the haulage plant concomitantly less expensive to install and maintain. Finally, for waters entering the workings above adit-level, simple gravity drainage can be used for dewatering. So great an advantage is this last factor that many long adits have been driven beneath districts formerly mined using shafts in order to remove the need for pumping from shafts. In most extensive mining districts, mining will eventually extend below the lowest feasible gravity-drainage level. In such cases, old adits are often maintained as routes for waters pumped from below, as they at least offer the potential to save the cost of pumping against the greater head needed to raise water all the way to the shaft collar.

Major drainage adits will never be truly horizontal, for an out-bye gradient (ie a gradient towards the portal at the ground surface) of 1 in 500 or more must be maintained if the adit is to be self-draining. Sopwith (1833) describes the typical dimensions of early 19th Century adits as "3 feet [0.91m] wide at the bottom, gradually widening to middle height, where it is 3½ or 4 feet [1.07 or 1.22m] wide, and from thence it has an arched form to the top, which is from 6 to 7 feet [1.82 to 2.13m] high". Historically, haulage in such adits was provided by ponies pulling large tubs on rails. For this reason, it is not unusual to find major access adits designated as "horse levels" in old records. Modern adits are rather more generous in proportions than the old horse levels, with both the median width and maximum roof clearance often being around 3m. This provides sufficient room for haulage of tubs by means of locomotives, or by rope haulage to a standing engine adjacent to the portal. Alternatively, work can be brought to bank by conveyor belts.

In certain circumstances, particularly where stratiform ore-bodies have a significant structural dip, the most economic mine access may be a ***decline*** or ***drift*** (Figure 2.1). Where the angle of dip of the decline can be less than about 6°, it should be possible to work the mine using free-travelling vehicles (usually diesel powered). Conveyor belts can handle steeper gradients, but cable-hauled rail tubs are likely to be the best option where the dip steepens beyond about 10°. Beyond these considerations, declines hardly differ at all in form from ordinary circum-horizontal adits.

Having obtained subsurface access via a shaft, adit or decline, the next major access feature of any deep mine is the "***level***" or ***major roadway***. These are simply underground tunnels, branching off the principal mine access. (The terminology of the point at which a main level branches off the principal access differs for the case of a shaft (for which the branch occurs at an "inset" or "shaft station"), an adit (which branches at "junctions") or a decline (which branches at "landings")). The largest levels may have sufficient space for two pairs of locomotive rails, which means that they can easily have diameters in excess of 8m. These larger diameter levels are also relatively uncluttered with fittings, and are engineered for permanence. As the major levels are traced in towards the working areas, the smaller district roadways ("main gates") are entered. In these district roadways, the diameter of the arch girders will usually decrease once more (to 4m or less). Furthermore, with a shorter access life, it is possible for the roof and wall supports to become less permanent in nature. Conveyor belts and other equipment associated with zones of active extraction can begin to use up much of the cross-sectional area of a district roadway.

Having traced the major access routes of a deep mine towards the working zone, we must now consider the various extraction techniques. There are two principal classes of extraction technique: Where the productive zone is essentially ***stratiform*** (ie disposed in layers), and the dip is low (less than, say, 15^{o}) then some of the classic "horizontal" extraction methods will be applicable, such as room-and-pillar and longwall. If a stratiform productive zone is steeply-dipping (as in classic hydrothermal vein / lode deposits), or if an ore-body is of irregular shape though with significant vertical extension, then extraction most commonly proceeds by means of the various "stoping" techniques.

Horizontal extraction methods are of two principal types:

- ***The "room-and-pillar"*** method (also called "bord-and-pillar", "stoop-and-room", and "pillar-and-stall") was the principal means of extraction of European stratiform deposits for many centuries, and remains the principal technique of deep mining for coal in the USA (Hartman, 1987). In this method (Figure 2.2), rooms are excavated on an approximately rectilinear plan, leaving substantial "pillars" of unworked ore / coal to support the roof. Typical dimensions of modern room-and-pillar operations in coal are (Hartman, 1987):

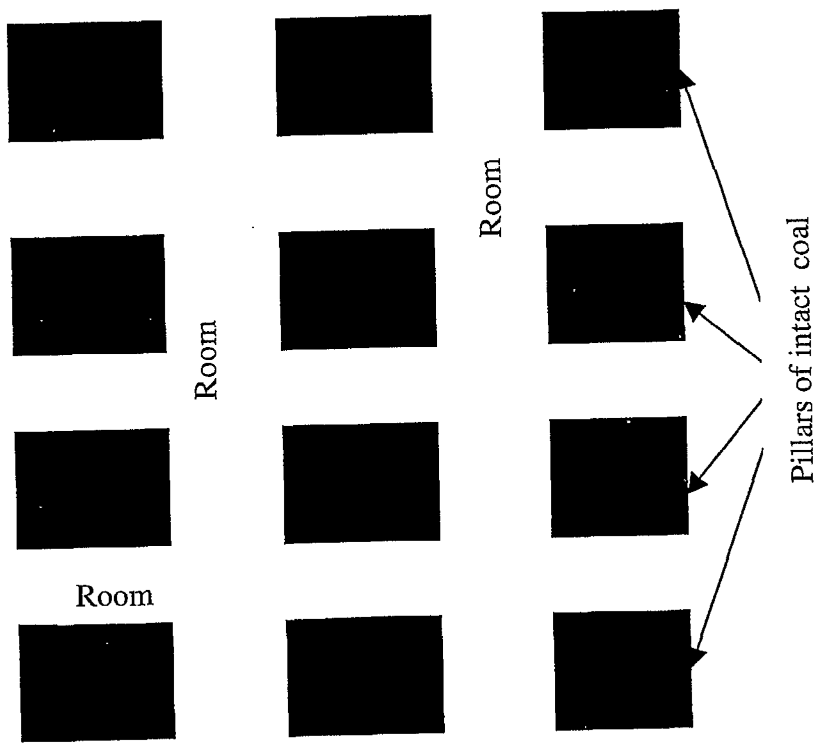
  "room" widths: 6m to 9m (larger openings requiring auxiliary roof support)
  "pillar" widths: 9m - 30m.

  These dimensions have been derived from centuries of experience in Carboniferous coal-bearing strata world-wide. Adjustments to these recommendations will be necessary in practice, depending on roof strata competence etc. In the Cheshire salt mines, for instance, rooms and pillars are usually both 30m – 35m wide, and 8m high. The maximum extraction rate normally attainable by single-pass room-and-pillar working in coal is around 48%. However, fully-mechanised room-and-pillar workings can extract as much as 60% in a single pass (Hartman, 1987). Even higher extraction rates (70 - 90%) are attainable if the roof-supporting pillars are removed during the retreat from a given district of a mine. (This strategy is variously termed "pillar extraction", "pillar recovery", "second mining" (the initial room excavation being "first mining"), "pillar robbing" or "working the broken"). With the pillars removed, the roof is left to collapse in the worked-out area, the accumulated roof-fall debris being termed "goaf" or "gob"[2]. One of the advantages of room-and-pillar is that it can operate in strata dipping as steeply as 20^{0} without problems; at such dips most longwall methods fail to be viable. For this reason, room-and-pillar retains a specialist niche even in mining regions where longwall is now the norm. Another advantage of room-and-pillar is economic: capital investment is relatively modest compared to longwall. This is precisely why the technique remains so popular in the USA, where most mining is undertaken by relatively small, private sector companies with limited investment capital. In Europe, where state-owned mining companies have been common until recent decades, the initial investment associated with longwall (which is, over the long-term, cheaper and more efficient than room-and-pillar) has not been such a disincentive.

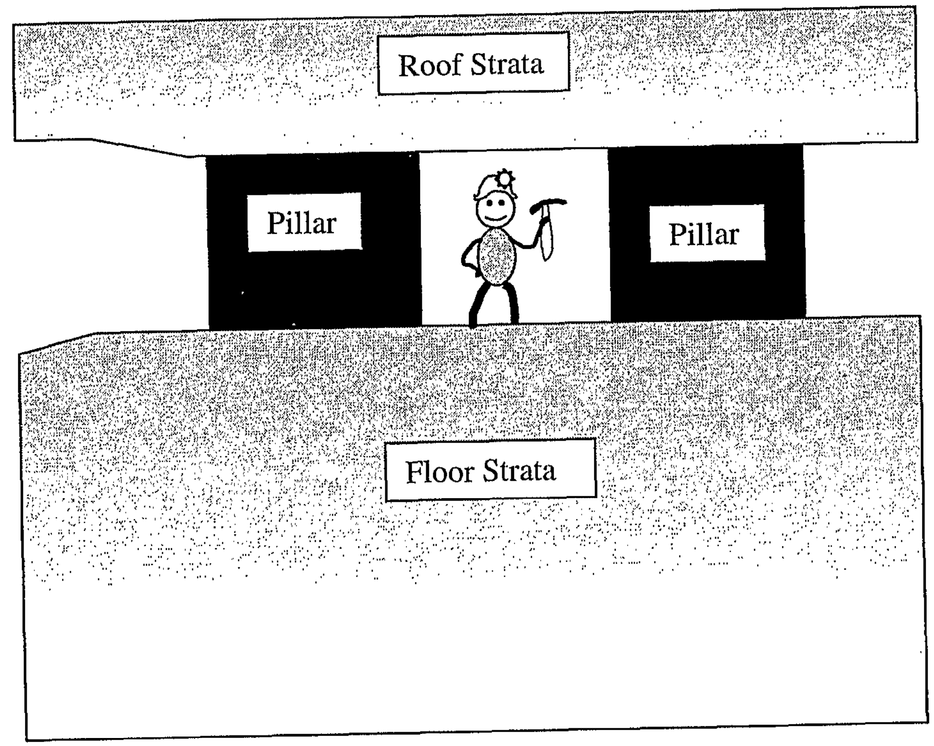
[2] "Goaf" and "gob" are both derived from the Welsh word "ogof", meaning a cave.

Figure 2.2. Layout of "room-and-pillar" mine workings

(a) Plan View



(b) View in Profile



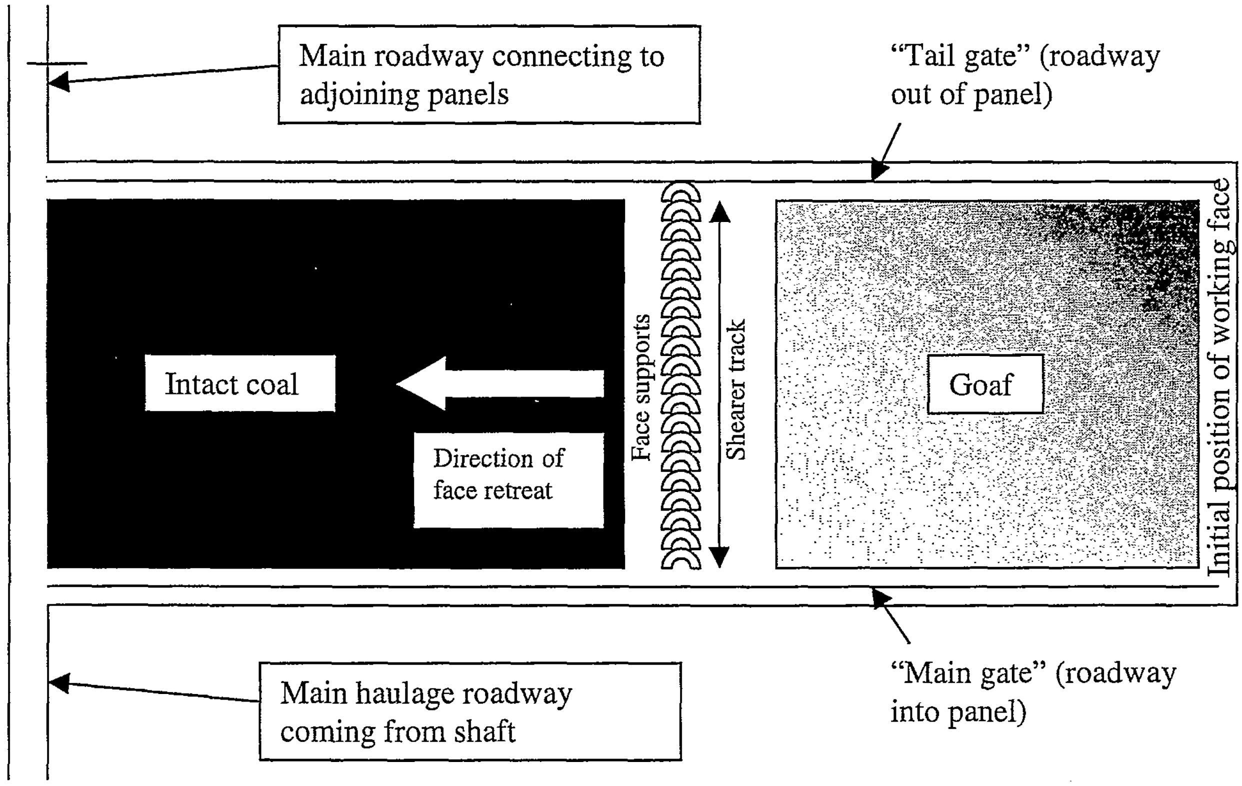
- ***Longwall*** extraction has become widespread only in the 20th Century. It is now the most common deep coal mining technique in Europe (accounting for more than 80% of total production) but has only begun to penetrate the US industry since the 1960s, where it still accounts for little more than 15% of deep-mined coal. The longwall technique offers the advantage of extraction rates as high as 90% in a single pass. The layout of longwall panels in a typical deep coal mine operation is shown in Figure 2.3. A drum shearer passes back and forth along the working face, trimming the coal as it goes. As the face advances (or retreats towards the main access routes, depending on the orientation of the shearer), self-advancing hydraulic supports gradually move (retract, advance, re-engage) to maintain roof support over the shearer. In the extracted area behind the hydraulic supports, the roof is left unsupported, and it collapses to form goaf (as in the pillar extraction phase of room-and-pillar mining). Each longwall panel will be up to 250m wide, and as much as 1 km in length. Where ground conditions are difficult, it may be necessary to reduce these panel dimensions, such that mines can effectively end up with a "***shortwall***" mining method, with panels as little as 40m wide by 200m long. (Further consideration of possible hydrological motives for a switch from longwall to shortwall are discussed in Section 2.4).

Extraction over a significant vertical extension is generally termed ***"stoping"***. At its simplest, stoping may be little more than the application of room-and-pillar techniques to steeply inclined, non-coal strata (in which case it is often termed "***stope-and-pillar***"; Hartman, 1987). Given the variations in ore grades of hydrothermal vein deposits etc, there is usually a strong incentive to make the pillars coincide with patches of low-grade ore. Hence pillars in stoping operations are often smaller, and of less regular plan-form than pillars in coal workings. This means that single pass mining (for pillar removal is rare in stope-and-pillar operations) can yield 60% to 80% recoveries.

In some mines, extraction is carried out by a so-called ***overhand stoping*** method, whereby successively higher portions of the vein are worked in an ascending manner from the main haulage roadway. (The alternative, *underhand stoping*, i.e. working downwards from the main haulage level, is rarely used anymore). The ore is tipped down chutes to draw-points (short insets driven into the vein from the main haulage roadway) whence it can be "mucked-out" and filled into tubs or onto conveyors. The waste rock (or "deads") is allowed to fill the underlying void between the draw-points. More modern overhand extraction is achieved by "***shrinkage stoping***". In this approach, the roof above the draw-points is brought down repeatedly by blasting, with the miners standing on the pile of previously blasted ore in order to prepare the roof for the next blast. Occasional mucking-out of the draw-points is necessary to maintain space for working on top of the pile of broken ore. Although shrinkage stoping has now been largely superseded in many mines, it retains a niche for situations where "dilution" of the ore by country rock must be minimised by carefully tailored blasting.

Figure 2.3. Layout of Typical Longwall Mine Workings.

(a) An individual longwall panel (retreat working mode)



(b) Layout of a series of panels in relation to main haulage and shaft access.

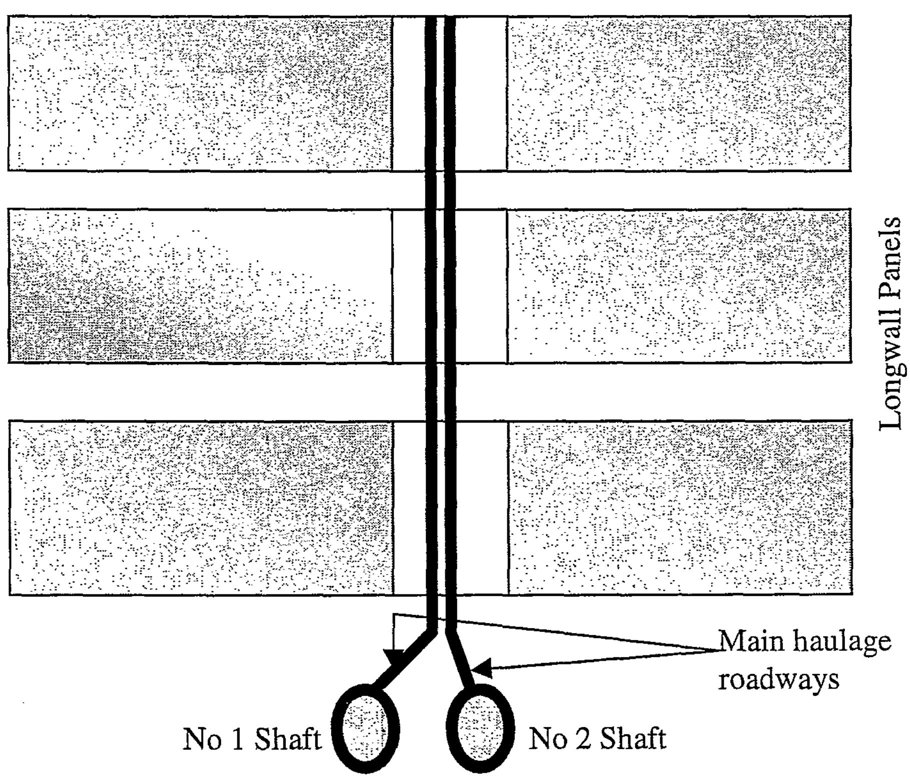
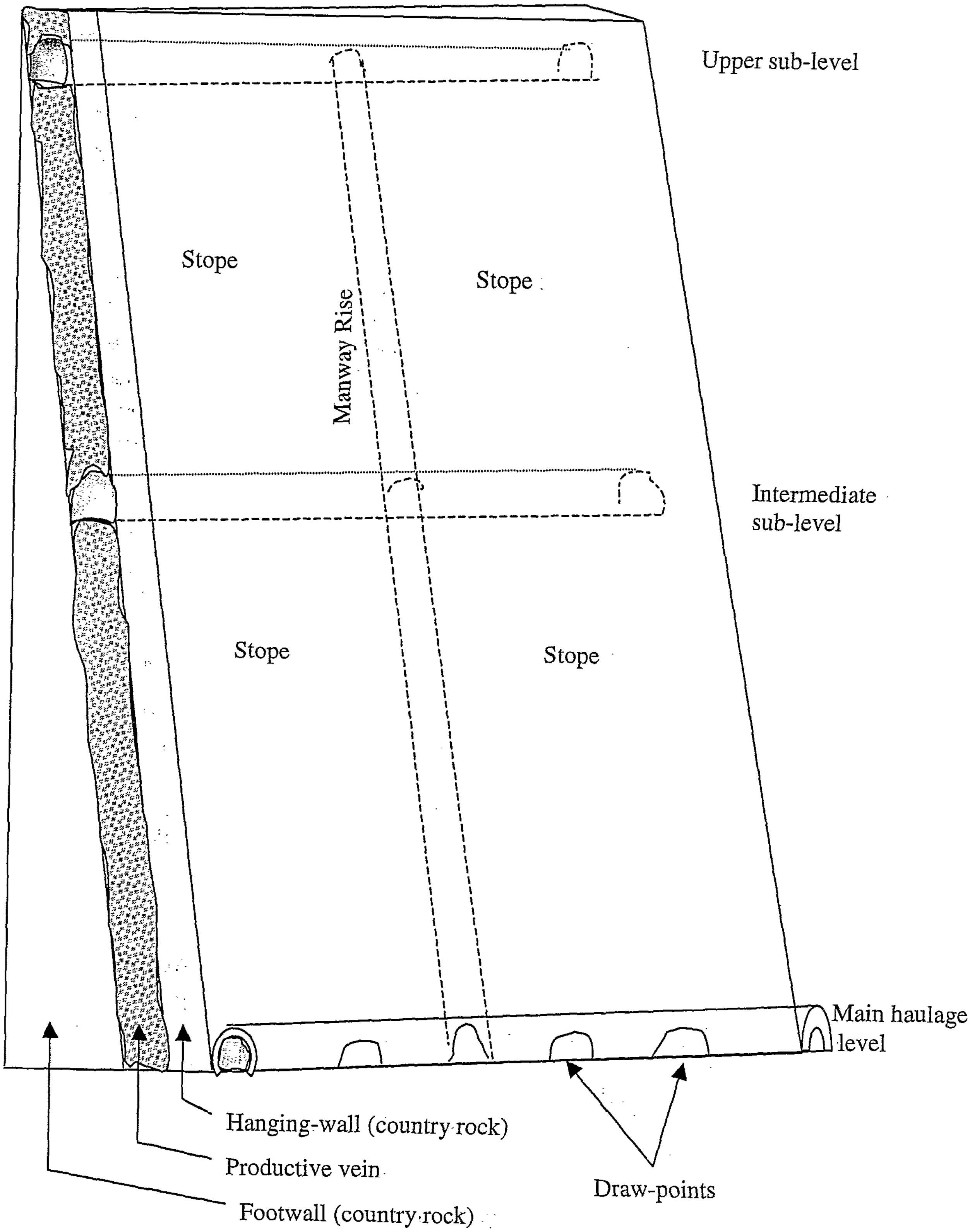


Figure 2.4. Layout of typical sub-level stoping workings in a near-vertical mineral vein.



In most modern stoping operations, extraction is achieved by "***sub-level stoping***", which is undertaken as follows (Figure 2.4):

- semi-permanent haulage levels are driven in the country rock in the hanging-wall of the vein, with short "***draw-points***" driven into the vein every10m or so.
- higher portions of the vein (above the draw-points) are accessed by "***manway rises***" (narrow shafts fitted with ladders). At discrete intervals, horizontal tunnels (termed "***sub-levels***") are driven along strike, within the vein, to the maximum extent of the working panel (typically 150m - 250m).
- drilling equipment is then hauled into the sub-levels, and is used to drill "longholes" (ie boreholes extending downwards, maybe two-thirds of the distance back towards the draw-points) in the first section of tunnel (along maybe 10m of strike). The longholes are charged with explosives, stemmed as necessary, and fired at the end of each shift. The broken ore falls under gravity into the draw-points below, whence it is extracted ("mucked out") and loaded into tubs or onto a conveyor belt.
- the drilling and firing of longholes is then repeated all the way back along the sub-level to the manway rise over the succeeding days, leaving an empty stope behind.

Sub-level stoping is a high-extraction rate "caving" method of mining, analogous in concept (though not in equipment) to longwall mining (Hartman, 1987). Variations of sub-level stoping (eg "vertical crate retreat") are recognised on the basis of the details of the positions and types of explosives used.

Hydrologically, the final voids created by all of the variants of sub-level stoping (and indeed by shrinkage stoping) are all essentially the same. As long as room-and-pillar voids stand open, they are also cavernous. In longwall workings, the collapse of the unsupported roof to form goaf results in fewer cavernous voids remaining after mining, with the exception of the major roadways, associated bays and shaft stations, all of which were engineered for permanence. Having thus an appreciation of the mine voids themselves, the final detail requiring appreciation is the effect which mining has on the condition of the roof and floor strata.

2.3. Effects of Mining on Surrounding Strata

In theory, in room-and-pillar working, and in many kinds of stoping in hard rock, the mined voids are expected to stand open virtually indefinitely. If they do remain open, they will have little impact on the surrounding strata, other than inducing an "envelope" of net-compression around the void. In some cases, the compression may be sufficiently great that some of the surrounding rock may deform in a ductile manner, migrating into the void in the form of "floor heave" (whereby the floor of the void bulges upwards, reducing the void space). This is fairly common in coal mines, at least where the floor of the seam is an incompetent seat-earth. At great depths (typically > 1000m), the migration of the surrounding rock into the void may occur violently, in the form of "rock bursts", with fragments of wall rock suddenly ejected into the void as fast as (and as dangerous as) bullets from a gun. At the shallow depths relevant to most mine water rebound studies, these effects are usually minimal.

Even where floor heave and rock bursts are unimportant, room-and-pillar voids and stopes often do collapse, for two main reasons:

(i) They may have been excavated with roof spans greater than those permitted by the strength of the country rock. In this case, the voids will never have been other than "meta-stable", and will have required frequent maintenance during the working of the mine. The stable roof span for fissile shale is very narrow, and hence where the roof of a void was originally shale, flaking of layers from the roof is common, and leads to "void migration", i.e. effective movement of the void upwards as roof debris piles onto the floor, below a growing roof hollow. The size of the final void after migration ceases is always less than the original mined dimensions, but is rarely zero.

(ii) There has been some post-mining change in the effective stress regime in and around the void due to changes in, for instance:

- loading (e.g. new construction occurring over shallow voids)
- buoyant support (if formerly flooded voids are once more dewatered)
- saturation (if the floor or roof strata become weakened upon wetting).

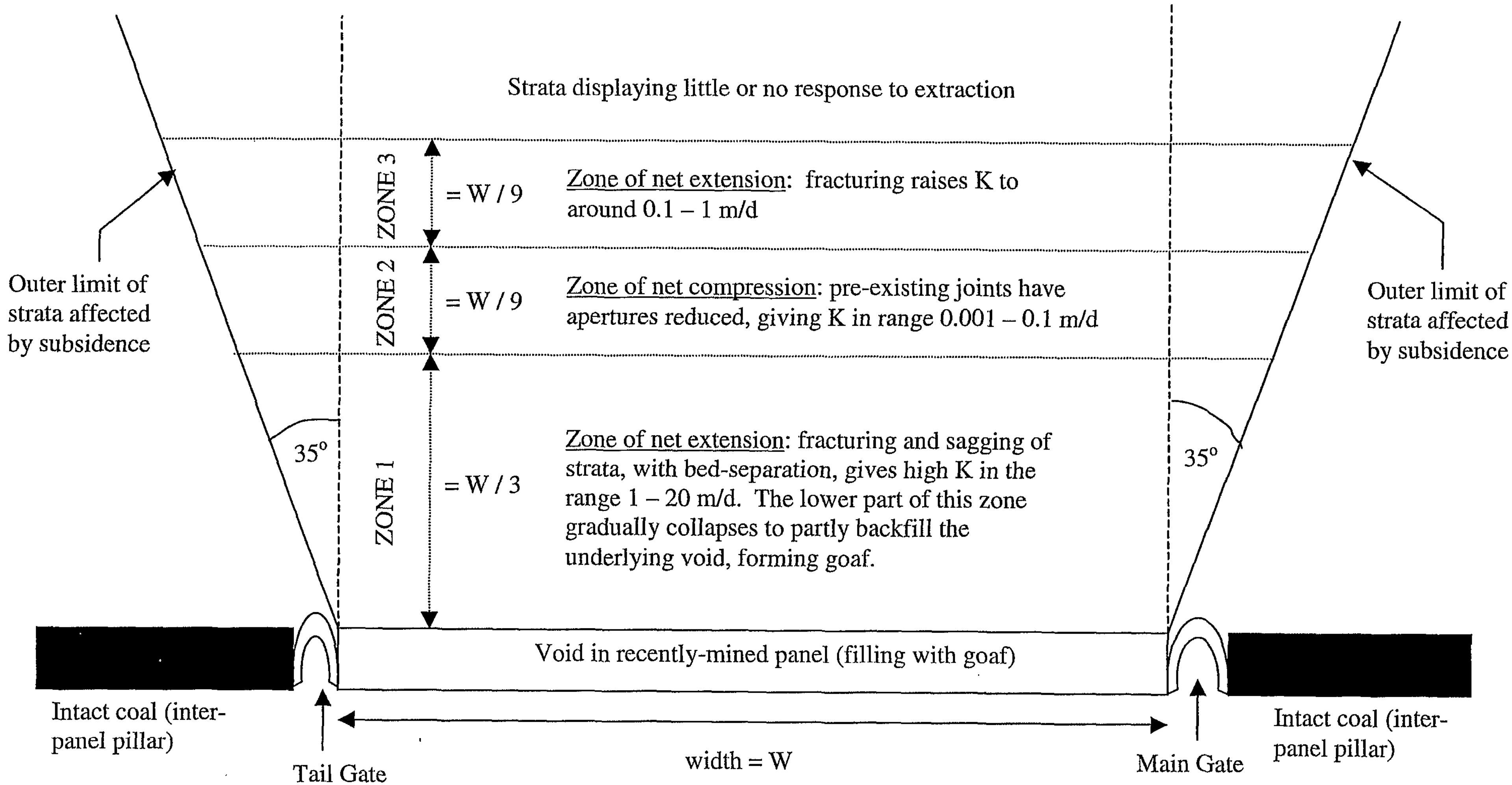
If collapse occurs, bending and fracturing of the overlying strata is inevitable. However the details of such bending and fracturing is poorly characterised for the case of unanticipated failure of voids which were designed to be permanent. Far more is known about the response of roof strata to deliberate "complete collapse" methods of mining, particularly longwall. As this response is accompanied by changes in the hydrogeological behaviour of the roof strata, it is worthy of brief review here. A full discussion of the rock mechanics processes associated with subsidence above deep mine workings is beyond the scope of this report; readers seeking erudite explanations on this theme are directed to NCB (1975), Orchard (1975), Whittaker et al (1979), Singh and Atkins (1983) and Whittaker and Reddish (1989). To simplify and summarise the hydrogeological consequences of these processes, two key points should be noted here:

(i) Stress fields which develop above an unsupported, recently created void cause fracturing and subsidence of the overlying strata over a vertical distance which is typically half as high as the void is wide (eg a 200m-wide longwall panel can be expected to affect around 100m of overlying strata).

(ii) The volume of rock affected by fracturing and subsidence can be resolved into three zones (Figure 2.5) essentially parallel to the void; in each of these zones, the changes in permeability in response to extraction are distinct, and may be summarised as follows (in ascending order above the void):

Zone 1: This is the first zone above the unsupported void, and is typically one-third as high as the void is wide (eg over a 200m-wide panel, Zone 1 would be about 65m thick). Zone 1 is characterised by collapse and fracturing of the strata (by a combination of sagging, extensional fracturing and bed separation), which increases the permeability of the strata (when compared to the pre-mining permeability) by factors as high as 60 to 80 (Whittaker et al, 1979; Singh and Atkins, 1983; Fawcett et al, 1984).

Figure 2.5. Schematic diagram of the zones of extraction-related subsidence above a recently worked longwall panel.



Field tests suggest that, in typical Carboniferous coal-bearing strata, hydraulic conductivities in the range 1 to 20 m/d are common in this zone (Whittaker et al, 1979; Minett et al, 1986; Aljoe and Hawkins, 1994; Ferry et al, 1994).

Zone 2: This is usually about 25% - 30% as thick as the underlying Zone 1 (eg over a 65m thick Zone 1, Zone 2 may be 15 - 20m thick). In Zone 2, the net effect of the subsidence-related stress fields is compression, so that the permeability of the strata remains at or below pre-mining values (or may even decrease). For instance, in Carboniferous coal-bearing strata, hydraulic conductivities (K) in the range 0.001 to 0.1 m/d are likely in Zone 2 (Miller and Thompson, 1974; Minett et al, 1986; Ferry et al, 1994; Aljoe and Hawkins, 1994). Zone 2 can thus function as a valuable "low-permeability barrier" above workings. For this reason, the position of Zone 2 in relation to overlying aquifers or flooded old workings critically controls whether subsidence will induce greater flows of water from above into the mine.

Zone 3: This is usually similar in thickness to Zone 2. Zone 3 is an extensional zone, so that permeability again increases (though not by so much as in Zone 1).

2.4. The Hydrogeological Behaviour of Mine Workings

2.4.1. Water as the enemy of the miner

Until very recently, most hydrogeological observations reported in the mining engineering literature have been made in connection with assessments of safety. Major disasters associated with catastrophic inrushes of water to deep mines began to occur with increasing frequency from the early 19th Century onwards. For instance, in 1815, miners in Heaton Colliery (Newcastle Upon Tyne) broke into abandoned, flooded workings of the nearby Jesmond Colliery. Altogether, this inrush cost 90 lives, with most being killed by starvation and asphyxiation, after being trapped above the water in isolated workings (Doyle, 1997). In 1837, 27 miners at Workington (Cumbria) were killed when extraction of a pillar in under-sea workings caused fracture propagation through 50m of cover to the seabed. The sudden inrush of sea water to the mine caused a visible whirlpool on the surface of the sea, and the mine was entirely flooded within minutes (Saul, 1959; Duckham and Duckham, 1973; Cook, 1982). North Levant tin mine in Cornwall was flooded in April 1867, with the loss of five lives, when adit clearance operations unexpectedly released a large volume of water from the nearby abandoned workings of Wheal Maitland (Vivian, 1990). In 1877, encroachment of the workings of Tynewydd Colliery (Rhondda Valley, South Wales) on nearby flooded workings caused an inrush which immediately claimed 4 lives. More remarkably, 10 miners survived for up to 7 days below the water table, in air pockets trapped in isolated up-dip workings. One of these men was sadly killed at the moment of rescue, when the release of compressed air sucked him into the hole dug from above by his comrades. The remaining 9 men survived, although all suffered from the bends following the rapid decompression (Llewellyn, 1992). This case illustrates just how lowly permeable unworked Coal Measures strata can be, and demonstrates why highly compressed mine gas is sometimes encountered in isolated pockets when drilling into old voids below the water table.

Despite the implementation of rigorous precautions against inrushes, water-related disasters continued into the 20th Century. Thirty-eight miners were drowned at South Elswick Colliery, Newcastle, in 1925 when water suddenly burst in from nearby flooded workings. Even after the formation of the National Coal Board (NCB) in 1948, catastrophic inrushes were occasionally encountered. In 1950, miners working up-dip in Knockshinnoch Castle Colliery, Ayrshire, narrowly escaped death when they holed into an overlying peat bog, which promptly liquefied and flowed into the mine (Duckham and Duckham, 1973). The most recent, major, water-related mine disaster occurred at Lofthouse Colliery, West Yorkshire, on 21st March 1973 (Calder, 1973). On that occasion, an advancing longwall face in the Flockton Thin Seam was suddenly flooded when it holed into an ancient, flooded shaft. Although the existence of this shaft was known, it was not thought that it had been sunk as deep as the modern workings. It was later discovered that the British Geological Survey held records which indicated that the flooded shaft had been sunk to the Flockton Thin Seam, but that these records had not been consulted by the NCB surveyors. Seven miners lost their lives at Lofthouse, despite prolonged rescue attempts inspired by the forlorn hope that at least some men might have been trapped alive in air pockets, as had happened at Tynewydd nearly a century earlier. This sadly proved not to be the case.

Lofthouse has left a major scar on the collective memory of the modern mining industry. Hindsight suggested that a change in the quality of the mine water encountered at the face could have provided advance warning of the disaster, had attention been paid to detail. Consequently, in the decades which followed, the NCB implemented a massive programme of mine water sampling in all coalfields with the purpose of detecting any such warning signs in future. The resulting mine water chemistry database has yet to be released publicly for hydrogeological use, but undoubtedly represents a major scientific resource.

With such a consciousness of potential disaster at large in the mining industry, it is not surprising that long and hard thought was put into the prospects for development of major undersea coal reserves off the coast of Northumberland and Durham between 1950 and 1990. The fruits of these deliberations are recorded in a number of remarkable papers, which still represent the most coherent body of knowledge on coal mining hydrogeology anywhere in the world. Some of the main findings of this research are summarised below, with particular attention being paid to aspects which are relevant to present-day environmental management needs.

2.4.2. Hydrogeological observations in working collieries

The earliest attempts to systematically investigate groundwater ingress to active workings were undertaken in the Durham and Yorkshire coalfields, and are documented in a series of classic papers by Saul (1936, 1948, 1949, 1959, 1970). These papers have positively influenced the thinking of several generations of mining engineers to the present day. Saul's papers describe the major factors governing "normal inflows" (as opposed to catastrophic inrushes) to deep coal mines in the UK. Four major conclusions of his work are most pertinent here:

1. In previously-unmined Carboniferous Coal Measures, the main sources of water to a new deep mine will be the major sandstones in the sequence; the future water make of a new deep mine can be expected to amount to between 20% and 40% of the mean annual rainfall landing on the outcrops of the sandstones in the local sequence (Saul, 1948).

2. Water makes its final entry into mine workings in a highly localised manner, appearing either as "feeders" (like underground springs) or as "drippers" (resembling 'rainfall' from a small area of the roof). Inclined boreholes driven upwards from headings into virgin ground, or backwards into goaf, can usefully

collect numerous "drippers" into a single flow, which will gradually diminish in rate as the goaf compacts under its own weight (Saul, 1970).

3. Mined ground is not so much a porous medium as a network of interconnected "breaks" (ie vertical water-bearing fractures, mostly corresponding to dip-parallel faults, and "laterals", i.e. beds of sandstone or worked seams); the discrete nature of most inflows to mines noted above is explicable in terms of the eventual interception of these discrete features by mine headings and panels (Saul, 1948, 1949).

4. In the absence of adjoining shallow workings or a steeply-dipping permeable sandstone in the sequence, new mine voids deeper than about 140m (or more than 140m below the sea bed, or the base of overlying aquifers as appropriate) are unlikely to encounter major feeders (Saul, 1948), save where faults provide short-circuiting connections to higher horizons (Saul, 1970).

These four observations represent a distillation of many decades of experience in some of the world's longest- and most thoroughly-worked coal sequences. Given the remarkable comparability of Carboniferous and older coal-bearing sequences world-wide (at least in terms of basic lithostratigraphy, there seems no reason to doubt that Saul's insights are applicable elsewhere in the world. This supposition is bolstered by the data given in Table 2.1, which presents a compilation of "normal" inflow rates from deep mines around the world, normalised to take the unit area underlain by mine-workings into account. Although the data available for inclusion in Table 2.1 are sparse, it does not seem too bold to conclude that:

(i) metalliferous mining fields are one to two orders of magnitude wetter than coalfields.

(ii) the coalfields show a remarkably low range of values, despite large differences in rainfall between the different localities. Indeed the narrowness of this range suggests that prediction of future dewatering requirements of new coal mines may be based with reasonable confidence on Table 2.1 alone.

The most obvious explanation of these observations is that many metalliferous deposits (and certainly the examples in Table 2.1) occur as vertical or sub-vertical veins, which reach the ground surface almost directly above active workings. By contrast, most deep-mined coalfields have shallow structural dips, so that the outcrop of a given seam will often be many kilometres from the active workings. Furthermore, immediately above coal workings, substantial thicknesses of roof strata (often including low permeability mudstones) will act as a barrier to infiltrating rain water. The narrow range of values for inflow rates given for the coalfields in Table 2.1 corroborates this deduction: Despite representing a wide range of climatic zones, the fact that the areally-referenced inflow rates are so similar suggests that it is the maximum permeability of overburden, rather than the rainfall rate, which is the prime limiting factor on normal inflow rates to coalfields. Indeed, even where the Coal Measures underlie major water bodies, such as an aquifer or the sea bed, so that availability of water at the ground surface will never be limited, normal inflow rates remain in the same narrow range.

Table 2.1 - Area-normalised "normal inflow" rates for selected deep-mining fields around the world.

Mining District name and location	Normal inflow rate in Ml/d/km^2	Source of data
Durham Coalfield, UK.	0.3	Younger (1993)
Dysart-Leven Coalfield, UK.	0.6	Younger et al (1995)
Jharia Coalfield, India	0.3	Gupta and Singh (1994)
Lower Silesian Coalfield, Czech Republic	0.2	Grmela and Tylcer (1997)
Nottinghamshire Coalfield, UK.	0.2	Dumpleton and Glover (1995)
Ruhr Coalfield, Germany	0.2	Coldewey and Semrau (1994)
Upper Silesian Coalfield, Poland	0.4	Różkowski and Różkowski (1994)
West Cornwall Tin Mines, UK	4 to 15	Author's own notes
North Pennine Orefield, UK	4 to 8	Author's own notes
Witwatersrand Goldfield, South Africa	60 to 90	Cook (1982)

The earliest detailed studies of water ingress into a mine in Coal Measures beneath a major aquifer were reported by Clarke (1962). Conceptually, it is reasonable to suppose that the magnitude of the "normal" inflow rates to coal mines will reduce markedly with increasing overburden thickness and increasing distance from the outcrop of the seam and/or its roof strata, or distance from the subcrop of the same against an overlying aquifer. Clarke (1962) sought to combine these two inflow controls (overburden thickness and distance to outcrop) into a quantitative rule, invoking Darcy's Law in justification. The fact that Clarke (1962) did serious violence to the meaning of Darcy's Law in the process has not hindered the widespread adoption of "Clarke's Rule", and propagation of his mis-use of the term "apparent hydraulic gradient", in mining engineering circles. In such circles, "apparent hydraulic gradient" is taken to signify "the head of water measured in the workings divided [by] the minimum distance to the point of replenishment by a free body of water" (Orchard, 1975). The "point of replenishment" equates to the outcrop (subaerial or submarine) of a water-bearing bed in the sequence above a worked seam, or to its subcrop "against unconformable water-bearing strata" (Orchard, 1975). The "apparent hydraulic gradient" thus defined is more a measure of structural dip than of the trend of any potentiometric surface. According to Clarke (1962), where the "apparent hydraulic gradient" exceeds about 1:5, then coal extraction will induce permanent feeders. Where this ratio is 1:7 or more, workings will generally remain dry.

It is only in the last three decades that attempts have been made to improve upon these empirical "rules of thumb" by means of process-based geotechnical research. Building upon the success of the NCB's "Subsidence Engineer's Handbook", Orchard (1975) sought to quantify the "safe working depth" for coal mines below major aquifers or the sea-bed. In terms of the zones of stratal deformation shown on Figure 2.5, the strategy for working beneath bodies of water is basically to ensure that Zone 2 lies considerably below the base of the aquifer or sea-bed. Specifically, mining should be undertaken such that the tensile strain at the base of an overlying water body does not exceed 10mm/m (Orchard, 1976; Singh and Atkins, 1983). More recently, Aston and Whittaker (1985) have demonstrated that for most undersea workings increased water ingress does not occur even where the tensile strain at the sea bed reaches 14 mm/m. Kesserü (1995) has also argued that a more rational approach to the minimisation of the risk of massive inflow would be to determine rock stress patterns and calculate the degree of safety they imply in relation to the available driving head in the water body. Nevertheless, the conservative 10mm/m tensile strain criterion has served well as the basis for mining regulations which stipulate that no working should be undertaken within 105m of the base of the overlying water body (Orchard, 1975). Workings adhering to such regulations have been documented as remaining virtually dry in the UK (Saul, 1970; Orchard, 1975; Aston and Whittaker, 1985), Canada, Australia, Japan, Chile (Singh and Atkins, 1983), Russia, China, Hungary and the USA (Kesserü, 1995). Indeed, on the basis of detailed statistical evaluations of maximum and long-term water yields of several hundred undersea longwall faces, Aston and Whittaker (1985) have concluded that wet conditions are most usually due to the presence of major faults which bring aquifers into contact with the roof strata of workings. Further examples of otherwise "safe" tensile strains inducing excessive water yields in the vicinity of faults are given by Singh (1986).

Where extraction is observed to induce increased inflows of water from above, it is common practice to attempt to draw Zone 2 closer to the mined void by narrowing the width of voids left unsupported. One option is to revert to the system of room-and-pillar, in which large pillars of intact mineral are left in place to support the roof indefinitely. However, this may result in mineral recovery rates which are too low to be economic. Consequently, in longwall mining situations, it is more common to reduce the width of panels from the typical 100 - 200m to "shortwall" widths of 50m or even less. There is a two-fold rationale for a switch to "shortwall":
(i) the tensile strain induced by subsidence is lessened, hence inducing less fracturing at the base of overlying source aquifers, and
(ii) the rate of retreat of the working face is sufficiently rapid that feeders will be "left behind" in the goaf, rather than encountered on the face (Orchard, 1975).

For instance, a switch to shortwall was successfully adopted in the Selby Coalfield, UK, after early 200m-wide longwall faces in Wistow Colliery induced substantial inflows from overlying Permian limestones. Peak flow rates were sometimes as high as 65 Ml/d, though feeders always declined over a few months to steady flows of only a few hundred cubic metres per day. After the width of faces was reduced to 60m ("shortwall"), no feeder greater than 13 Ml/d was ever encountered, and all feeders have also declined dramatically in flow rate, so that the total residual water ingress from the overlying limestones five years after mining commenced at Wistow was very low, at around 0.7 Ml/d.

Recourse to shortwall extraction in an attempt to reduce water ingress has also been described from undersea workings by Saul (1970), though in the case he describes, absolute water yields were not greatly reduced following the switch to shortwall; however, the rapid movement of the shortwall faces was of benefit to face workers as feeders were "left behind" in the goaf. Aston and Whittaker (1985) subsequently explained that the apparent failure of the shortwall strategy to reduce water inflow rates in the case described by Saul (1970) can be ascribed to the occurrence of a major fault which has since been discovered adjacent to the workings. For these inflows to have been avoided, the total tensile strain would have had to be minimised much closer to the workings than would have been predicted from the thickness of cover to the base of the overlying aquifer.

It is now accepted that new workings can proceed beneath flooded old workings as long as the principles outlined above are observed (eg Orchard, 1975; Singh and Atkins, 1983; Cain et al, 1994; Reddish et al, 1994). Nevertheless, where flooded old workings lie up-dip from the new mine, along the same horizon, then problems of water ingress can be much more difficult to avoid. Indeed, from the mid-Nineteenth Century onwards, numerous instances have been recorded in which the cessation of pumping in abandoned workings has led to the inundation of newer workings down-dip (e.g. Taylor, 1857; Saul, 1936; Coldewey and Semrau, 1994). These experiences prompted the development of regional, external dewatering schemes in many areas, commencing as early as 1857 (Taylor, 1857; Saul, 1936).

The earliest regional dewatering scheme to be implemented appears to have been that which operated in the South Staffordshire Coalfield, UK, from 1873 to 1920 (Saul, 1959). Pumping at rates of up to 65 Ml/d from old workings in each of four "pounds" (i.e. ponds in modern terminology; see the Glossary and Section 2.4.3 below) was maintained to facilitate active mining at depth. Subsequently, similar schemes have been implemented in several UK coalfields, including Fife (Younger et al, 1995a); Durham (Harrison et al, 1989; Sherwood and Younger, 1994; Younger, 1993; Younger, 1997a); South Yorkshire (Saul, 1936); Derbyshire (Peters, 1978); Nottinghamshire (Peters, 1978; Awberry, 1988; Lemon, 1991) and Leicestershire (Smith, 1996). In Germany, the deep mines of the Ruhr coalfield are dewatered regionally by a system of 28 pumping shafts, which has been progressively developed as deep mines have gradually closed since 1920 (Coldewey and Semrau, 1994). Similar systems have also been used for many decades in Hungary (Kesserü, 1997) and Poland (Rogoz, 1994; Rózkowski and Rózkowski, 1994).

2.4.3. Mine water rebound, ponds and flooded workings.

As mentioned in Section 1.3.1, "mine water rebound" is the most common name for the process whereby formerly dewatered mine voids gradually fill with water until a surface overflow point (or a decant point into an overlying aquifer) is encountered. There are some critical differences between the mode of hydrogeology encountered in working mines (described in Section 2.4.2) and the hydrogeological behaviour of mined systems during rebound. These differences are largely attributable to one fact: in working mines, the huge voids comprising the mine itself are kept dry by strategic pumping, whereas during rebound, the mine voids themselves become the principal conduits for water movement. The hydrogeologist who wishes to predict mine water rebound, or to make a physical interpretation of monitoring data arising during rebound, needs to understand the profundity of this difference. During working, the contrast in permeability between intact strata and near-void strata affected by subsidence (Figure 2.5) is a matter of major interest. The contrast in that case is on the order of three to four orders of magnitude in terms of hydraulic conductivity (e.g. compare Zone 2 with Zone 1 in figure 2.5). The contrast between near-void strata and the voids themselves spans no fewer orders of magnitude, albeit open voids are so "permeable" they fall outwith the bounds of Darcian classification. If we go on to contrast intact strata with open mine voids, the contrast in terms of hydraulic conductivity is in excess of seven orders of magnitude.

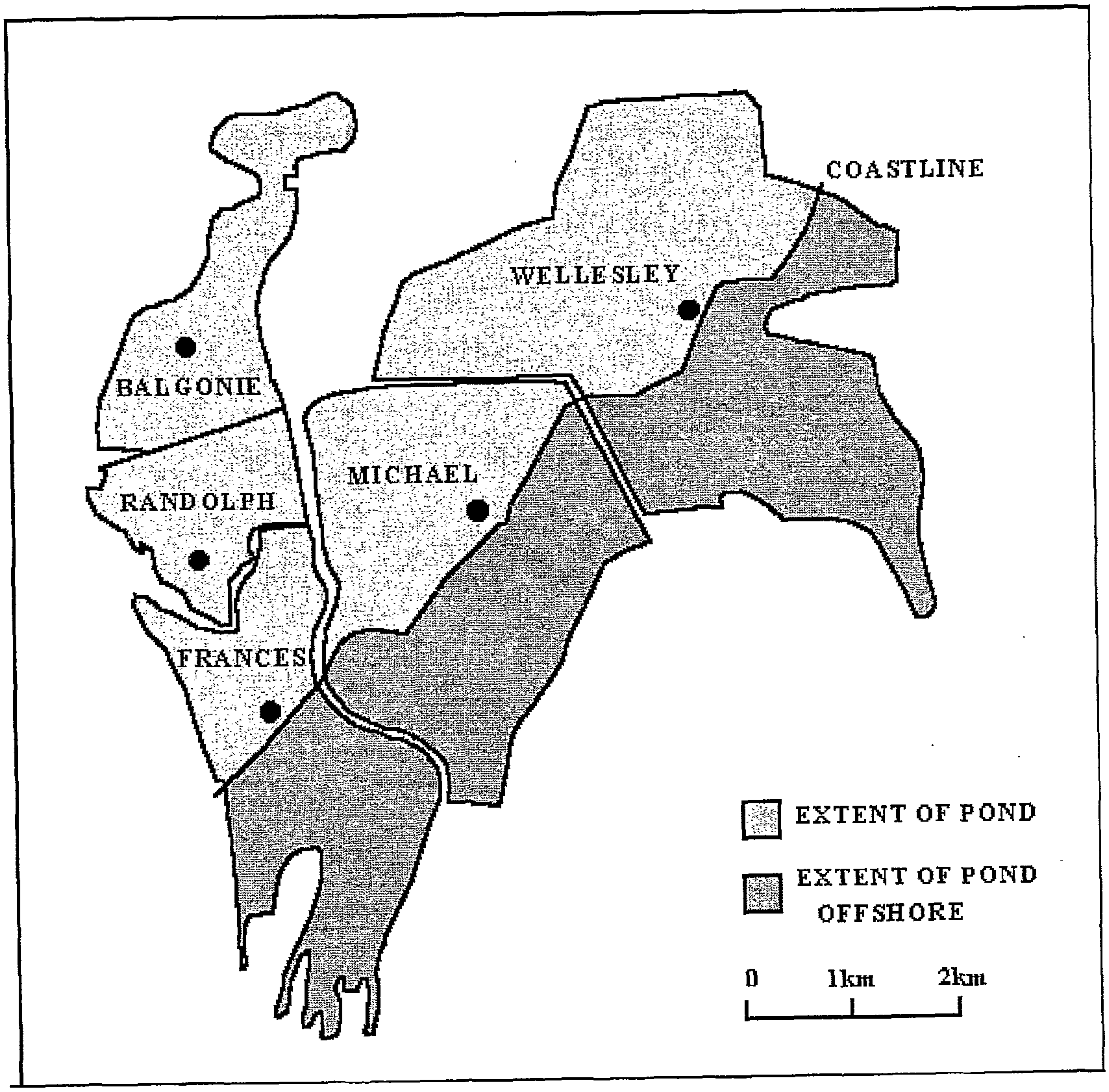
Recognition of this marked contrast is nothing new. Indeed, it has long been implicit in the actions of surveyors and engineers in the UK coal industry, particularly in the manner in which they conceptualise volumes of interconnected workings as 'ponds'[3] separated by barriers of unworked coal (Minett, 1987). Inherent in the definition of a pond is the concept that the mine-workings within any one pond are extensively inter-connected (on multiple levels, if working occurred on multiple levels) so that water rising within any one pond will display a common level throughout that pond. At certain elevations, adjoining ponds may be connected via discrete "overflow points". Typical features forming inter-pond overflow points include:

- roadways (which were often driven between formerly separate mines during war time, to ensure security of egress in the event of bombing)
- an area in which two adjoining goaf panels coalesce
- old exploration boreholes
- permeable geological features (e.g. the margins of a basaltic dyke, a limestone bed, or an open fault).

Figure 2.6 shows the distribution of five major ponds in one UK coalfield. Each of the major ponds encompasses the workings of numerous collieries, but each corresponds more-or-less to the "take" of the last major colliery to be worked in its vicinity. This compartmentalisation of coalfields has its origins in the system of private ownership of mineral rights and/or mining leases, which dominated mining economics until the middle of the 20th Century. Even under state ownership in the latter half of the 20th Century, division of responsibilities between different management teams within the NCB and successor organisations favoured the maintenance of largely separate "takes" in most coalfields. Further examples of ponds in major English coalfields are presented in Chapter 4.

[3] The term 'pond' can be replaced by regional variants: e.g. in Fife, a pond is generally termed a 'basin', in Staffordshire a 'pound' and in the USA a 'mine pool'.

Figure 2.6. Ponds in the Dysart-Leven Coalfield of eastern Fife, Scotland (after Sherwood, 1997).

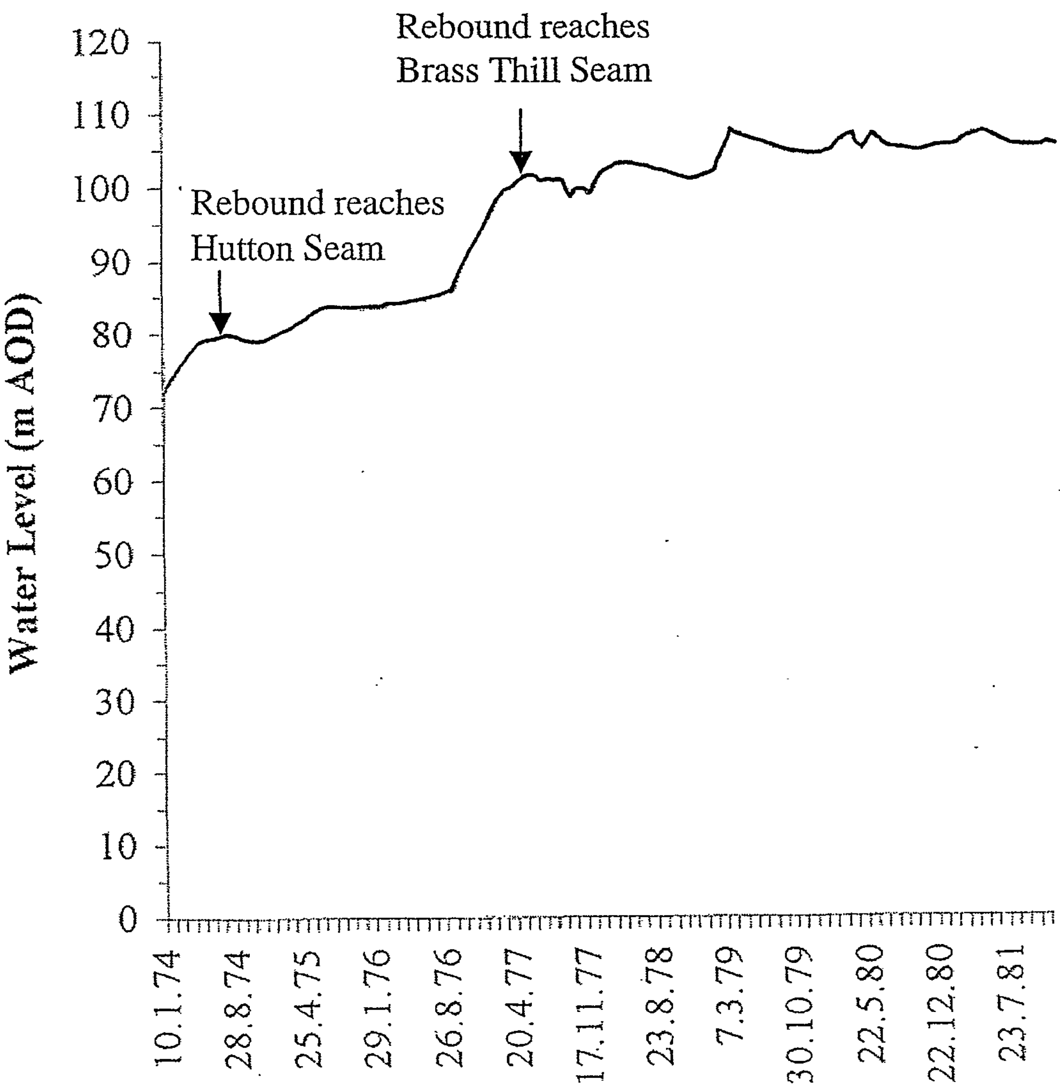


Implicit in the recognition of ponds is an expectation that the process of rebound will occur independently in two (or more) adjoining ponds until such time as the water level in one or more of the ponds reaches an overflow point. Inter-pond transfers of water will then occur until the difference in head between the two ponds either side of each overflow point is minimised. If the overflow feature is "unrestricted" (i.e. offers very little resistance to inter-pond flow, e.g. a large-diameter roadway), then the "minimised" head difference between adjoining ponds will tend to zero. Where the overflow feature is "throttled" (i.e. *does* resist flow to some degree, which might be the case if the feature in question is a body of goaf, a narrow borehole, or a natural geological feature), the "minimised" head difference between the two ponds may amount to a difference of several metres, especially if the ponds differ in recharge area. To some degree, we can judge whether the overflow feature from one pond to another is unrestricted or throttled if we have a record of seasonal water levels in the pond with higher water level. A pond with an unrestricted overflow will respond so rapidly to recharge that water level fluctuations will be minimal (<< 0.5m), whereas seasonal water level fluctuations in a pond with a throttled overflow may be quite marked (> 1m).

Water level rise during the rebound period in any one pond is a function of only two features: the total recharge to the pond (i.e. the sum of rain-fed recharge and any head-dependent inflows from adjoining aquifers and/or other ponds), and the distribution of storage capacity within the pond. The distribution of storage capacity is closely related to the presence of mine voids and the zones of disturbed strata associated with them. As implied in Figure 2.5, for typical longwall panel widths in the range 100m to 250m worked in UK collieries in the 20^{th} Century, strata can be expected be disturbed to an elevation of 40 – 100m above each worked seam. The processes of goaf formation, bed separation and extensional fracturing all serve to impart a total porosity of a few percent to Coal Measures strata which were previously very "tight". The rate of water level rise in a pond can therefore be expected to be relatively rapid as inter-seam intervals of intact strata are traversed, but relatively slow as the water surface rises through worked horizons and associated disturbed roof strata. This is indeed what is commonly found in practice: Figure 2.7 gives the example of the Ladysmith shaft in southwest County Durham, showing the stepped nature of the water level recovery, with flattened portions of the curve corresponding to the periods when the named seams and their roof strata were flooding.

In the early stages of mine water rebound, when inter-pond head differences are high, flow through the large, open mine voids, can be confidently predicted to be turbulent. Using the analysis of the hydrogeological behaviour of karst conduits developed by Smith *et al* (1976), it is clear that flow in most open mine voids *must* be turbulent whenever the flow velocity exceeds one millimetre per second. Only under the most quiescent of post-rebound conditions, therefore, would laminar flow be expected in the mine voids themselves. This theoretical deduction is amply borne out in practice.

Figure 2.7. Annotated mine water rebound record for the Ladysmith Shaft (NZ 194255) in southwest County Durham in the late 1970s, illustrating the stepped nature of the recovery curve, reflecting the presence of worked seams and disturbed roof strata at discrete depth intervals.



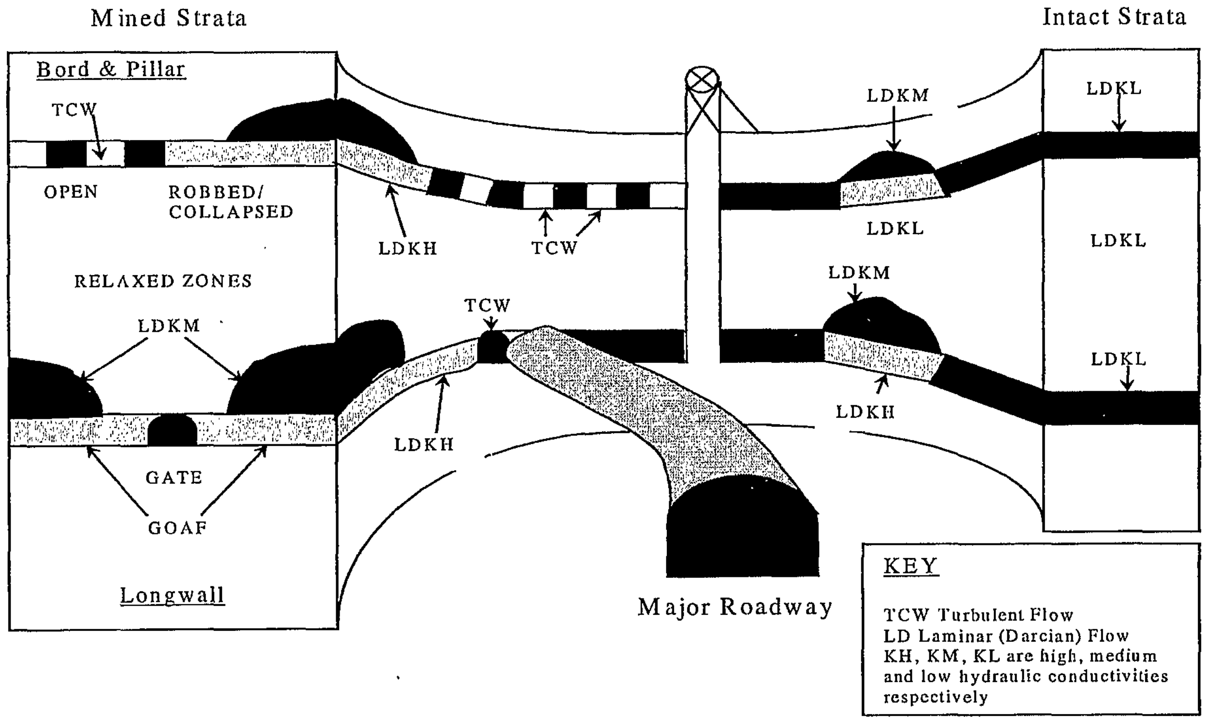
For instance, in a series of tracer experiments in the Forest of Dean coalfield, Aldous and Smart (1988) found that roadways above the regional water table transmit water at velocities of up to 16 km/d, while roadways below the water table (which are consequently subject to lower hydraulic gradients) display velocities approaching 0.5 km/d. These lower velocities still imply turbulent flow. Transitional flow conditions (i.e. between turbulent and laminar) are implied by velocities in the range of 3 to 20 m/d, obtained by Aljoe and Hawkins (1994) from flooded roadways subject to an immeasurably low hydraulic gradient in an abandoned mine in western Pennsylvania, USA. Turbulent flow of water in open roadways and other mine voids is also evidenced by the elevated suspended solids concentrations of some mine water discharges, which imply subsurface velocities well in excess of 1mm/s to account for entrainment of sediment. In unsaturated voids, erosion by rapidly flowing mine water has been known to prevent the blockage of open roadways which would otherwise have been sealed by vigorous floor heave (P Aldous, personal communication, 1995).

To conclude, flow in abandoned deep mines, particularly during rebound, seems to occur by a combination of:

- laminar, Darcian flow in the subsided country rock and goaf, and
- turbulent flow in open roadways and mine voids

The spatial distribution of these differing flow regimes in a mined area are illustrated schematically on Figure 2.8. In the following Chapters we will show how this conceptualisation of mined ground underpins analytical methods for analysing mine water systems.

Figure 2.8. Distribution of zones of potentially turbulent and laminar flow in an abandoned mine (after Younger, 1995d).



Generalised Conceptual Hydrogeological Model for an Abandoned Coalfield

3 PREDICTING REBOUND: A. MANUAL METHODS

3.1. Back of an Envelope?

All hydrogeological analyses must start somewhere, and a piece of paper with approximate dimensions of 22cm x 11cm is as good a place as any. For the first step in analysis is the definition of an adequate *conceptual model* for the system to be studied. The term "conceptual model" has a formal definition in hydrogeological modelling circles, one version of which is as follows:

> A conceptual model is an assemblage of simplifying assumptions about a complex, real system, which achieves a valid representation of that system, including all major features, whilst avoiding unnecessary detail.

A piece of paper of the size mentioned is often just about large enough to accommodate the list of major "assumptions" which are the bones of a conceptual model. Justifications for those assumptions may run to several pages and plan sheets.

In an "ordinary" hydrogeological modelling study, typical assumptions may include statements on whether the aquifer can be regarded as homogeneous and / or isotropic, what the aquifer boundaries are and how they might best be represented (fixed head, specified flux etc). In assembling a hydrogeological model of a mine water system, the suite of possible assumptions is somewhat broader, taking into account the peculiarities of mined systems as outlined in Chapter Two.

It is the purpose of this chapter to provide brief guidelines on the construction of simple, conceptual models for abandoned coalfields (a task which is best done manually), and then to demonstrate some manual calculation / graphical construction techniques which may (in certain circumstances) provide a rebound prediction to a first approximation. The following two chapters provide details of computer-based analyses which may be used to obtain more exhaustive rebound predictions. These computer-based techniques can only be applied if supported by at least some of the manual methods outlined in this Chapter, and a good understanding of the mine characteristics.

3.2. Defining System Extent and Lateral Boundaries

The first task in a mine water rebound analysis is definition of both the internal geometry and the external boundaries of the system to be modelled.

Because the contrast in permeability between mined and intact ground is so vast, the overall geometry of a mine system is usually well-defined (at least laterally) by the areal extent of the workings. Where the outermost workings adjoin intact coal and / or shale, the outer limit of workings may be conceptualised as a simple zero-flow boundary. In such a case, it remains only to source information on the areal extent of the mine workings.

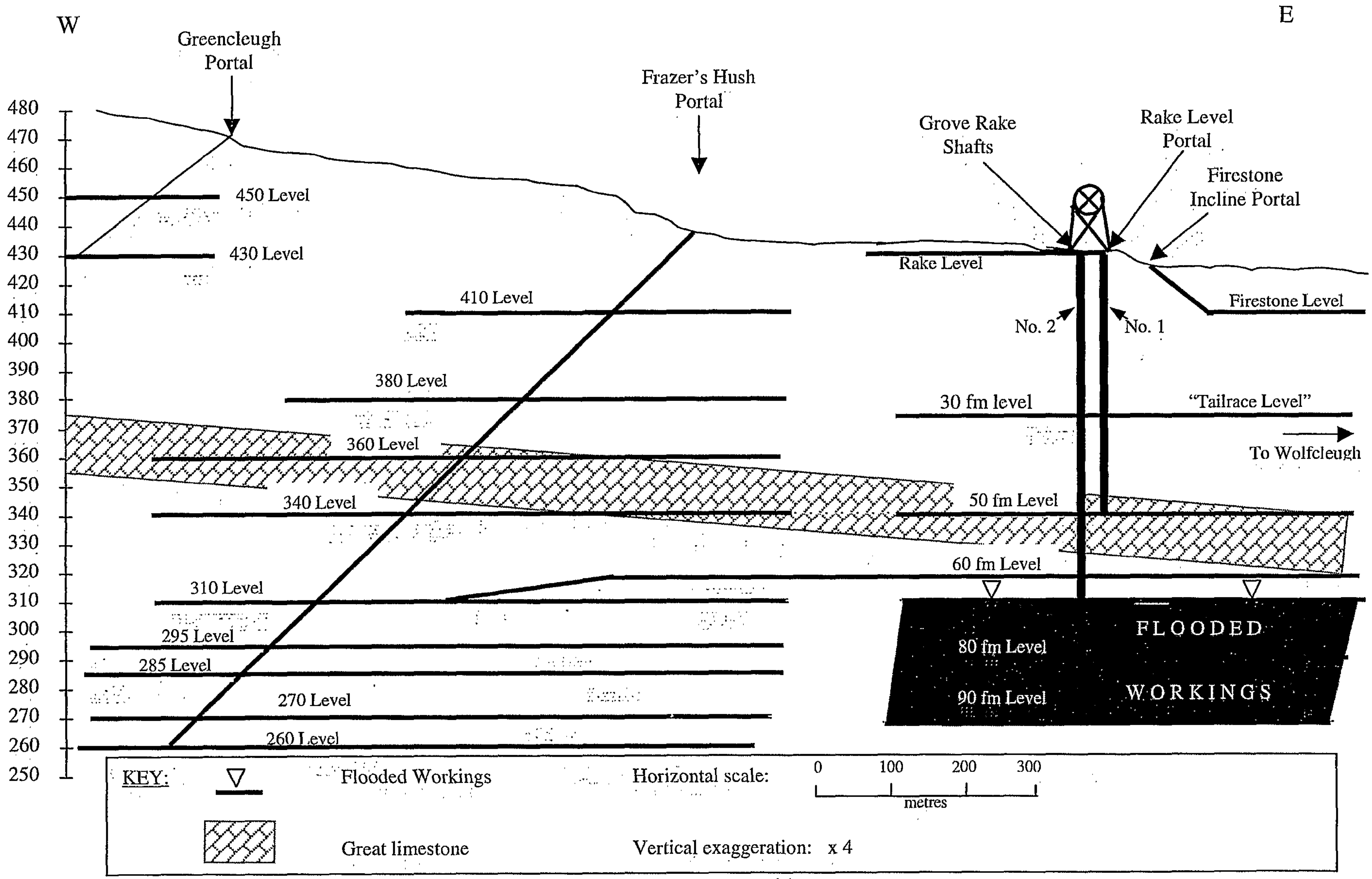
The areal extent of abandoned mine workings is comparatively well-documented in the UK. Mine abandonment plans have been produced and deposited systematically in accordance with UK law since 1872, so for any mine worked more recently than this (i.e. most of the larger mines), mine abandonment plans should be available. The abandonment plans themselves are typically at large scales (usually 1:2,500). However, 1:10,000 seam-by-seam summary sheets are available for most of the major UK coalfields. This smaller scale is particularly useful for hydrogeological purposes, though reference to the original 1:2500 abandonment plans will usually be necessary to resolve (or double-check) important matters of detail. Coal mine plans were all prepared in accordance with national guidelines (see NCB, 1977), and many of the same surveying conventions also found their way into metals mines as ex-NCB staff transferred their skills elsewhere. Before attempting to interpret mine plans for the first time, reference to these surveying practice guidelines (NCB, 1977) is strongly recommended. It is also advisable to secure the assistance of a former NCB surveyor when first scrutinising plans; it is generally not difficult to access such expertise, particularly if contact is made with the local minerals branch of the Royal Institution of Chartered Surveyors.

Coal mine abandonment plans are primarily held by the Coal Authority at Bretby, Nottinghamshire. For metal mines, the plans are usually held at County Record Offices, though some plans may also be preserved in the archives of mining companies, in mining museums (the Scottish Mining Museum at Newtongrange, for instance, has most Scottish abandonment plans), in BGS offices (though the stock is not always exhaustive) and in the libraries of learned institutions. Although not well-publicised, local offices of the government's Land and Mineral Valuation office usually also hold extensive collections of abandonment plans for both types of mine.

In addition, catalogues of coal abandonment plans are often available in the local history sections of libraries. In some of our work at larger scales (considering rebound over areas of 1000s of km^2) we have found that merely plotting the locations of formal abandonment plans onto a 1:25,000 base quickly yields a map showing the overall extent of workings in a given seam. Although the layout of major roadways will not be revealed by this method, the directions of such roadways can often be correctly inferred from the layout of the worked panels in relation to shafts and geological structure (as strike roadways were always more popular than dip roadways). Many BGS Memoirs also include summary maps showing areas of workings in the principal seams, at about the same resolution attainable by processing abandonment catalogues.

In certain mining fields, the hydrogeological boundaries of the system will not coincide simply with the outer perimeters of the workings. This will most commonly be the case where major, water-bearing sandstones or limestones of regional extent locally intersect deep mine workings. Such is the case, for instance, in parts of the South Yorkshire Coalfield, where the White Rock, the Parkgate Rock and other thick sandstones provide regional flow pathways for waters which eventually enter mine workings. An even more extreme case occurs in the North Pennine Orefield, where the last working deep mine (Frazer's Grove) intersects the Great Limestone, a 20m-thick bed of laterally persistent, variably karstified rock. More than two-thirds of the water make of Frazer's Grove was sourced from the Great Limestone (Younger, 1998b). Where water-bearing rocks extend outside of the main body of workings, much of the "water make" during working will be head-dependent, and is thus unlikely to contribute greatly to the post-closure water make of flooded abandoned workings where these drain to a topographic low point *above* the sub-crop of the water-bearing horizon within the mine. Figure 3.1 shows a vertical cross-section of Frazer's Grove mine.

Fig 3.1. Vertical Cross Section (West to east) of Frazer's Grove Metal mine



In setting up a model for a mined system with a major head-dependent component in its water make, it may be possible to specify the boundaries of the simulation domain to coincide with the outer limit of workings, but the boundary conditions themselves will need to be represented as head-dependent flux boundaries, rather than zero-flow boundaries. Only where a major water-yielding bed lies *above* the final surface overflow point from the workings will it be appropriate to specify a specified-head boundary. The general lack of cases in which specified-head boundaries may be used validly in modelling mine water systems poses a problem for the application of standard groundwater modelling packages to such systems. This is because all of the common numerical methods for solving the groundwater flow equations have a requirement that head be specified upon at least one point (and often more than one point, depending on grid shape) on the outer boundary.

3.3. Defining Initial and Final Conditions

3.3.1. Initial conditions

The "initial conditions" at the commencement of mine water rebound are defined by the water level in the workings at the time the pumps were switched off, and the final dewatering rate before the withdrawal of pumps began.

Where workings were already partly flooded before the final cessation of dewatering, a reliable value for the water level before the start of rebound may well be available. Where a mine was actively worked until a few weeks before the cessation of dewatering, then it may only be possible to estimate the initial water level by reference to the state of operations at the end of extraction. In many cases, this will mean that the effective "water level" in the workings corresponds to the elevation of the deepest part of the dewatered workings during the final phases of extraction. (Since many mines close precisely because their final reserves are concentrated in the deepest parts of the take (which are also the most expensive to work), the last workings are often also the deepest). Defining an initial water level in this way is fraught with uncertainties, because the details of local hydraulic gradients in strata around the deepest workings are not known. It is to be expected that the latest workings will have been provoking transient hydraulic responses in the surrounding strata. These transient phenomena are complex, and are never characterised in day-to-day mining operations. Hence the initial rate of fill of the deepest workings is difficult to predict in all but the most simple of mining situations. In practice, we have often found it best to derive an appropriate "initial water level" by trial-and-error during rebound simulations.

The final dewatering rate before commencement of rebound is an important figure, as it indicates the total water make of the workings up to that point. This water make will include a component of head-dependent inflow, the estimate of which is a necessary part of rebound analysis. Estimation is aided if an upper-bound can be fixed by knowledge of the patterns of dewatering in the final months before closure. In an ideal situation, a record of dewatering rates (at, say, weekly resolution) for the last few years of working will be available. In most cases, the mine manager will have only an approximate idea of the quantities of water pumped. The new regulations governing the abandonment of mines stipulate that the mine operator will provide summary information on dewatering rates prior to abandonment. Care should be exercised in interpreting such figures, for in most cases the total dewatering rate of the mine will apparently be declining for several weeks (if not months) before final abandonment, as salvage teams gradually withdraw local pumps from remote districts of the mine underground, and mine water begins to fill localised pockets of storage in the mine instead of making its way to the main pumping lodge.

3.3.2. Final conditions

By "final conditions" we mean the equilibrium water level and discharge rate of the mine system after rebound is complete.

The equilibrium water level in the system post-rebound will be defined by the position and elevation of "decant points", via which the water will flow either to a surface water course or to an overlying aquifer.

After studying several hundred mine water discharges throughout Britain, we have gained the impression that the overwhelming majority of decant points to surface water are man-made features. The most common features forming surface decants are old adits ("drifts"). Probably second in frequency are overflowing shafts, with old exploration / ventilation boreholes accounting for a minority. Mine water discharge through un-mined outcrop areas is extremely rare; indeed, we can think of no unequivocal example. That is to say, while discharges are very commonly associated with seam / lode outcrops, the actual point of emergence is almost invariably via an old mine entrance on that outcrop. In undertaking rebound analyses, therefore, the prediction of surface decant points is normally pursued as follows:

- determine the outcrop patterns of the major worked horizons (use BGS mapping if possible)
- scrutinise archival records to ascertain the locations of old mine entrances (adits, shafts etc) on the outcrop area
- determine which of the old entrances lies in the lowest topographic position on the outcrop; this will be the most probable decant point, and should be subjected to field survey, including trial-pitting to determine its condition if possible. (NB in the case of coal mines, no such works should be undertaken without prior discussion with the Coal Authority)
- determine the distribution of other old mine entrances which lie higher than the most probable decant point, for if roof falls etc prevent the most probable decant point from flowing, then higher decant points may become active.

In the case of flow to an overlying aquifer, the final water level attained in the mine system will depend also on the permeability of the decant route (which is probably immeasurable, unless it is a shaft or borehole) and the head in the overlying aquifer. Only one unequivocal case of mine water rebound into an overlying aquifer has been documented in the UK to date. This occurred in the late 1970s when water in the vicinity of Mainsforth Colliery, County Durham, rebounded into the overlying Magnesian Limestone aquifer. The driving head in the mine workings (which are fed by recharge in a hilly area to the west) began to exceed the head in the Magnesian Limestone around 1975, and eventually the head in the Magnesian Limestone itself rose by around 10m, as the leakage it used to supply to the underlying mines was rejected, and upflow from the mines augmented recharge (Younger, 1995c).

The decant routes from the mine to the overlying aquifer at Mainsforth were provided by numerous subsidence-related fractures which had been propagated upwards during working (Clarke, 1962; Cairney and Frost, 1975). Indeed, Mainsforth was one of the main sites where the NCB learned (by trial-and-error) how to adapt mining practices to allow dry working below aquifers (Clarke, 1962). Consequently, the fact that Mainsforth made a lot of water from the overlying Magnesian Limestone was well-known (Cairney and Frost, 1975) and the existence of decant routes in the opposite direction was therefore undoubted. The only issue at the time of abandonment was whether the rebound in the mined system would ever generate sufficient head to cause upward flow. Given that the outcrop of the deepest worked seam (the Brockwell Seam) lies at about 125m AOD a few km to the west of the Magnesian Limestone scarp (which lies at around 90m AOD in this area), the potential for development of a driving head should have been clear.

These deliberations in relation to Mainsforth illustrate a general point of importance when assessing mine water rebound below major aquifers: upward flow into an aquifer requires not only decant points, but a driving head, which will largely be topographically controlled. While this general point is easy to make, practical application to specific cases may not be so clear as in the case of Mainsforth.

The equilibrium discharge rate from a mined system after rebound is complete can be estimated with reasonable accuracy if the head-dependent proportion of the total water make is known. Thus in the case of Frazer's Grove Mine, where the head-dependent inflow from the Great Limestone is determined on hydrochemical grounds as amounting to 1.2 Ml/d out of a total water make of 1.9 Ml/d, the net post-rebound discharge (to the Great Limestone aquifer) is estimated to be on the order of 0.7 Ml/d (Younger, 1998b). Similarly, head-dependent inflows of as much as 60% of the total water make have been estimated for South Crofty Mine, Cornwall, which would leave a balance of around 2.8 Ml/d to emerge at the surface, via the Dolcoath Deep Adit (Knight Piésold and Partners, 1997). Coastal collieries are another important example where the total water make during mining is clearly greater than the total amount of flow which will be available for discharge after rebound is complete. For instance, at the Michael and Frances Collieries in Fife, Scotland, Younger *et al* (1995) showed that approximately 45% of the total water make was sourced offshore, leaving only 18 Ml/d sourced by terrestrial recharge, and therefore potentially available to contribute to post-rebound discharge. In terms of the larger inland coalfields, the water make of any one mine may well be head-dependent to some degree, particularly if inflows occur from nearby flooded workings or major sandstones. However, if the coalfield as a whole is considered, the overall proportion of the dewatering rate which is head-dependent may be very small, since the major sandstones and all flooded workings (which are themselves rain-fed) are contained within the overall Coal Measures outcrop boundary. This is why estimates for the post-rebound discharge rates for entire coalfields can be equated with the total rain-fed recharge rate during maximum drawdown. Recharge estimates for coalfields are discussed in Section 3.5 below.

3.4. Defining "Ponds" and the General Sequence of Rebound

The concept of "ponds" in relation to mine water rebound was outlined in Section 2.4.3. From the definition given in that section, it should be obvious that the practical recognition / designation of ponds involves consideration of the available mining information with the purpose of identifying distinct volumes of workings which are heavily interconnected internally, but which have relatively few inter-connections to nearby ponds. Where seam summary sheets (1:10,000) or mine abandonment plans (1:2500) are readily available, the procedure of identifying ponds is relatively straightforward, if somewhat time-consuming. The following procedure is advocated:

1. Assemble all of the plans concerning the lowest-worked seam in the area of concern. Trace (or digitise) all areas where the workings are isolated.

2. Now examine the plans for the next seam up-sequence, and see if the areas of isolated workings identified in the plan for the lowest seam are also isolated at this horizon. If they remain isolated at this horizon, or if new areas of isolated working are identified at this horizon, then one or more candidate pond(s) have been identified. Check all candidate ponds for isolation by examining the largest-scale plans available for major roadways connections and shafts inter-connecting to higher or lower seams. If discrete connections are identified, note their nature (e.g. roadway, shaft, borehole, goaf overlap etc) and note the elevation of the highest point along the connecting route. (Note that elevations shown on pre-1970s ex-NCB plans are in feet above the old mining datum (which was fixed such that "zero" lay at 10,000 feet below Ordnance Datum). Plans prepared post- 1971 show elevations relative to Ordnance Datum).

3. Repeat step 2 for all higher seams, until:

   - all distinct ponds are delineated, and
   - the elevations at which adjoining ponds are joined by overflows are catalogued.

   The supposition is that once water levels have exceeded the elevations of major, open, overflow features, then further water level rises in adjoining ponds will occur in unison.

4. Check all interpretations for consistency with geological structure by constructing suitably scaled cross-sections in two or three directions across the study area, so that the effects of folding and faulting on the relative positions of seams in the different ponds are clearly understood, and taken into account when interpreting overflow points.

Although the above 4-step process is written with specific reference to coalfields, the same logic can be applied to metals mines, though the analysis will generally involve scrutiny of long-sections of individual veins.

To illustrate how ponds are identified, and how their influence on the rebound process varies over time, consider Figures 3.2 and 3.3. These figures illustrate the identity of several ponds in the north-eastern part of the Durham Coalfield, in relation to worked areas in the principal worked seams. It is evident from examination of these two figures that the water bodies associated with individual ponds will be rather limited in areal extent as long as water levels are restricted to the two lowest seams (the Busty and the Harvey). By the time the time water levels reach the Hutton Seam, all but two of the initial ponds will have merged into a single hydraulic unit. At the level of the Maudlin Seam (which was the most extensively worked seam in this area) none of the original ponds retain their definition.

Figure 3.4 illustrates the general sequence of rebound which would be anticipated in this area, shown in the form of a schematic rebound hydrograph[1]. This shows that water level rise will

[1] n.b: the time scale is arbitrary at this stage of analysis

be characterised by periods of slow recovery (during the periods when the seams and their associated, fractured roof strata are flooding), punctuated by periods of more rapid rise (as the tight, inter-seam strata are flooded). This pattern is common in many multi-seam coalfields.

To turn Figure 3.4 into a quantitative prediction of rebound, the arbitrary time-scale on the x-axis must be replaced with a real chronological scale. To do this, we need to estimate the availability of water to the flooding system, which in turn demands the estimation of recharge.

3.5. Estimating Recharge Rates for Mining Fields

Recharge estimation is a notoriously difficult task in all branches of hydrogeology (Lerner *et al*, 1990). Upon first consideration, therefore, the call to estimate recharge in such a complex hydrogeological environment as deep mining fields might seem quixotic. However, recharge estimation in the mining environment benefits from an almost unique advantage: the common availability of long records of steady-state dewatering rates. Where the water make of a particular mine system had few head-dependent components, the total dewatering rate during peak production provides a useful upper-bound estimate for the rate of recharge to the workings. Even where the head-dependent component of the water make was substantial, the total dewatering rate provides one of the two terms needed to calculate the long-term recharge rate. If some reliable estimate of the amount of head-dependent inflow can be made (which may well be possible, on piezometric and / or geochemical grounds), then the irreducible recharge rate (ie the rate which will also apply post-rebound) simply equals the total dewatering rate minus the head-dependent inflow component.

Where dewatering records are not available, and / or the proportion of head-dependent cannot be determined *a priori*, independent means of estimating recharge to mine workings are required. At their simplest, these can be obtained by measuring the area underlain by workings (in km^2), and then multiplying this figure by 0.3 (the mean of the coalfield water make figures quoted in Table 2.1), to obtain an estimate of the head-independent recharge in Ml/d. For instance, if a mine system underlies 12 km^2, one would expect it to have a water make on the order of 3.6 Ml/d. (It is only likely to exceed this figure if there is an unusually high head-dependent influx from an adjoining aquifer).

Figure 3.2. Ponds of the north-eastern Durham Coalfield, illustrating how separate ponds merge as the water floods higher, more widely-worked seams.

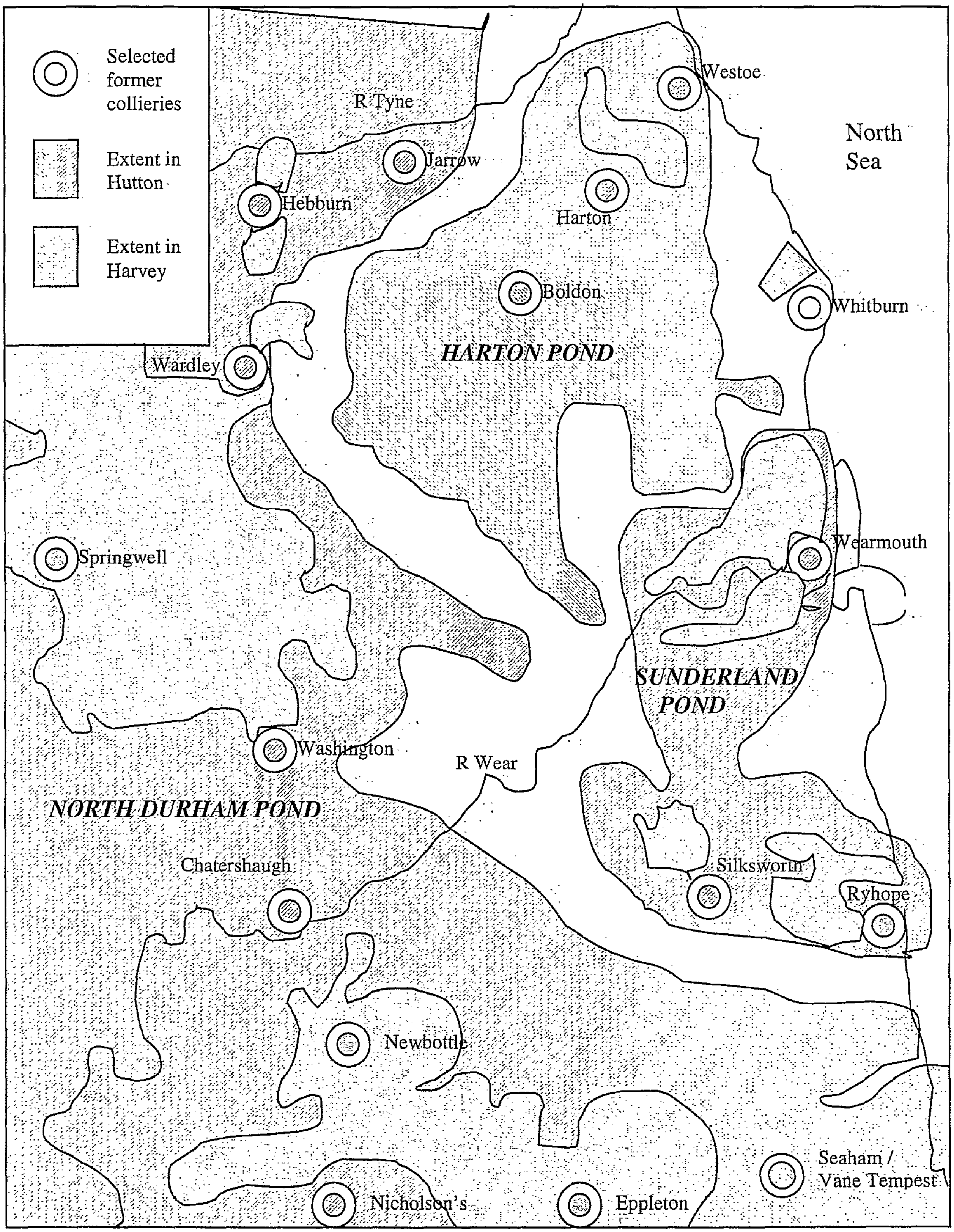


Figure 3.3. Schematic west-to-east cross-section across the area shown in Figure 3.2, illustrating the general vertical layout of pond areas. Faults, and minor workings in seams above the Maudlin, are omitted. Seams are drawn dashed where heavily worked, solid where intact. (Not to true scale).

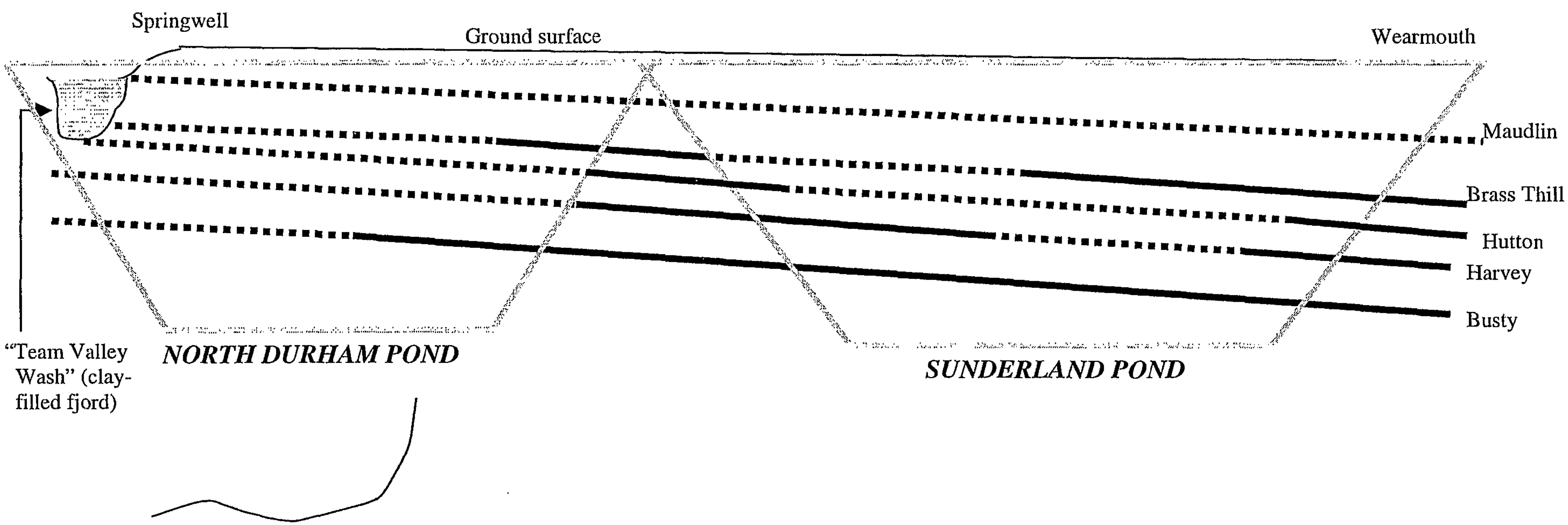
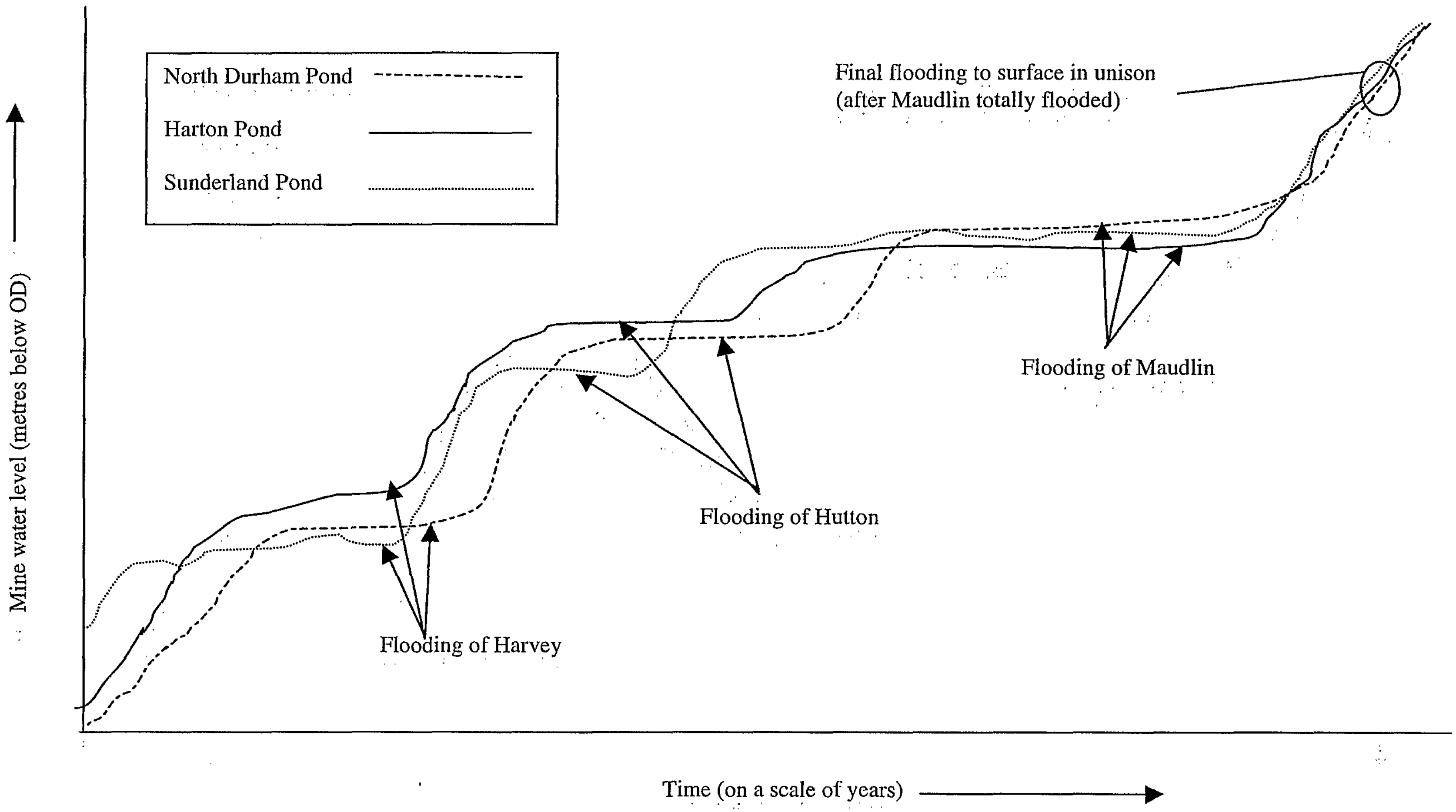


Figure 3.4. Schematic representation of the anticipated patterns of rebound in the system of ponds illustrated on Figures 3.2 and 3.3.



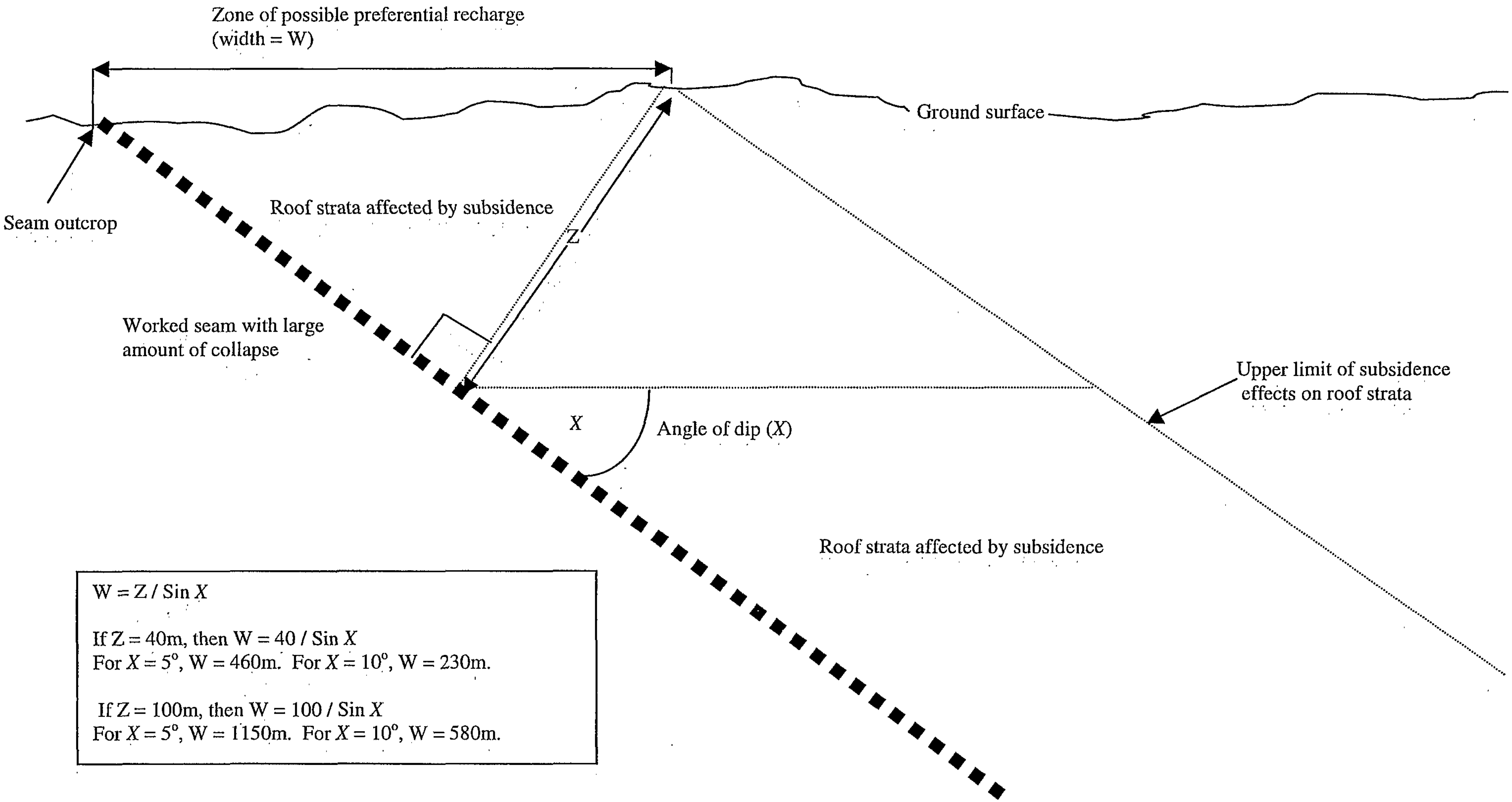
A rather more elegant means of determining the rain-fed recharge to a system of mine-workings is to perform classic soil moisture budgeting calculations, from which a time-series of "potential recharge" may be obtained. By "potential recharge" we mean the difference between the effective rainfall (= total rainfall minus actual evapotranspiration) in a given period and any soil moisture deficit which has to be satisfied in that period. This "potential recharge" is available to percolate to the water table, as long as the unsaturated zone is sufficiently permeable to allow this. In most areas of un-mined Coal Measures strata, the chances of significant volumes of recharge penetrating to more than 140m depth is vanishingly small (Saul, 1948; see Section 2.4.2).

Where thick sandstones are locally present in the Coal Measures, recharge may penetrate to depth, leading to the observation of Saul (1948) that the water make of a deep mine can be expected to amount to between 20% and 40% of the mean annual rainfall landing on the outcrops of the sandstones in the local sequence. The same probably also applies to the case in which there are extensive shallow workings above deep active workings. In the absence of thick drift, preferential recharge to the Coal Measures may be expected in those areas where the zone of collapse above old workings extends to the ground surface. Figure 2.5 and its associated text provides guidance on how far stratal disturbance can be expected to extend above a collapsed void. In essence, disturbance of one sort or another might be anticipated for a vertical distance equal to about 0.55 times the maximum span width of the collapsing void. For instance, above a worked panel 200m wide, strata can be expected to display altered permeabilities up to 110m above the original seam roof.

As most major seams were worked to exhaustion near to outcrop long ago, and the shallow workings have frequently collapsed (wholly or in part), then "zones of possible preferential recharge" are likely to be present on the up-sequence margins of the outcrops of most major seams. In these zones, the "potential recharge" can be regarded as being converted entirely to "actual recharge". Outside of these zones, varying proportions will be diverted laterally by low-permeability strata to form surface runoff.

Although the width of individual collapsed panels is not likely to be known, old workings will generally generate roof-stratal disturbance up to a height (Z) which may attain between 40m and 110m above the seam. If the dip of the seam under study is known (as is likely, given that dip values are often shown on BGS maps) or can be calculated from the outcrop pattern, then it is possible to map the zone of possible preferential recharge associated with each seam. Figure 3.5 shows the simple trigonometry required for this mapping exercise. Using the sorts of typical low dips common in many English coalfields, it turns out that the zones of possible preferential recharge may extend on the up-sequence sides of seam outcrops for distances of hundreds of metres, if not even more than 1 km. Where successive seams are likely to have been worked to outcrop, and where these seams are less than 110m apart stratigraphically, then their roof-stratal disturbance zones might coalesce with each other, producing a continuous belt of possible preferential recharge which may be several km wide.

Figure 3.5. Logic of determining the "zone of possible preferential recharge" associated with the outcrop area of a worked coal seam.



3.6. Simple Rebound Predictions

3.6.1 Predicting rebound on a "void filling" basis

At its simplest, rebound prediction could be made by determining the volume of coal extracted in a given mine, pond, or coalfield, then comparing this volume with a "reliable" long-term average recharge rate, to obtain an estimate of the time which will elapse before the workings are full to overflowing. However, life is rarely so simple, for several reasons:

(i) This approach ignores the possibility that head-dependent sources of water may accelerate the initial rate of rebound, leading to rates of rise which are more rapid than could be explained by invoking rain-fed recharge alone.

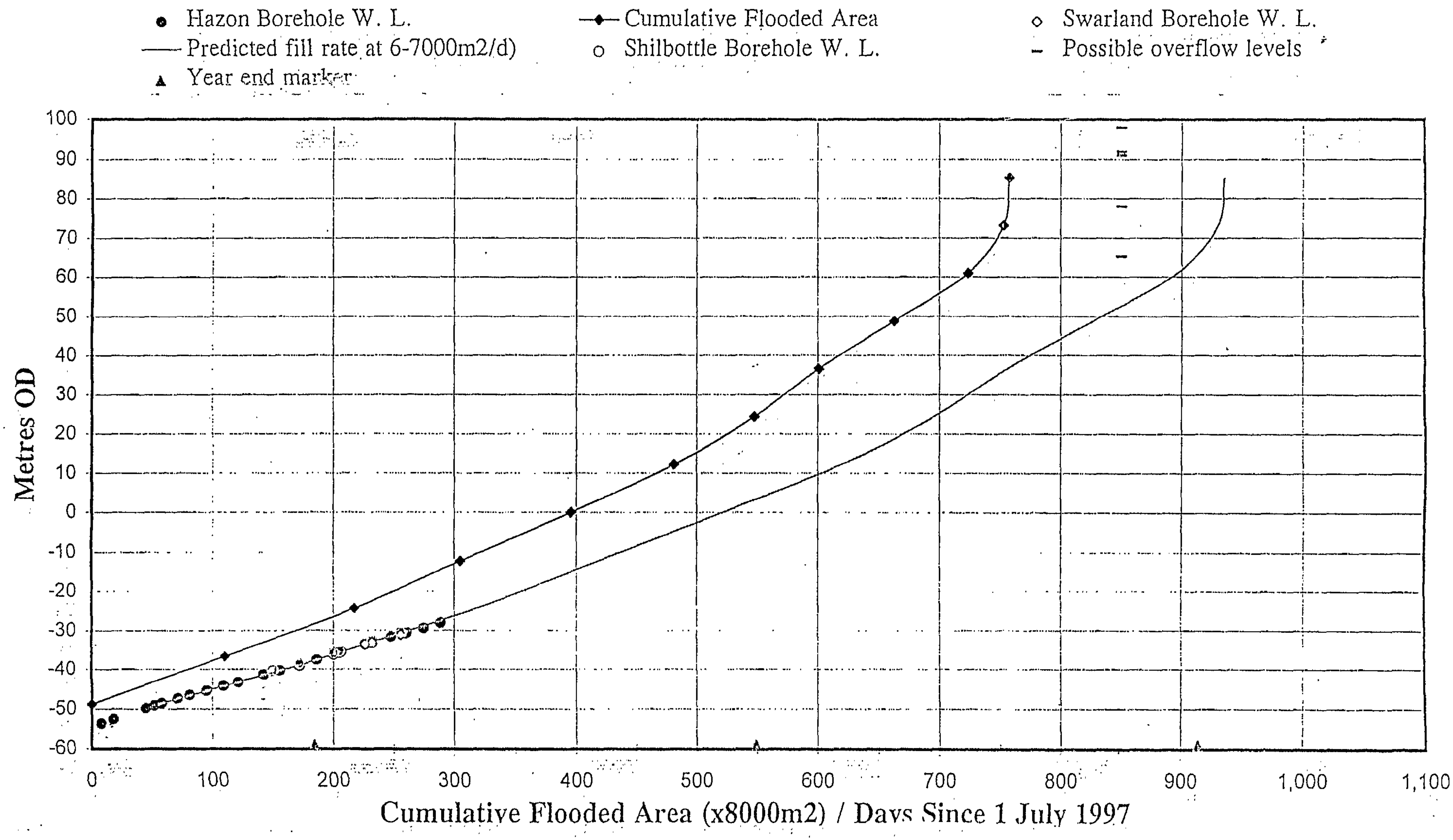
(ii) Estimation of the volume of coal extracted might be made reasonably easily for a small, isolated mine system for which good plans and seam sections are available. However, for large coalfield areas, extracted volume might only be estimable from production figures, which are invariably quoted in units of mass rather than volume. Conversion from mass to volume demands that assumptions are made about the density of coal (while a mean relative density of 1.3 is often quoted, local variations may arise due to differing ash contents etc).

(iii) The volume of coal extracted is not necessarily a good guide to the relict void volume, particularly in longwall mining areas, since some shales will expand to fill former voids, either by floor heave (see Section 2.3) or in the process of spalling to form goaf.

Nevertheless, the appeal to logic made by concepts of void filling are considerable, and have led to their widespread adoption. One of the most important recent applications of this approach was undertaken by staff[2] of the Northumbria Area office of the Environment Agency as part of the initial evaluation of rebound in the Whittle Colliery system in north Northumberland. Figure 3.6 shows the graphical means employed in that analysis. Firstly, a "hypsometric curve" is plotted, which is basically a cumulative frequency graph of mine void volume. The information needed to construct this curve is obtained by digitising the areas of worked panels shown on mine plans, multiplying these by seam thickness, and then tabulating the total volume of mine voids in a number of user-defined discrete depth intervals from the deepest workings up to the surface. Having obtained this hypsometric curve, the future evolution of water levels in the workings is predicted by assuming that the shape of the rebound curve will follow the shape of the hypsometric curve (the axes of the two curves are offset on Figure 3.6 for the sake of clarity, but could be drawn to coincide). Water level monitoring points during rebound are plotted on Figure 3.6, showing that water level rise at Whittle has so far conformed very closely to this model. For a relatively simple system such as Whittle, where nearly all workings are restricted to a single seam, void-filling calculations on this basis are clearly useful.

[2] Primarily Martin Kershaw and Paul Butler

Figure 3.6. Hyspometric curve approach to estimating mine water rebound on the basis of void-filling calculations. Illustration prepared by Environment Agency Northumbria Area, as part of the Whittle Colliery investigation.



Where multiple seam / multiple pond systems are analysed, the timing of lateral transfers of water from pond to pond introduces complexities (as illustrated in Figures 3.2 – 3.4) which are not readily amenable to a simple hypsometric approach.

At best, void-filling calculations may provide a useful preliminary estimate of possible rates of rebound, which can be valuable when deciding whether a particular rebound problem is a short-term emergency or a long-term phenomenon for which there is sufficient time to develop a cost-effective solution. At worst, void-filling calculations can be misleading, leading to serious misjudgements over the scale and timing of an appropriate response to possible rebound problems. For this reason, it is suggested that they are never used as the sole means of estimating the rate of rebound in a given area.

3.6.2 Predicting rebound using a "specific yield" approach

When discussing the application of void-filling calculations, conceptual difficulties frequently arise in relation to the porosity values implicit in the model. These difficulties relate primarily to the fact that goaf has a finite porosity, and a porosity which is almost certain to be closer to that of a gravel (i.e. 0.3 or less) than to that of an open chamber (≈ 1). The responses of some deep-mined systems to seasonal changes in recharge, and / or to fluctuations in pumping, illustrates that goaf panels do have specific yield properties analogous to those encountered in sedimentary aquifers. For instance Minett *et al* (1986) have described the hydrological analysis of a pumping system installed to dewater previously-flooded old coal workings in Northumberland, in advance of opencast reworking. In their analysis, it soon became apparent that the old workings (and associated fractured roof-strata) were dewatering in a manner which suggested they had a specific yield on the order of 2 to 5%. Other studies have yielded estimates of specific yield for mined Coal Measures of a similar magnitude (Sherwood, 1997). This suggests that, instead of performing void-filling calculations which unreasonably assume that all voids remain open, it may be more appropriate to model rebound such that goaf panels and associated roof-strata are modelled as layers (40m to 100m thick) with specific yields on the order of 2 to 5%, with intervening intact strata being assigned low values (10^{-3} or less) reflecting the high specific retention of virgin Coal Measures lithologies, in which pore necks are very small. If this assumption is made, then rebound calculations for multiple-seam ponds will produce stepped rebound curves similar to that shown in Figure 3.4

As this approach is implemented in the GRAM model, further discussion is reserved for Chapter 4.

3.6.3. Fitting an exponential curve to observed data

Groundwater recovery curves in most hydrogeological environments have a shape which conforms to an exponential distribution. Indeed, the fitting of recovery curves to one such exponential distribution (the Theis function) provides one of the principal means of estimating aquifer transmissivity and storativity. In mine water rebound studies, rebound curves conforming to exponential distributions are also encountered. As in the case of "ordinary" groundwater recovery, the exponential distribution reflects the fact that the filling of the dewatered area proceeds ever more slowly as the head differences between the dewatered area and surrounding aquifers are reduced. In other words, an exponential rebound curve indicates gradual reduction in head-dependent inflow to a formerly dewatered area. As explained in Section 3.3, this is particularly common in cases where a single mine system intersects aquifers of regional extension. In such cases, it may be feasible to predict mine water rebound by fitting an exponential curve to the early rebound data, and assuming that the progressive reduction in head-dependent inflow to the voids will occur at much the same rate during the remainder of the rebound period. Figure 3.7(a&b) illustrate some potential pitfalls of this approach. In Figure 3.7 (a), a simple case of rebound for a mine in an extensive aquifer is correctly predicted by fitting an exponential curve to the early rebound data. In the second case (Figure 3.7(b)), the early data to which the curve was fitted related to a period when an extensively-worked lower seam (of relatively high specific yield) was flooding. When the rebound proceeds into relatively low-permeability roof strata, the actual rebound curve is displaced sharply upwards from the presumed exponential distribution, albeit it subsequently conforms to another exponential curve as head-dependent inflows are reduced. In the final case (Figure 3.7(c)), the early data related to rapid filling in an above-seam interval, and subsequent rebound is more gentle. Obviously, of the three cases, 3.7(b) would be the most alarming in practice, as it would shorten the time-scales for preventative action considerably in relation to those originally envisaged.

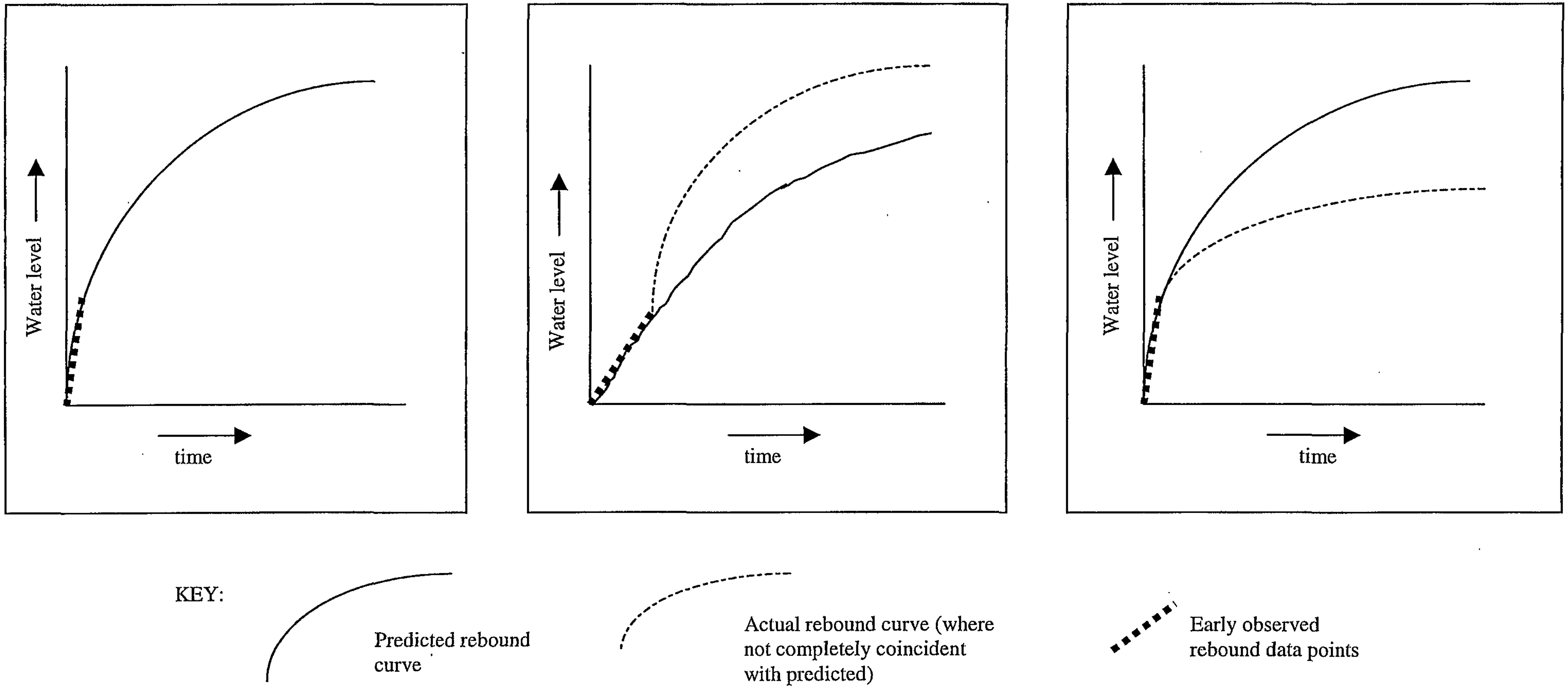
Complexities such as these strongly suggest that rebound prediction will best be undertaken using some technique which can make maximum use of the available data on the locations and geological relations of the workings. Such techniques are the subject of the next two chapters.

Figure 3.7. Some pitfalls of predicting rebound using an exponential distribution fitted to early rebound data.

(a) Small mine in a regional aquifer system.

(b) Layered system, with early data corresponding to flooding of a seam

(c) Layered system, with early data corresponding to inter-seam rebound



4. PREDICTING REBOUND: B. THE GRAM APPROACH

4.1 A Brief History of GRAM

GRAM (Groundwater Rebound in Abandoned Mineworkings) is a semi-distributed, non-Darcian groundwater flow model designed to simulate the process of mine water rebound in extensive systems of mine-workings. At the heart of the GRAM code is the mining engineers' concept of "ponds", as outlined in Section 2.4.3 above.

The roots of the GRAM code go back to 1992, when British Coal announced the discontinuing of dewatering in the Durham Coalfield. A preliminary environmental appraisal, commissioned by Easington District Council with the support of the NUM, made a semi-quantitative prediction that cessation of dewatering in the coalfield might lead to surface water pollution (Younger, 1993).

The National Rivers Authority commissioned a rapid groundwater modelling project to extend the quantitative analysis. This project was undertaken by Wardell Armstrong, who, for want of any accepted alternative, used a standard finite element groundwater modelling package (AQUA) for the purpose. British Coal criticised the Wardell Armstrong model on the grounds that flow in abandoned mines does not follow standard Darcian hydraulics. In response to that criticism, the consultants to Easington District Council and the NUM (i.e. the nascent University of Newcastle mine water research team) proposed to prepare a modelling code which was based unashamedly on the concepts and language used by British Coal's mining engineers. This language was very familiar to the Newcastle team, since it was only 6 years since a major mine water hydrology PhD study (funded by, and undertaken with, British Coal Opencast) had been completed in the University (see Minett *et al.*, 1986).

The development of a new modelling code, based on a novel hydraulic analysis, is not a trivial task. It was clear that the new model must allow for vast changes in permeability and storage properties over short distances (between worked and un-worked strata) and for turbulent flow in roadways and other large voids. Derivation of a physically-based model with these allowances was clearly going to be a time-consuming task (the fruits of which are now described, 6 years later, in Chapter 5 below). However, when the development of a new code was being mooted in early 1993, it was clear that any code would need to be up and running within a few months, rather than years. The first public reference to the code which would eventually be dubbed "GRAM" was made by Younger and Sherwood (1993), and further details were offered by Sherwood and Younger (1994). These early efforts achieved their purpose, in illustrating that the original Wardell Armstrong predictions over the timing of rebound were indeed compatible with the predictions obtained by taking British Coal concepts and language at face value. By March 1994, British Coal had realised that the pumps would be kept running, and, later that year, were handed over to the newly-formed Coal Authority, who have run them ever since. The Coal Authority has since stressed that there are no short- or medium-term plans to terminate the Durham dewatering scheme. So the GRAM models for Durham were shelved, but they had not even begun to gather dust before they were recalled into action. The next outing for GRAM was in Scotland, where it was used on behalf of the Forth River Purification Board in a rapid hydrological evaluation (in less than 10 weeks) of proposals by British Coal to cease dewatering in the Dysart-Leven Coalfield, Fife (Younger *et al*, 1995). Again, GRAM was used as a focal point for debate in a highly contentious situation, which was eventually resolved when the Coal Authority signed up to an agreement with the Scottish Environmental Protection Agency (SEPA) to allow rebound only until the mine water in the easternmost shafts is at –56mOD, at which time a pump-and-treat scheme will be implemented to prevent uncontrolled mine water discharges.

The major phase of development of the GRAM code was undertaken by Julia Sherwood, as part of an EPSRC-funded PhD studentship, which was notionally associated with the Environment Agency / Northumbrian Water Ltd R&D project which formed the basis of much of the other work presented in this report. Long after the Dysart-Leven Coalfield had ceased to be contentious, it was used as a test-bed for the development of GRAM (Sherwood, 1997).

In 1997, GRAM headed south for the first time, when it was applied by Burke (1997) to simulate rebound in part of the South Yorkshire Coalfield (see also Burke and Younger, 1999). Most recently it was used by Walker (1998) to simulate rebound in the North Derbyshire / North Nottinghamshire Coalfield. Some of the results from these applications of GRAM will be reviewed briefly in Section 4.3 below.

4.2. The GRAM Model: Theory and Practice

4.2.1. The GRAM Algorithm

The central concept upon which GRAM is based is that large mined systems can usually be resolved into a number of discrete ponds (see Section 2.4.3), which are largely hydraulically distinct (at least in the early stages of rebound), but which can exchange water with each other via a small number of (usually well-defined) inter-pond overflow points. Typical features forming overflow points include:
roadways
areas where adjoining goaf panels coalesce
old exploration boreholes
permeable geological features

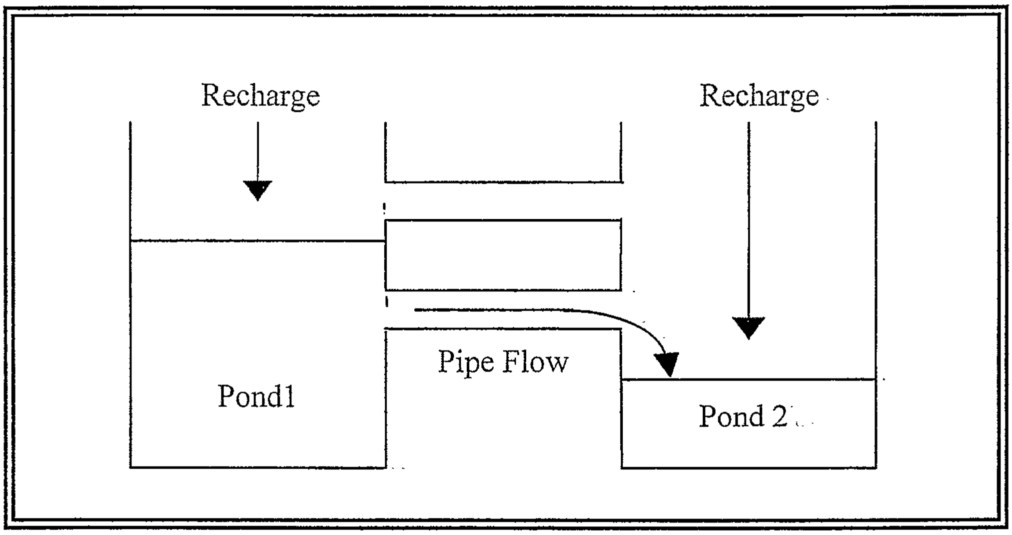
The way in which water rises in neighbouring ponds has been described and illustrated in Section 3.4. In GRAM, the analysis of complex multiple-pond systems is expedited by making some simplifying assumptions, as follows:

- the ponds are laterally bounded by vertical sides of intact (i.e. un-mined) strata through which there is no flow.
- the plan geometry of the ponds can be of any shape.
- the hydraulic gradient *within each pond* is assumed to be flat.
- water level rise within each pond can be represented by a "specific-yield" type of storage change (see Section 3.5.2), responding to the sum of recharge to the pond plus or minus any inter-pond exchanges of water via overflow points.
- flow between ponds and discharge to the surface can be modelled using standard pipe-flow equations. (In reality, connections between worked areas are most often in the form of roadways, which often have a truncated circular section, lending itself to the use of pipe-flow equations).
- inter-pond overflow points flow full-bore.

- the specific yield value may be constant in any one pond, or vertically layered according to the positions and style of working of seams.

A schematic diagram of how two ponds might interact is shown in Figure 4.1.

Figure 4.1. Schematic diagram of flow between two ponds in a roadway, showing the logic of using pipe-flow equations to model water transfer.



Using the simplifying assumptions listed above, it has been possible to derive an algorithm for GRAM which has modest data requirements, realistically aimed at the limited data sets that are usually available. In particular, the fact that the only direct modelling of flow processes relates to inter-pond flows (modelled by pipe-flow formulae) means that hydraulic conductivity values are not needed. This removes a major headache which arises when trying to apply traditional groundwater models to mine water rebound situations (Sherwood, 1997). While parameter values for the pipe-flow equations are needed, these can often be calibrated by matching model response to observed water level data.

The mathematical details of the GRAM model are outlined in Appendix I. Further details are given by Sherwood (1997) and Sherwood and Younger (1997).

One of the main advantages of the GRAM model is its speed of execution. GRAM is easily implemented on PCs, and the relatively short run-times even on this platform mean that the code can readily be used to quantify uncertainties by means of Monte Carlo simulation. A range of GRAM input parameters can be represented by probability distributions; however, two factors influence the choice of which parameters should be treated probabilistically:

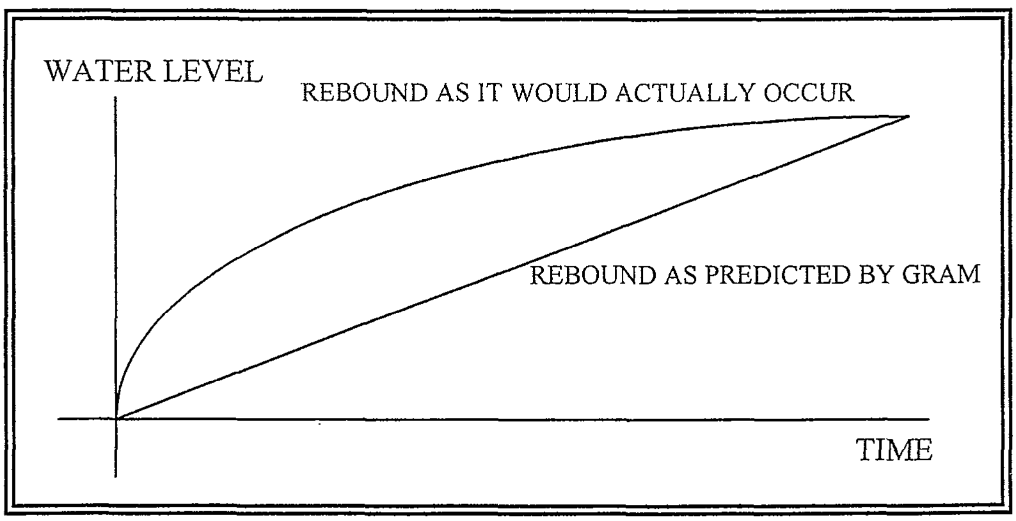
- the perceived reliability of the data, and
- how sensitive GRAM is to errors in their estimation.

The main implementation of GRAM is programmed to allow Monte Carlo simulations to be conditioned on probability distributions representing the storage coefficient, the percentage run-off (and therefore indirectly the recharge rate) and the roughness of any pipe. The use of a probability distribution for pipe roughness is advised for any pipes which were not flowing during the calibration period (because the water levels never exceeded their invert elevations), and are thus un-calibrated. The moments of the pipe roughness probability distribution can be defined from data relating to pipes that were flowing during the calibration period.

Like any model, GRAM is no panacea. Some of the more obvious limitations of GRAM include:

- sensitivity of the model to estimates of specific yield, for which field values are rare in the mining environment
- the lack of explicit modelling of unsaturated zone behaviour (although this can be mitigated to some degree by using Monte Carlo simulation to analyse uncertainties)
- the lack of explicit modelling of river-aquifer interactions. Although baseflow to rivers can be represented as discharge to the surface through a discrete point, outflow through stream beds and induced recharge are not directly quantified. In reality, induced recharge is not likely to be an important component of the water balance of a coalfield area, because of the mining precautions taken to prevent inrushes.
- the assumption that inter-pond pipe-flow occurs full-bore. Although this assumption adds to the simplicity and execution speed of the code, it means that a flow which is a few centimetres deep in the bottom of an intact roadway during the calibration period is represented by a small diameter pipe. In other circumstances, this roadway may be able to transmit far greater volumes of water.
- the assumption of a flat hydraulic gradient within each pond means that graphs of rebound may have an artificially uniform gradient (see Figure 4.2). If the water table within a pond was not totally flat (which is likely to be the case in a reasonable number of cases), then the actual rebound curve may be rather more curved than GRAM would suggest. Hence the initial predictions of water level will be inaccurate. However, the estimate of the total time taken for the pond to fill remains valid as the most significant factors controlling the system are recharge and storage volume.

Figure 4.2: Notional comparison of the likely actual rebound trend in a pond which did not have an entirely flat initial water table, and linear rebound as predicted by GRAM



4.2.2. Using GRAM in practice: platforms, data input and output.

For most of its applications to date, GRAM has been implemented as a FORTRAN code running under Windows 95 / NT. However, the basic GRAM algorithm is sufficiently simple that it can also be programmed to run in purpose-written spreadsheets. For instance, Younger (1998b) implemented a customised version of the GRAM algorithm to run as a worksheet in Microsoft EXCEL™ when simulating mine water rebound in the North Pennine Orefield, where two ponds were strongly influenced by head-dependent groundwater inflows.

In the FORTRAN implementation of GRAM, the data are input via a series of ASCII files. Although the volume of data is not large, finding the location of a calibration variable can be time consuming, particularly to the uninitiated user. Therefore a pre-processor (WinGRAM), written in Visual Basic 3.0 and designed to run in a Windows™ environment, has been developed. WinGRAM displays the data from the input files in a format which allows the user to easily pinpoint pieces of data. It also has the facility to universally enter data for any parameter for all the ponds or pipes. WinGRAM is complemented by an Excel 5.0 Macro postprocessor (also written in Visual Basic 3.0), which is used whilst fitting the model. The macro automatically imports the output file containing the water level data and displays it with the recorded data on a graph. It also calculates the residual mean and the absolute residual mean of the differences between the two data sets. This allows the user to concentrate on the conceptualisation of what is happening in the system rather than the mechanics of moving data around.

The input data required for a standard GRAM simulation are as follows:

- precipitation (mm),
- evaporation (mm/year),
- attenuation of recharge over time (see Appendix I)
- area of each pond (m^2),
- area of surface catchment of each pond (m^3),
- specific yield (and its vertical distribution, if desired)
- initial water level (mAOD)
- percentage surface run-off,
- abstractions (m^3/timestep),
- marine inflow, where there are undersea workings (m^3/timestep),
- inflow from adjacent mines or aquifers (m^3/timestep),
- the number of pond connections, and for each their:
- height (m),

- roughness coefficient (mm),
- diameter (m),
- length (m),
- kinematic viscosity (m^2/s),

- the number of surface discharge points and for each their:

- height (m),
- roughness coefficient (mm),
- diameter (m),
- length (m)
- kinematic viscosity (m^2/s),

- which parameters will have Monte Carlo simulation applied to them and for each a probability distribution

The execution of the GRAM code follows the sequence shown in Figure 4.3. The use of the Monte Carlo loop is optional, for GRAM may be run in straightforward deterministic mode. If the loop is used, a minimum of 1000 realisations is usually sought.
GRAM outputs the following data:

- A water balance consisting of a comparison of the change in storage volume and the difference between the volume of water entering and leaving the system,

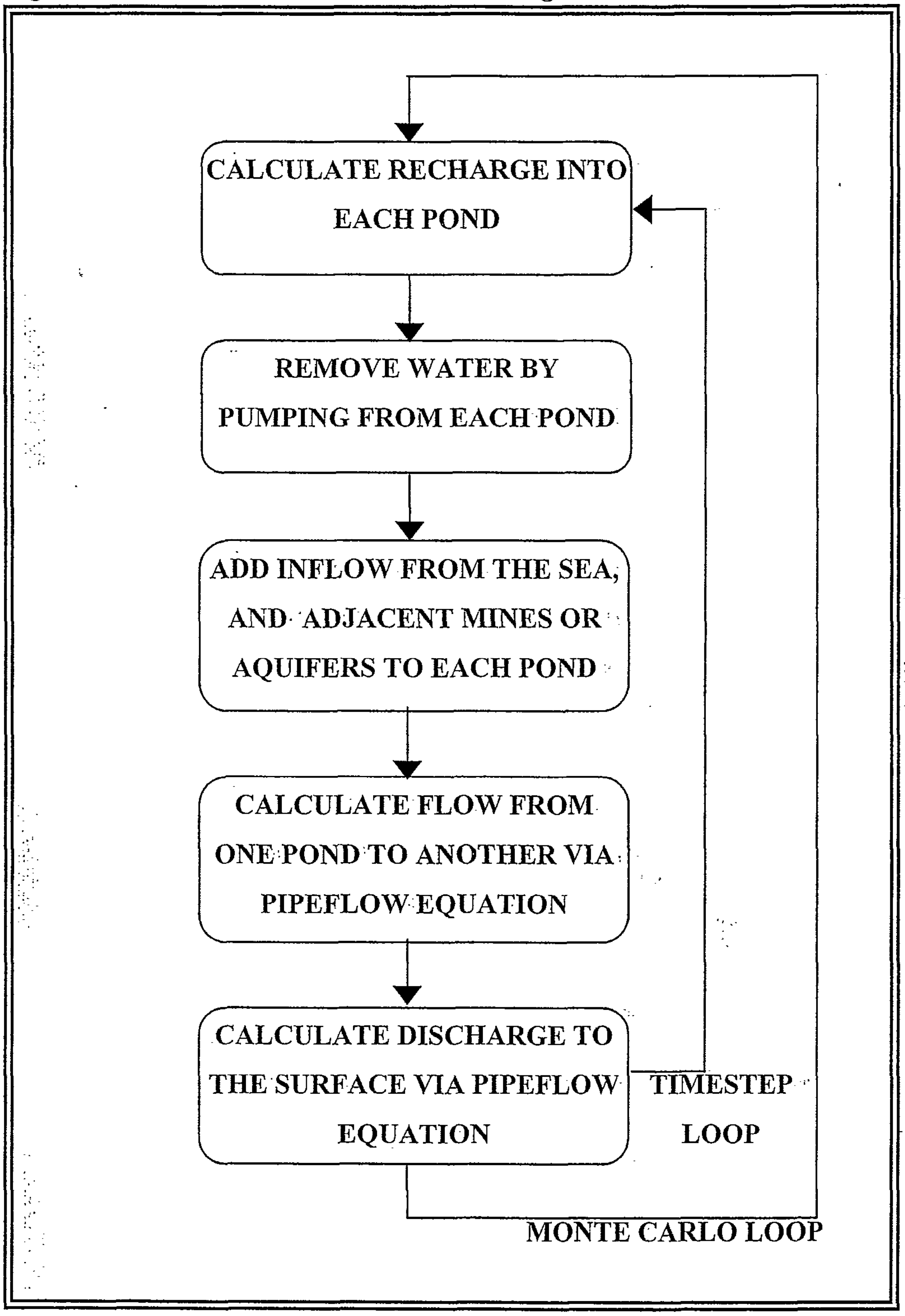
- water level time-series for each pond,

- the time when surface discharge commences at each discharge point,

- the volumetric flux of each surface discharge over time and

- the average volume of flow from each surface discharge point.

Figure 4.3. Flowchart of the basic GRAM algorithm



4.3. Examples of the Application of GRAM

4.3.1. The Durham Coalfield

GRAM was first developed as "a bespoke lumped-parameter model" (Younger and Sherwood, 1993) which represented the central area of the Durham Coalfield as a system of ponds. At the time this model was developed, public access to mine plans was restricted, ostensibly on the grounds that the plans were "in transit" from Tursdale, Co Durham, to the new national coal mining records office in Bretby, Nottinghamshire. Under these difficult circumstances, definition of ponds was therefore made on the basis of geological reasoning (e.g. locations of major faults), consultations with ex-miners, and identification of areas in which the water level in workings was similar (using data published by Harrison *et al.*, 1989). Four rectangular ponds were identified in this manner (Younger and Sherwood, 1993). When seam summary sheets were eventually released via the NRA some months later, it was found that the four ponds were similar in total area, and reasonably coincident in position, with distinct ponds identifiable on plans of the seams between the Brockwell and the Hutton. The Durham application of GRAM suggested that total rebound of the coalfield would require around 40 years, with rebound being complete in the southern part of the coalfield first, then sweeping northwards along the axis of the Wear Valley (Younger and Sherwood, 1993; Sherwood and Younger, 1994). If this was allowed to happen, it was predicted that as much as 60 $Ml.d^{-1}$ of mine water would eventually discharge into the River Wear and its tributaries. Although these predictions differed from those of Wardell Armstrong (1993) on points of detail, both studies came to similar conclusions about the time required for complete rebound and the future quantity of mine water discharge.

4.3.2. Dysart-Leven Coalfield.

As mentioned in Section 4.1 above, the most thorough application of GRAM has been to the Dysart-Leven Coalfield in East Fife, Scotland, where it was used to inform debate between the Forth River Purification Board and British Coal over the possible consequences of ceasing dewatering in the Frances and Michael collieries. The early predictions concerning this coalfield have been presented by Younger *et al* (1995), and subsequent refinement of both the GRAM code and the predictions for this area are fully documented by Sherwood (1997) and Sherwood and Younger (1997). Hence only a brief summary will be given here.

Study of mine plans revealed that the workings of the Dysart-Leven Coalfield could be grouped into a system of five ponds, the positions and areas of which are indicated on Figure 2.6 (p. 34). Detailed analyses of mine plans revealed the nature and positions of inter-pond overflow points, which were mainly old roadways, with a few corresponding to areas of coalesced goaf panels. As substantial areas of the three coastal ponds actually lie under the sea bed, it was important to take into account the loss of recharge from the sea when the water level in the mines reached sea level. Hydrochemical interpretations were used to discriminate the marine and terrestrial components of the total water make (33 $Ml.d^{-1}$) at Frances and Michael. It was concluded that 45% of the total water make of these coastal shafts was of marine origin. Consequently, the post-rebound onshore discharge rate was predicted to amount to around 18 $Ml.d^{-1}$, of which 1.6 $Ml.d^{-1}$ were predicted to enter the River Leven, 2.6 $Ml.d^{-1}$ to enter the River Ore, and the remainder to enter small coastal streams and the Firth of Forth.

At the time of writing, rebound is underway in this area (although it will be arrested by renewed pumping once the water levels reach –56mOD). Monitoring of the rebound data show that initial recovery in the coastal ponds was more rapid than the initial GRAM predictions (probably for the reasons outlined in Figure 4.2 and associated text), though the prediction of overall time to total rebound (were pumping not re-commenced) is in agreement with the original median GRAM prediction of around 18 years (K Parker, Coal Authority, personal communication, 1998).

4.3.3 South Yorkshire.

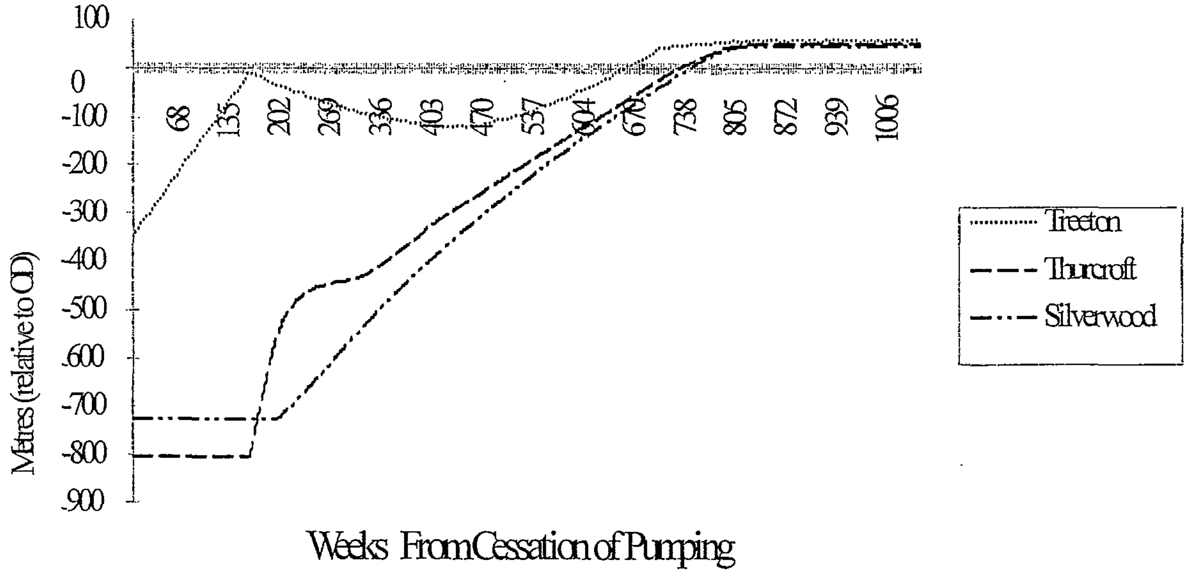
The great South Yorkshire collieries of Treeton, Thurcroft and Silverwood, which mined the Coal Measures to the east of Rotherham, were abandoned in the early 1990s as part of the programme of colliery closures which also swept the Durham and Dysart-Leven coalfield into oblivion. Dewatering was terminated at Treeton in late 1991, at Thurcroft in 1993, and at Silverwood in 1995. Treeton was a wet colliery, which coped with a total water make of 2.7 $Ml.d^{-1}$. Thurcroft was relatively dry (water make of only 0.016 $Ml.d^{-1}$), and Silverwood had an intermediate water make of just under 1 $Ml.d^{-1}$. The only remaining colliery in South Yorkshire (Maltby) deals with a very limited local water make of < 0.01 $Ml.d^{-1}$, and is also protected by limited pumping at Carr House pumping station, which is intended to prevent water from Silverwood and adjoining mines migrating eastwards through old workings.

To understand the fate of the 3.7 $Ml.d^{-1}$ of mine water formerly pumped from the three east Rotherham collieries, Burke (1997) used GRAM to simulate rebound in the area. Burke identified three possible scenarios for rebound in this area:

1: No hydraulic connection between Silverwood and Maltby Collieries
2: There is limited hydraulic connection between Silverwood and Maltby
3: There is excellent hydraulic connection between Silverwood and Maltby

For the first scenario, Burke (1997) obtained rebound time-series for the three collieries as shown on Figure 4.4. The patterns of the recovery curves can be explained as follows: Treeton, with a high water make, floods quickly until a connection in the Barnsley Seam allows down-dip migration of water into the relatively dry workings of Thurcroft. Thurcroft then begins to flood rapidly, until a connection to Silverwood slows its rate of rise, and provokes rapid rebound in Silverwood. As the water levels in Thurcroft and Silverwood begin to harmonise, rebound picks up again in Treeton, and all three systems discharge to the surface within a few months of each other. The other two scenarios studied by Burke (1997) produce similar rebound patterns. However, the timing of rebound to surface is later in Scenario Two as some of the recharge is diverted to Maltby. Under scenario three, the entire water make of the three collieries passes eastwards to Maltby Colliery, thus postponing surface discharges until the eventual closure of that mine.

Figure 4.4. GRAM predictions of rebound in the Treeton, Thurcroft and Silverwood Collieries, South Yorkshire, assuming no flow passes to Maltby Colliery (after Burke, &Younger 1999).



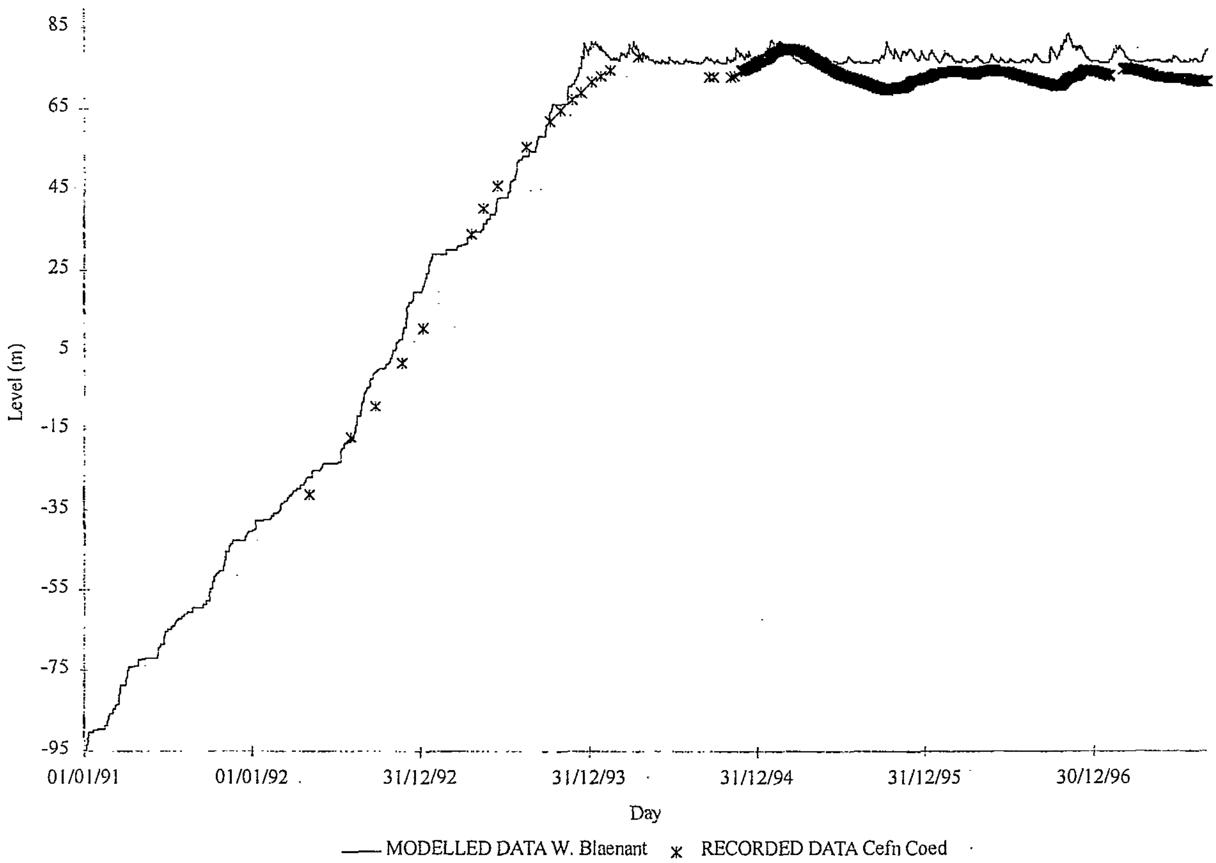
4.3.4. North Derbyshire / North Nottinghamshire Coalfield.

GRAM was applied by Walker (1998) to investigate possible rebound scenarios in the vicinity of nine recently-abandoned collieries in the Bolsover area. Four ponds were delineated by Walker (1998), and GRAM simulations predicted that mine water discharges totalling between 3 and 6 $Ml.d^{-1}$ might emerge into the River Doe Lea near Bolsover between 10 and 20 years after the end of dewatering. In this study, particular emphasis was given to mine plan analysis, precluding extensive calibration and testing of GRAM. The predictions cannot be regarded as definitive, but they have served to pave the way for later, more detailed, modelling work.

4.3.5. Blaenant - Ynysarwed.

The mine water discharge into the Neath Canal at Ynysarwed (Younger, 1994; Ranson and Edwards, 1997; Ranson *et al*, 1998) arose after rapid rebound in deep mine-workings formerly accessed from Blaenant Colliery. Simulation of rebound in this system has been undertaken using the 3-D physically-based model described in Chapter 5 (see Younger and Adams, 1998). During that exercise, GRAM was used to improve estimates of the initial water level in the workings before rebound, which was not easily discerned from the data provided by British Coal. An impressively close fit between GRAM predictions and the observed rebound data was obtained with minimal calibration (Figure 4.5). This illustrates that, in this case, head-dependent inflows were negligible, and the entire water-make of the workings was sourced from recent recharge.

Figure 4.5. GRAM simulation of rebound in the Blaenant – Ynysarwed system compared with observed rebound data (after Younger and Adams, 1998)



4.3.6 Whittle Colliery.

In a similar application to that for Ynysarwed, GRAM was used to undertake rapid "what-if" modelling of the Whittle - Shilbottle abandoned collieries in Northumberland, prior to constructing a fully 3-D physically based model. In this area, there are two obvious mine water ponds and one "dubious" pond. The two "obvious ponds correspond to the relatively recent (i.e. largely post-WW II) workings of Whittle Colliery and Shilbottle Colliery, which are known to be inter-connected by a roadway which was already flooded at the time of the study. The water level in these two ponds is essentially the same, and is rising at around 10cm per day from a position of several tens of metres below OD. To the north of the Shilbottle workings is an isolated body of older workings (the "Bilton Bank Workings") in which the water level is steady at about 50m AOD. The key issue investigated using GRAM was whether the Bilton Bank workings were in any way connected to the Shilbottle pond. From simulations taken using a range of feasible recharge scenarios, it was swiftly concluded that the Bilton Banks *must* be overflowing to Shilbottle Colliery, otherwise the rate of rise observed in the workings simply cannot be explained. Subsequent application of the 3-D physically-based model has borne out these deductions. This has important implications for pollution prevention plans, as it suggests that "drowning out" of the connection with Bilton Bank (which is thought to be provided by a fissured limestone above the seam) will provoke mine water rebound to the north of the current area of concern. This confirms that the mine water rebound must be arrested at (or preferably below) 50m AOD if surface water pollution is to be precluded.

4.3.7. Frazer's Grove Fluorspar Mine.

The most recent application of the GRAM algorithm was to the last working fluorspar mine in the North Pennines, in which the deepest workings are scheduled for abandonment by January 1999. Younger (1998b) identified two large ponds and two small ponds in the Frazer's Grove mine complex. He also used hydrochemical evidence and underground observations to deduce that much of the water make of the mine is head-dependent inflow from the Great Limestone. On this basis, Younger (1998b) implemented a customised version of the GRAM algorithm in an EXCEL spreadsheet to produce a simple deterministic simulation of rebound which indicated the kinds of time-scales over which decisions over the future management of the site would have to be made.

5. PREDICTING REBOUND: C. PHYSICALLY BASED MODELLING IN 3-D

The third modelling approach, which has been developed and implemented in this programme of work, is a development of an established method of simulating the water cycle in catchment – based hydrological models. In the past twenty years, the increasing power and versatility of modern computers has enabled the development of fully 3-D, spatially distributed models. In such models, the hydrological catchment is discretised into a series of elements, and the non-linear equations which govern the flow of water and the transport of contaminants and sediments, are solved using numerical methods. Usually models tend to be applied separately to surface water and groundwater problems, but at Newcastle University, the SHETRAN modelling system has been developed which couples the surface and subsurface processes together into an integrated, fully 3-D model of water flow and transport (Ewen et. al. Draft paper). In SHETRAN, a modular system of linked components has been adopted which facilitates this.

5.1. Methodology

The physically based methodology which has been adopted for the modelling of abandoned mines after pumping has ceased employs a variant of the system described above. The major advantages of this modelling approach over the simpler models described above are:

(i) It is capable of identifying the location of mine water discharges in addition to the time of discharge and the volume discharged. Lumped models usually require an *a priori* knowledge of the location of these discharge points.

(ii) Predictions of magnitude and timing of mine water discharges are improved due to better representation of groundwater flow through mined systems, for example through the incorporation of sloping strata, by the model, improved estimates of recharge rates and improved modelling of boundary conditions.

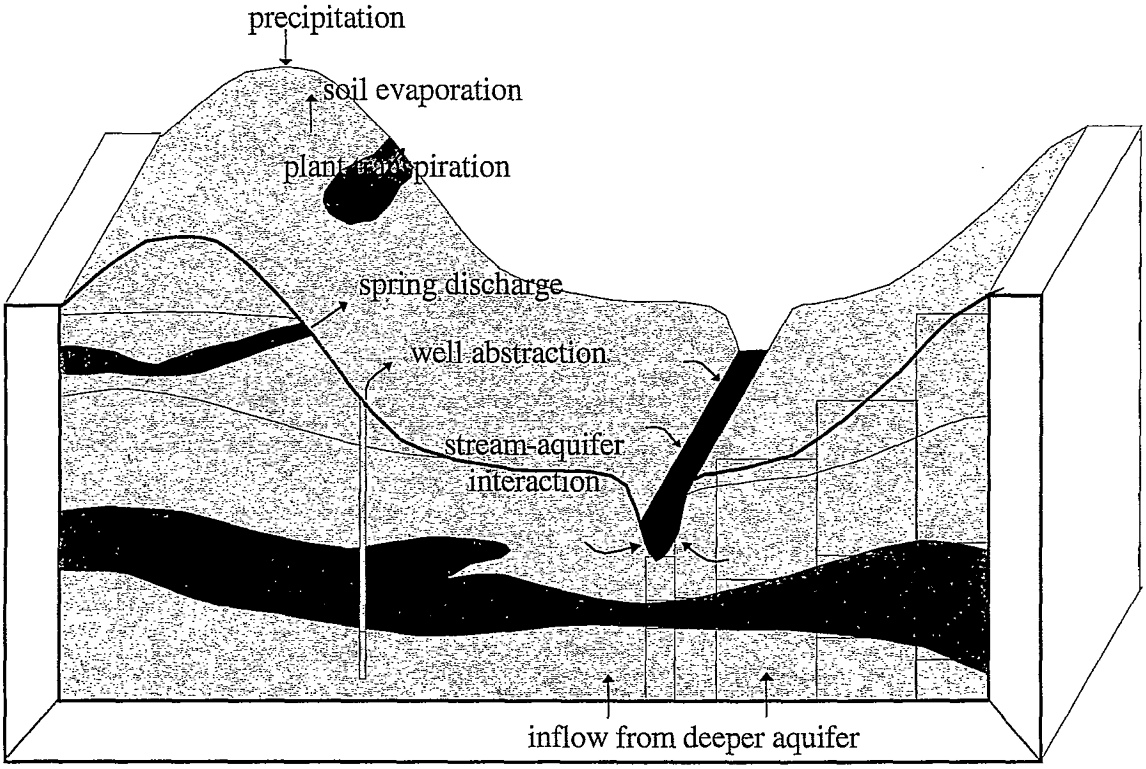
(iii) Preferential pathways of flow such as open roadways and shafts can be explicitly modelled. This may have important implications if water quality modelling is to be carried out as a post-processing exercise.

(iv) Good representation of the mined system in regards to the choice of model parameters and identification of boundary conditions should produce accurate results without extensive calibration of parameters and also allow future predictions of rebound and surface discharges to be made with confidence.

5.1.1 The VSS Component of SHETRAN

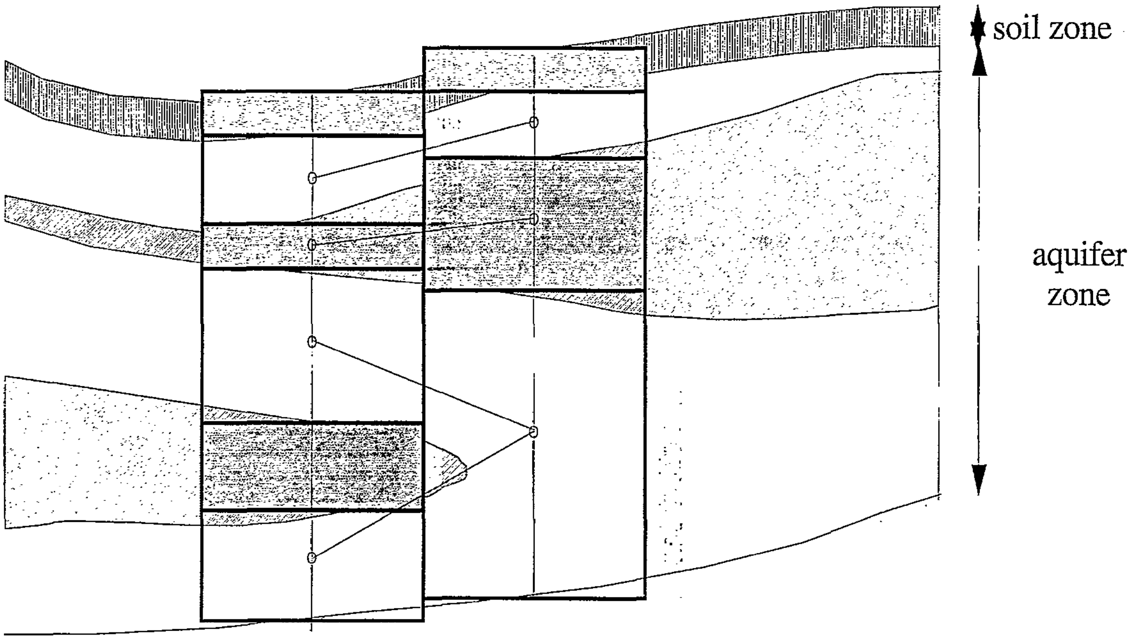
For the modelling of 3-D groundwater flow in variably saturated porous media, the VSS (Variably Saturated Subsurface) component of SHETRAN, has been developed, (Ewen *et al.*, draft paper). This component allows the modelling of complex geological structures by the model. The VSS component is fully coupled with the surface processes, so groundwater flows can be controlled by surface conditions and spatially varying recharge. Fig 5.1 depicts the physical processes and source/sink terms, which can be simulated by the existing VSS component.

Figure 5.1. The Variably Saturated Subsurface (VSS) Component for SHETRAN



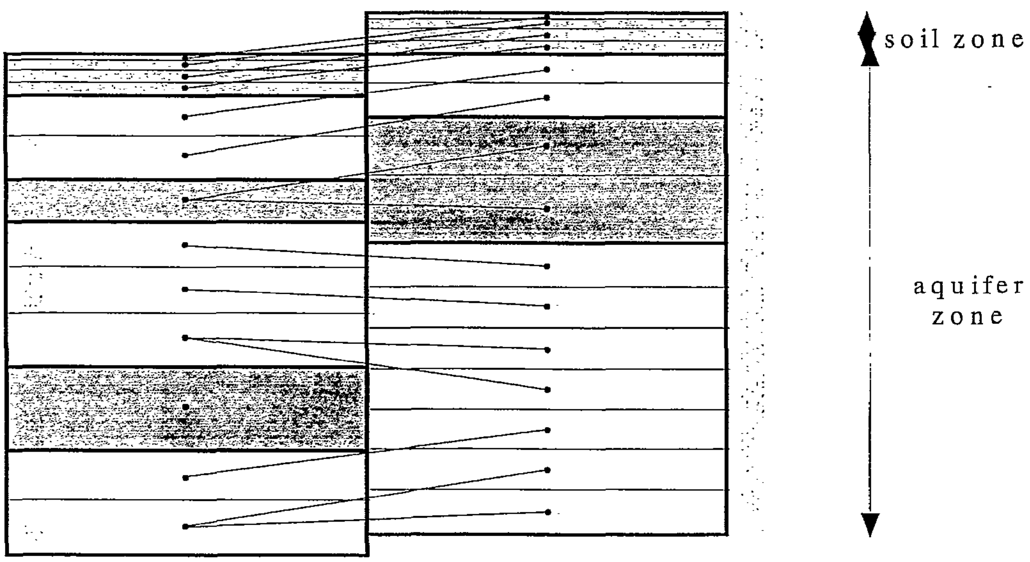
In this application of SHETRAN, processes at the ground surface are not directly modelled, however the front-end of SHETRAN is utilised for the input of data and control of the simulations. The VSS component then obtains the catchment configuration, boundary conditions and source and sink terms (e.g. well abstraction flows), from the SHETRAN model. For mine water modelling, a time-varying recharge term is specified as the top flow boundary condition. The VSS component is capable of simulating the combination of saturated and unsaturated conditions found in abandoned coalfields. VSS discretises the aquifer into a series of laterally connected vertical columns (Fig 5.2.). In this application of the VSS, the soil zone indicated in this figure is not represented directly in the model as the top layer is usually either Coal Measures strata or drift deposits (e.g. boulder clay).

Figure 5.2. Geological Layering and Connectivity in VSS



Each column comprises a series of computational cells which represent the heterogeneous nature of the aquifer. The modeller can select which cells are connected laterally in order to simulate sloping strata (Fig 5.3).

Figure 5.3. VSS Computational Cell Definitions and Interconnections



The equations governing the simulation of three-dimensional flow in variably saturated porous media are complicated, and for convenience summarised in Appendix II

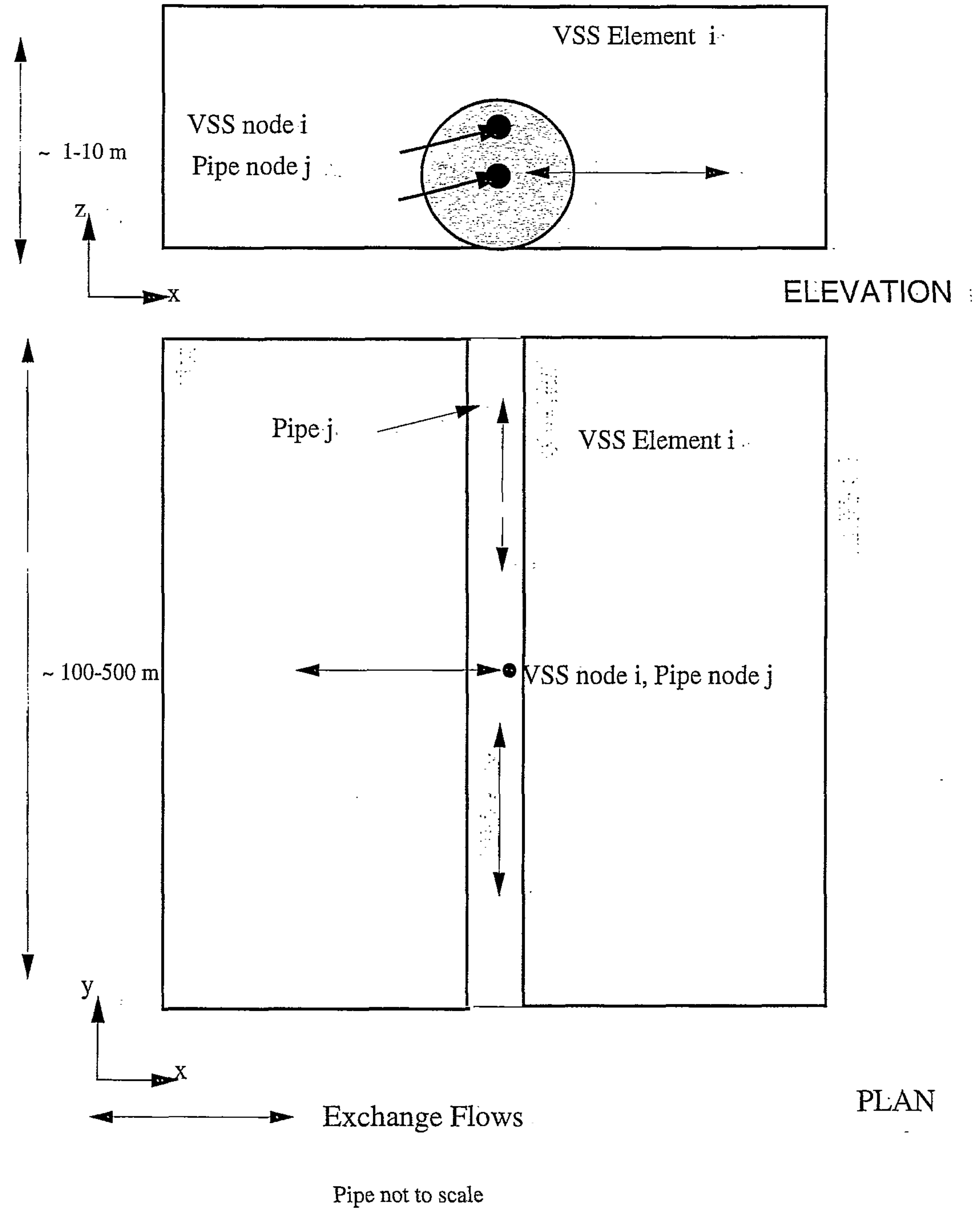
5.2. Adapting VSS to Simulate Turbulent Flow

In regions where the flow regime is likely to be turbulent (non-Darcian), as described in Chapter 2, groundwater flow cannot be directly simulated by the existing VSS component. In order to simulate turbulent flow, a pipe network model has been adopted which will allow the larger flow pathways to be discretised by a series of interconnected conduits (*pipes*). This network model has been coupled to the VSS model in order to model the interactions between the aquifer (representing the intact strata) and these conduits in the mined out strata. The combined component is known as VSS-NET.

The solution of a pipe network, to calculate pressure heads and flow rates, requires a numerical algorithm, as the equations for turbulent flow are non-linear. In a pipe network, the pipes are joined together at *nodes* where sources and sinks are added into the network. One method referred to as the Gradient Algorithm (Todini & Pilati 1989), has been applied to several networks using a microcomputer program, for comparison with other methods (Salgado *et.al.* 1989). For the simulation of turbulent flow in mined strata this method has been chosen as the most computationally efficient and robust numerical method currently available.

The connections between the pipes and the columns in the linked VSS-NET component are shown in figure 5.4.

Figure 5.4. Configuration of Pipes and VSS Elements in VSS-NET



Each pipe is connected to a cell in the vertical column and flows between the pipe and column are calculated at each model timestep. The equations employed and the solution algorithm are described in Appendix II

5.3. Data Requirements

Physically based modelling requires large quantities of hydrogeological data for the model parameters and validation. The data requirements are summarised in Table 5.1. below, which also lists the sources which have been employed to obtain the data.

The data requirements are quite onerous although the model is relatively insensitive to certain parameters, in particular the pipe roughness and unsaturated zone parameters. The actual diameter of the pipes used in the NET component of the model may differ from the real roadway dimensions as: (i) circular pipes are used in NET, (ii) the model runs under the assumption that the pipes are assumed to run full for the majority of the simulation period This may require a pipe of smaller cross-sectional area than the roadway, as the roadway may be only be partially full of water. The available literature on hydrogeological parameters such as hydraulic conductivity is fairly sparse for Coal Measures, especially for mined-out strata, so some calibration of these parameters will inevitably be required. In locations where sandstones associated with the Coal Measures are used for water supply, for example the Pennant Sandstones in the South Wales coalfield, there may be pumping test data available from the Environment Agency.

Table 5.1. Data Requirements for Physically Based 3-D Modelling

Data Requirement	Source
Measured groundwater levels	Environment Agency boreholes British Coal / Mining Company records
Historical pumping rates	British Coal / Mining Company records
Recharge rate	Environment Agency rainfall data Environment Agency runoff data (river gauge flow records) MORECS[1] actual evaporation and soil moisture deficit data
Surface topology	Ordnance Survey mapping,
Seam elevation and location of outcrops	BGS mapping, mine plans, shaft drilling records
Strata type and thickness	BGS mapping, mine plans, shaft drilling records
Location of workings	Mine plans
Location of major roadways	Mine plans
Boundary conditions (e.g. fixed heads, no flow)	BGS regional hydrogeological mapping Environment Agency measured levels
Model Parameters: 1, VSS	
Porosity	Environment Agency pumping tests Literature
Specific yield / storage	Environment Agency pumping tests Literature
Hydraulic conductivity	Environment Agency pumping tests & packer tests Literature
Unsaturated zone parameters for soil hydraulic (e.g. soil water retention curve)[2]	Literature
2, NET	
Pipe length	Mine plans
Pipe diameter	Physical basis, see text below
Pipe roughness	Physical basis, see text below
Node elevation	Mine plans
Location and elevation of fixed heads (e.g. adits and shafts)	Mine plans BGS mapping

[1] MORECS - Meteorological Office Rainfall and Evaporation Calculation System.
[2] The van Genuchten equation (van Genuchten, 1980) equation is usually employed to represent the soil hydraulic functions.

5.4. Recharge Estimation

The estimation of the recharge rate to the workings is a crucial step in the data pre-processing. The model is particularly sensitive to the recharge rate and poor results will be obtained if the rate is incorrect. It is important to use daily values wherever possible, otherwise weekly or monthly values can be used in the model. Currently, two approaches have been adopted for the case studies investigated using the model. The first adopts the approach of Touma (1996), in utilising a daily or weekly soil water accounting model. If the net rainfall (the rainfall (P) minus any actual evapotranspiration (E_A)), minus the surface runoff (S) exceeds the soil moisture deficit at the start of the timestep SMD_t, recharge (R) will occur. This can be expressed as:

$R = [P - (S + E_A)] - SMD_t$ **Eqn. 5.1**

All values are in mm per timestep except SMD which is in mm.

This method has been applied successfully to the modelling of the Northumberland and Durham coalfields.

The second method is to employ a quasi-physically based lumped model such as NUARNO (Adams *et al.* draft paper) to calculate the recharge, using a similar method. The model performs its own soil moisture accounting, so this method can be used if MORECS soil moisture deficit data are unavailable. This approach has been used in the Ynysarwed-Blaenant case study, described briefly below (see 5.7). This technique requires calibration of the model discharge predictions against an observed historical river flow record.

SHETRAN can also calculate recharge directly. However extensive soil and vegetation data sets for the catchment including spatial mapping of soil and vegetation types, are required since a fully physically based vegetatative canopy model is incorporated (Parkin, 1996). Calibration is not usually required if the data are of good quality, however these data are often prohibitively costly to obtain. In the future the coupling between SHETRAN and macro-models such as NUARNO will probably become fully automatic so that pre-processing to determine recharge rates given meteorological data will not be required.

5.5. Modelling Method

The application of the SHETRAN to abandoned coalfields requires a similar approach to that employed when modelling using GRAM. Essentially three stages are required: firstly, the data files sets for the model simulations are created, where necessary using pre-processor programs or spreadsheets. Secondly, the model is then run for a historical time period for which observed groundwater levels and/or adit discharges are known. The results are compared with the historical data, and if found to be unacceptable, changes are made to the model parameters or input data, and the model is re-run. This procedure is carried out until acceptable results are obtained. Compared to GRAM, run times are quite long due to the small timesteps required to simulate flows in the pipe network model. However, the model should require fewer calibration runs if the choice of parameters, physical discretisation of the mined area, and recharge rates in the model are good representations of actual conditions.

If a long period of observed data are available it should be possible, if there has been some calibration of model parameters, to validate the chosen parameter set by running the model, without further calibration, and comparing the results with the observed data during the validation period.

Thirdly, if future simulation of groundwater rebound and possibly surface discharges are required, the model can be run with "synthetic" recharge data. If a sufficiently long time series is available then one method of generating synthetic data is to extract dry, wet or average years from this record and use them as the recharge input for the predictive simulations.

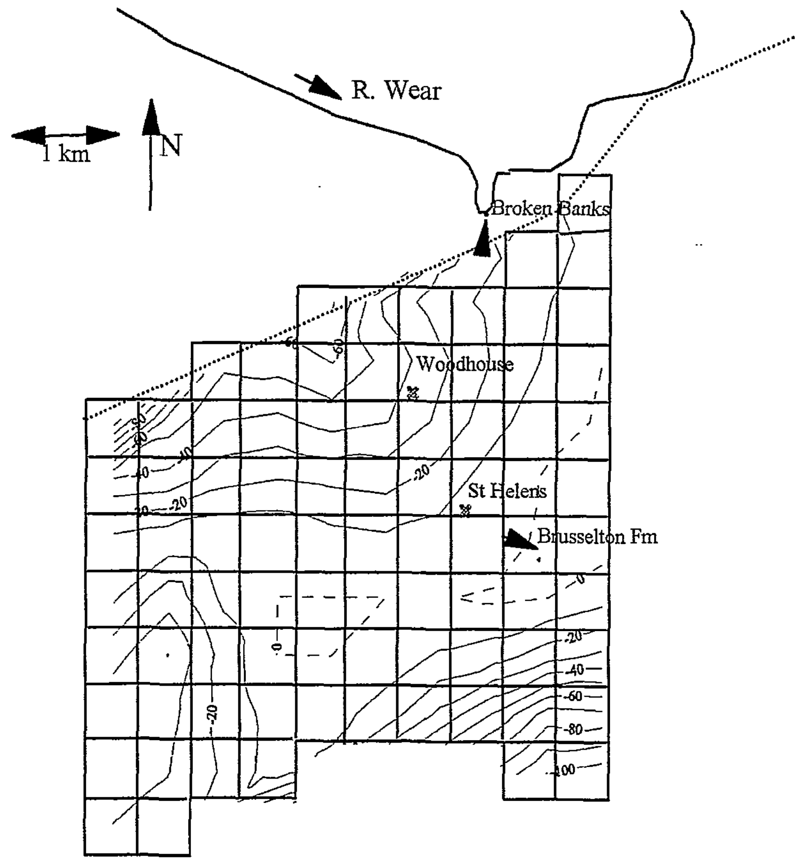
The model has generally been applied to mines comprising interconnected roadways linking mined-out longwall panels, but could also be applied to older mines which used the bord and pillar extraction method, if open roadways represent the major groundwater flow paths in these mines. The model can also be used to investigate different configurations of roadways and the interconnections between mined areas. Computational requirements make it desirable to use models like GRAM for large, interconnected coalfields such as the Durham coalfield, and SHETRAN for focussing on smaller areas such as the Bishop Auckland region. It is possible to use the results from GRAM simulations (e.g. the water levels in a "pond") as a boundary condition for SHETRAN.

5.6. Bishop Auckland Case Study

To illustrate the capabilities of the model, some results are included from the Bishop Auckland Case Study. The primary aim of the study was to demonstrate that the model could correctly predict the emergence of mine water at several locations around Bishop Auckland. The area had been previously modelled using MODFLOW (Lancaster,1995) with little success, as the model was not capable of converging when all the coal seams worked in the locality were included. Lancaster eventually simplified his model to simulate only the most worked seam (the Brockwell). The data from the MODFLOW simulations were used to set up the initial and boundary conditions. The northern edge of the model domain was defined by the major Butterknowle fault, which provides a well-defined no-flow boundary. To the south, the outcrop of the Brockwell seam was used to define a no-flow boundary. The east and west boundaries were less clearly defined. However from the pumping records of the old collieries in the area, and the work of Minett (1987), it became apparent the regional groundwater flow was from west to east, and that the inflow into the modelled area could be presumed to equal the net outflow across the eastern boundary.

Figure 5.5 depicts the model grid used to perform the SHETRAN simulations. The finite-difference grid comprises 500m square grid cells. The contours depict the depth of the phreatic surface at the end of December 1991. The dotted contours indicate areas where the potential of flooding exists. There are two discharges of mine water known in this area. The first originates from a spring next to St Helens Colliery shaft (NZ 197269), and emerges to flood pasture at Brusselton Farm. The model has predicted this area will be flooded at this time. There is a second discharge of mine water into a small stream at Broken Banks (NZ197295), which flows into the River Wear, which, in terms of iron loading, is the most serious in the area, which was also predicted by the model.

Figure 5.5. Bishop Auckland Case Study, Modelled Area



Key

AMD Discharge

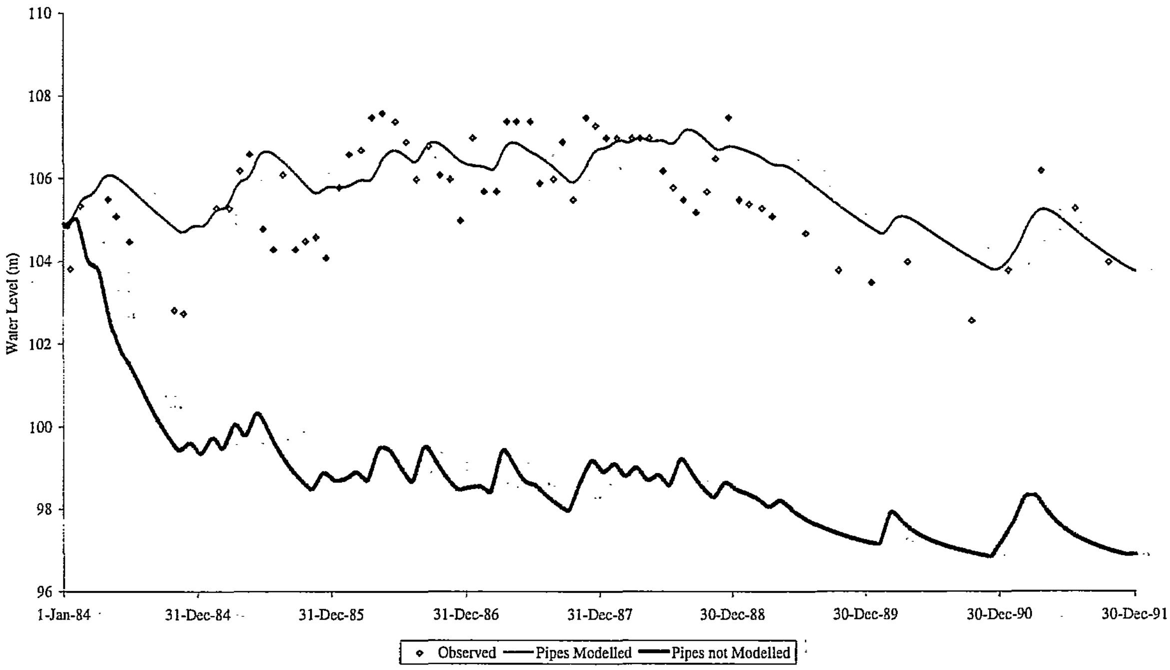
Butterknowle Fault

Colliery Shaft

Contour of Phreatic Surface Depth (m)

To simulate this discharge, a roadway network was included in the Busty seam, which was extensively worked just south of the discharge. The model predicted discharge values of the same order of magnitude as a single measurement of the flow rate taken in April 1994 (Younger & Bradley, 1994). In order to fully validate the model, a longer time series of observed flow rates are required, however as a first application the results were quite encouraging. Figure 5.6 shows the measured and predicted groundwater levels at Woodhouses Shaft for an eight year simulation period (1984-91). During this period, groundwater levels appeared to be fluctuating naturally as a function of precipitation, rebound in this coalfield having finished some years earlier (see Figure 2.7). The difference between modelled and observed levels is usually less than 2 metres and the shape of the observed recession curve during the 1988-9 drought is well represented. For comparision, the figure shows the results from a simulation where roadways in the Harvey and Hutton seams were not modelled using a pipe network. There has clearly been a significant improvement in the predicted groundwater levels as a result of the enhancement of the model.

Figure 5.6. Measured and Predicted Water Levels, Woodhouses Shaft 1984-91



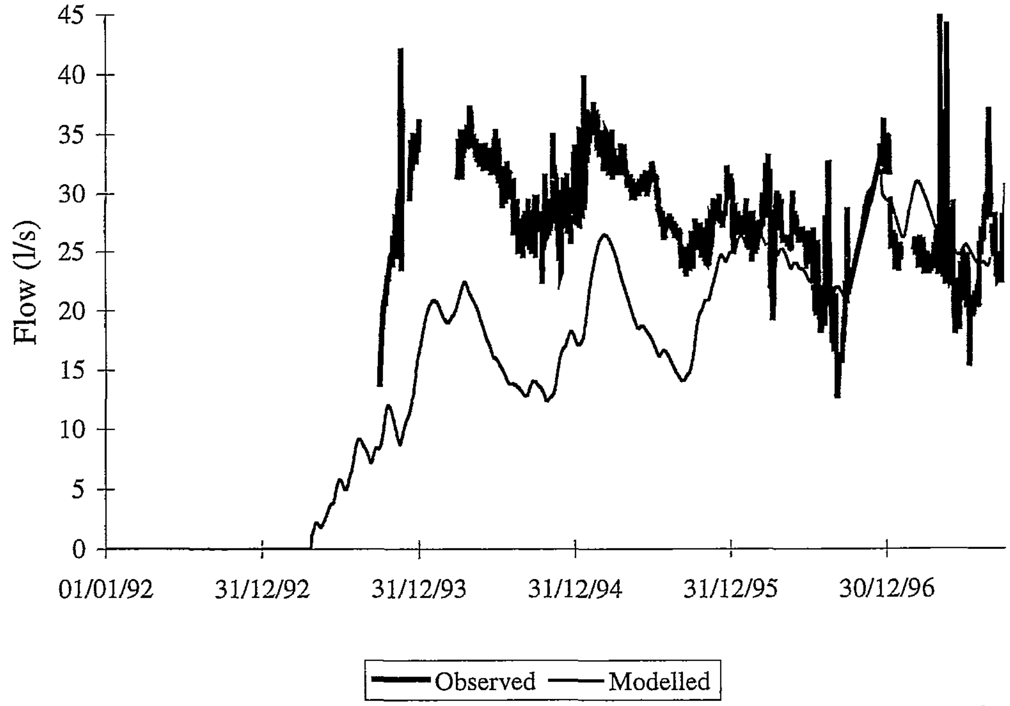
5.7. Ynysarwed – Blaenant Case Study

The polluting discharge from the former Lower Ynysarwed Colliery in South Wales is well documented and has already been discussed in Chapter 4 above. SHETRAN has been applied to the simulation of the polluting discharge from the old mine adit, (SN 809019) which flows into the Neath Canal.

The discharge first emerged unexpectedly in early1993. British Coal had previously predicted that the barrier of intact coal left between the flooded Lower Ynysarwed workings and the de-watered workings of East Blaenant colliery would prevent the emergence of mine water from the adit, once East Blaenant was allowed to flood up. When SHETRAN was applied to the connected mines, the results clearly showed that when East Blaenant workings became flooded to an elevation above the adit entrance (19.4 m A.O.D), minewater began to flow, across the barrier, in the Pennant Sandstone measures above the unworked coal, into the Lower Ynysarwed workings, augmenting the adit discharge. British Coal had predicted that the mine water would emerge at Cefn Coed shaft in the Blaenant system, the top of which is at 76m A.O.D., where it was planned to hydraulically control the mine water (Younger 1995d).

The VSS-NET component has been used to simulate flow in the major roadways of Lower Ynysarwed. Fig 5.7 shows the modelled and observed discharges from the adit for the time period January 1992 to August 1997. The observed discharge indicates a flashy response to rainfall compared to the smoother modelled discharge. This indicates that there may be preferential flow pathways between the surface and the workings, which have not been incorporated in the model. These pathways could be caused by mining-induced fracturing in the Pennant Sandstone measures above the worked No. 2. Rhondda Seam. However, the model appears to have correctly predicted the time at which the mine water emerges at the adit (the Environment Agency reported this to be February to March 1993), within reasonable error bounds. The results of this case study were improved by using the initial water levels predicted by the GRAM modelling exercise (see Chapter 4), and the hydraulic conductivities of the Pennant Sandstone strata were calibrated in order to produce a "best-fit" to the observed flow. The fit appears to improve with time, and from late 1995, observed and simulated flows closely match.

Figure 5.7. Modelled Discharge from Ynysarwed Adit 1992-7



6. PREDICTING MINE WATER QUALITY EVOLUTION

6.1 A Selective Overview of Mine Water Chemistry

The chemistry of mine waters is a subject of sufficient scope that it could fill several volumes on its own. Hence it is proposed here only to review the essentials which are required to understand and manage mine water rebound problems, with an emphasis on simple techniques for predicting the quality of future discharges. For more thorough treatment of the subject in general, and a gateway into the wider literature, the reader is referred to recent developments and reviews presented by Younger (1995a, 1999) and Banks *et al* (1997).

A few key points must be emphasised before going on to discuss the evolution of mine water quality during and after rebound. The first key point relates to the acidity of mine waters. Notwithstanding the popularity of the term "acid mine drainage", the majority of polluting discharges from deep mines are in fact net-alkaline. By "net-alkaline" we mean that the water has an excess of total alkalinity over total acidity. In the contrary case, a water is "net-acidic" if the total acidity exceeds the total alkalinity (Hedin *et al*, 1994).

Acidity and alkalinity are not mutually exclusive properties of a water, for they relate to different (albeit interacting) components of the dissolved load. Total alkalinity relates to the ability of a given water to resist the lowering of pH if strong acid is added to the solution. In most mine waters this property is dominated by the activity of the bicarbonate ion (HCO_3^-). By contrast, the total acidity (i.e. an expression of the ability of the water to resist raising of the pH if strong alkali is added) is dominated by the activities of a number of metals (Fe, Mn, Zn, Al etc) which have the tendency to form hydroxide solids (e.g. $Fe(OH)_3$ in the presence of free hydroxyl ions (OH^-). These are the very metals which tend to render mine waters ecotoxic. The precipitation of these hydroxides has the tendency to release protons (H^+), thus raising pH. For instance:

$$Fe^{3+} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+ \qquad (6.1)$$

Hence the total acidity of a water with many tens of mg/l of such metals bears little relation to its pH, for the absolute concentrations of H^+ ions only approach mg/l levels at the lowest pH values.

Armed with this information, it is possible to calculate the total acidity of a mine water using the following expression (modified after Hedin *et al*, 1994):

$$Acidity_{calc} = F\,.\,[2Fe^{2+}/56 + 3Fe^{3+}/56 + 3Al/27 + 2Mn/55 + 2Zn/65 + 1000(10^{-pH})] \qquad (6.2)$$

where all metal values are in mg/l. If the factor F is set to 1, $Acidity_{calc}$ will be in meq/l. If it is set to 50, then $Acidity_{calc}$ will be in "mg/l as $CaCO_3$". Hedin *et al* (1994) show that acidities calculated in this manner correlate very closely with corresponding titrated acidities.

Acidity is generated in mine waters by the well-known sequence of reactions associated with oxidation of pyrite (FeS_2). Overall, these reactions can be summarised as follows:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O \rightarrow Fe(OH)_3 + 2\ SO_4^{2-} + 4\ H^+ \qquad (6.3)$$

As Appelo and Postma (1993) correctly note, this is one of the most prolific acid-generating reactions in nature. In practice it is rare for the entire process illustrated by equation 6.3 to go to completion at any one site. For instance, the oxidation of pyrite frequently leads to one of two situations:

(i) dissolved ferrous iron (Fe^{2+}) is present in solution, and/or

(ii) intermediate solid phases (ferrous-ferric hydroxysulphate minerals, such as römerite, coquimbite, copiapite etc), which are commonly termed "acid-generating salts" (Bayless and Olyphant, 1993), replace all or part of the pyrite crystals under vadose weathering conditions. When their localities are eventually flooded (either during seasonal water table rise in shallow systems, or during the rebound process), these acid-generating salts rapidly dissolve and greatly elevate the acidity of the mine waters.

Subsequent transport of the dissolved ferrous iron and other components of acidity to the Earth's surface presents opportunities for ferrous iron to be oxidised to the ferric form (Fe^{3+}), which can then hydrolyse according to Equation 6.1 to form the familiar orange or red "ochre" (ferric hydroxide or oxyhydroxide) which blankets the beds of streams which receive mine water discharges.

Net-acidic mine waters arise where three factors are satisfied:

(i) there is pyrite available for oxidation

(ii) the pyrite is exposed to atmospheric oxygen for at least long enough to generate Fe^{3+} ions (which can then further oxidise pyrite in the absence of oxygen)

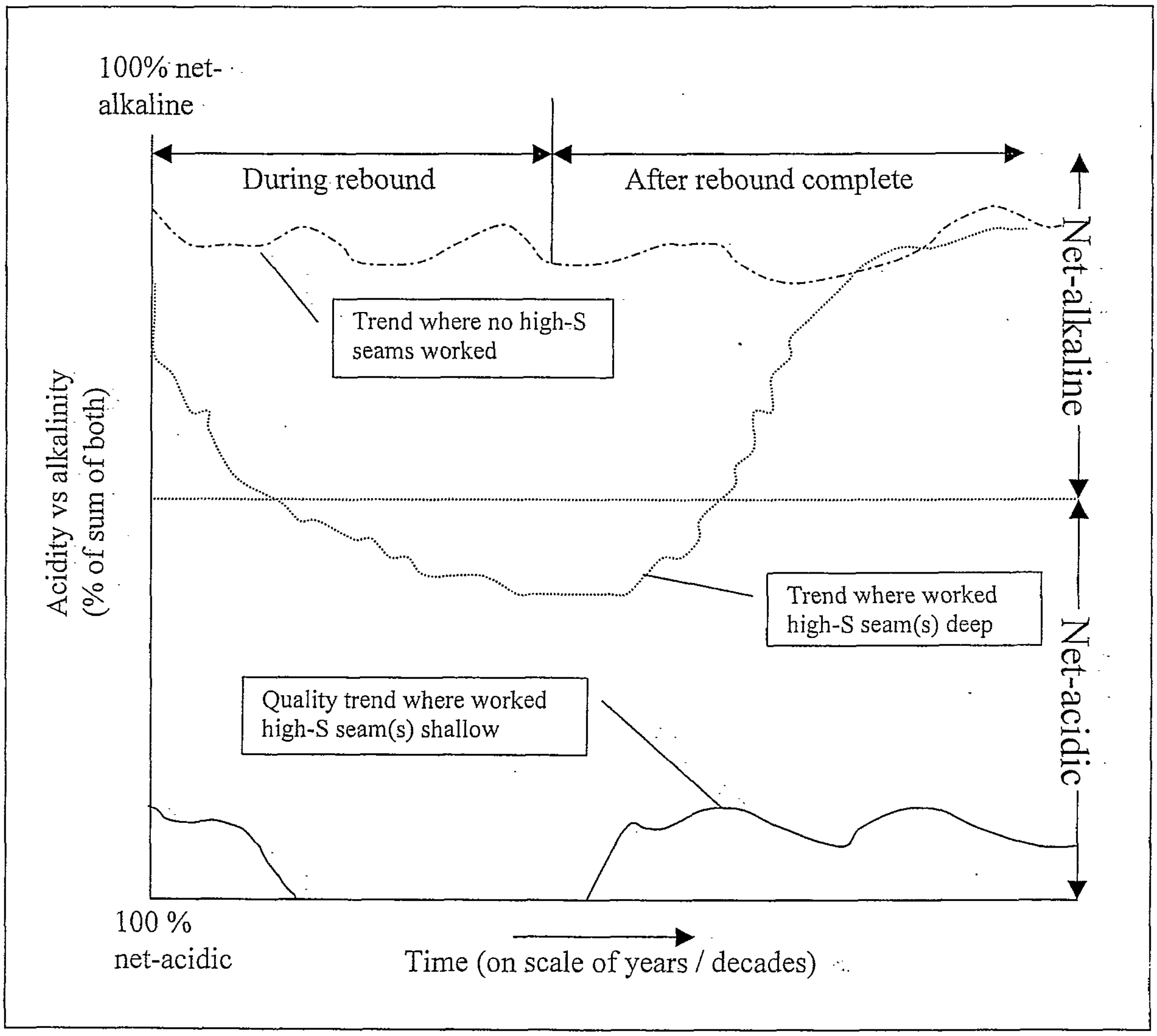
(iii) there is moisture available, both to participate in the oxidation reaction (including the formation of acid-generating salts under vadose conditions), and to transport the dissolved products away from the mineral surfaces.

The availability of pyrite for oxidation can be assessed by determining the total sulphur content (usually expressed in weight %) of a seam and associated strata. Where this has been done, it has always been found that pyrite availability varies widely from one seam to another. The main geological control on this variation is degree to which marine waters were involved in the diagenesis of coal-bearing strata, with brackish water facies having the highest pyrite content, marine facies the next highest, and terrestrial facies the lowest (e.g. Caruccio and Ferm, 1974; Morrison *et al.*, 1990; Younger, 1994, 1999; Younger *et al.*, 1995). The variations in total sulphur between seams are often persistent across entire coalfield areas, reflecting the widespread nature of marine incursions during sedimentation and early diagenesis. As an example of total sulphur variations in one coalfield, Table 6.1 summarises average values for the principal seams of the South Yorkshire Coalfield. All other factors being equal, therefore, one would expect the Parkgate seam to be a more prolific acid-generator than the other seams.

Table 6.1. Total sulphur contents of seams in the Rotherham area (after Eden *et al*, 1957 and Burke, 1997).

Seam	Total Sulphur (weight %)
High Hazel Seam	1.3
Barnsley Seam	0.8
Swallow Wood Seam	1.0
Parkgate Seam	2.1
Silkstone Seam	0.9

Figure 6.1. Schematic trajectories in acidity-alkalinity balance during and after mine water rebound for three different hydrogeological settings.



In reality, all other factors rarely are equal. For instance, the process of mine water rebound alters the circumstances attending the second two factors. For instance, if voids are flooded, the total acidity of the mine water may increase initially as acid-generating salts in the strata are dissolved, but further pyrite oxidation will tail off in the deeper parts of the system as atmospheric oxygen is excluded by the saturation of voids. On the other hand, the increased quantity of water in the voids provides enhanced migration potential for contaminant metals. The interplay of these factors results in dynamic changes in mine water quality during and after rebound. In particular, the overall balance of acidity and alkalinity may follow a variety of trajectories over time according to the local hydrogeological setting. Figure 6.1 illustrates the three possible trajectories.

As suggested by Figure 6.1, if high-sulphur seams are present in the worked sequence, then a trend towards increased net-acidity is often observed during rebound. After rebound, this trend will slowly reverse (usually showing an exponential decline over time), and (except where there are very shallow workings in the highest-sulphur strata), the mine water will tend towards net-alkalinity within a time-period of not more than 4 decades after surface discharge first commences. Numerous examples of this type of behaviour for mined systems throughout Britain have been documented and discussed by Younger (1997a) and Wood *et al* (1999). Two examples will suffice here. At Wheal Jane, the trend of decreasing iron (and therefore decreasing total acidity) over the first few years after completion of rebound was as shown on Figure 6.2. However, as the worked strata at Wheal Jane extend to ground surface, and had phenomenally high total sulphur contents of as much as 15%, pyrite oxidation occurs in the (post-rebound) zone of water table fluctuation. Consequently the mine water remains net-acidic, and is unlikely ever to achieve net-alkalinity. By contrast, Figure 6.3 shows the trends in pH and total iron concentrations for the Garth Tonmawr coal mine discharge in the Pelenna catchment, South Wales. In this case, the scope for post-rebound pyrite oxidation is rather more limited, and the water has long-since reverted to being net-alkaline.

In summary, the most polluted mine waters tend to be associated with the strata richest in pyrite, and where mined pyrite-rich strata are present in the shallow subsurface, mine water discharges may remain net-acidic in perpetuity. In the far more common case that only a few mined seams had elevated pyrite contents, and these seams are present deep below ground level, the discharge may be net-acidic for a limited period of time (not greater than 40 years), after which it will become net-alkaline once more. Where there are *no* high-sulphur seams in the worked sequence, a mine water may remain net-alkaline throughout the rebound process and ever after.

Having noted that the majority of "mature" mine water discharges (i.e. those which have been flowing for several decades) are net-alkaline, it should be noted immediately that "net-alkaline" does not equate with "non-polluting". Numerous examples could be given of mature, net-alkaline mine waters which contain up to 40 mg/l total iron, and which can therefore cause serious degradation of all but the largest receiving water courses. In fact a dissolved iron concentration in excess of 0.5 mg/l is sufficient to cause conspicuous staining of a streambed, and concentrations in excess of 1 mg/l can lead to smothering of the benthos with ochre. This can prevent photosynthesis and thus cause severe ecological degradation (Jarvis and Younger, 1997). Some net-alkaline mine waters also contain elevated concentrations of manganese, which is not considered ecotoxic, but can be problematic in terms of possible water supply use of a given water course. Less frequently, zinc can occur at concentrations up to 30 mg/l in net-alkaline mine waters, and in the North Pennine Orefield

Figure 6.2. Temporal trend in total iron concentration for the Wheal Jane mine water in the years immediately following the completion of rebound in January 1992 (after Younger, 1997a).

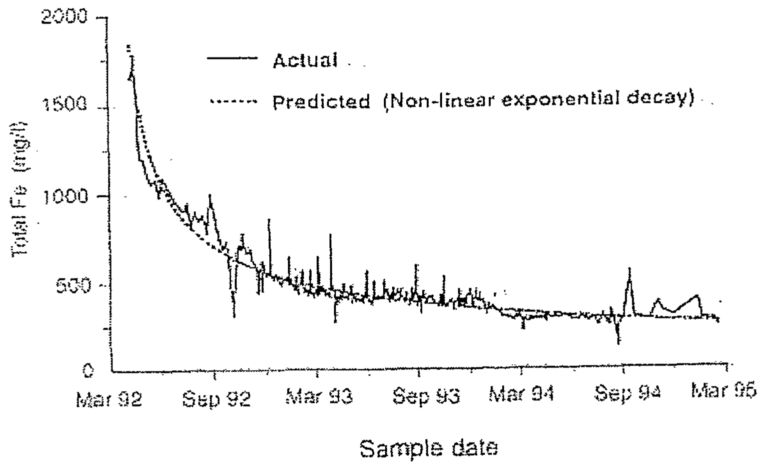
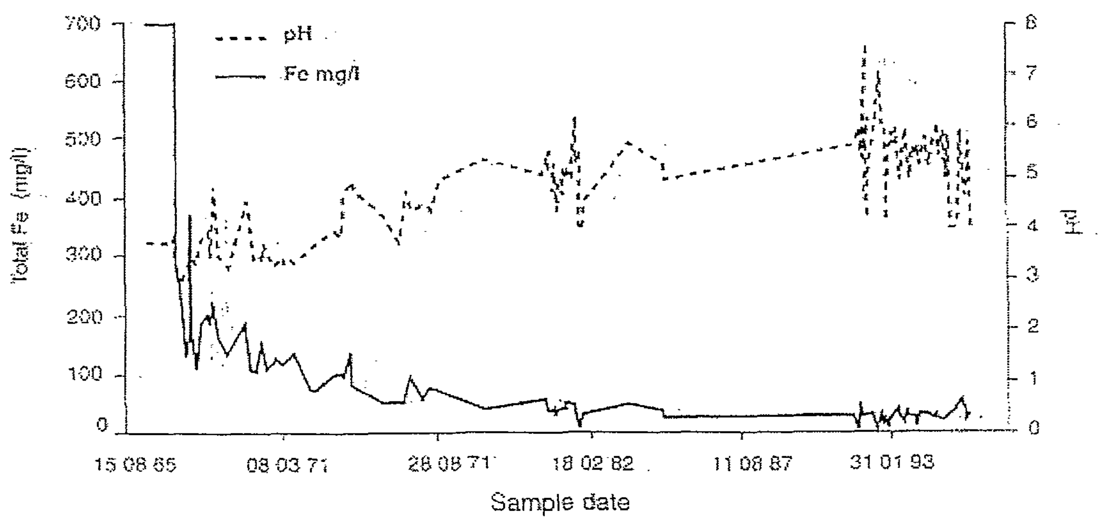


Figure 6.3. Temporal trends in pH and total iron for the Garth Tonmawr discharge, South Wales, from the time of first emergence in the 1960s into the 1990s (after Younger, 1997a).



there are numerous mature mine waters which contain 5 – 10 mg/l of Zn (derived from dissolution of sphalerite, ZnS) even though dissolved iron is less than 1 mg/l. Zinc-rich mine waters are often net-alkaline even during rebound, since the ZnS oxidation reaction is *not* acid-generating:

$$ZnS + 2O_2 \rightarrow Zn^{2+} + SO_4^{2-} \qquad (6.4)$$

In the course of flow through surrounding strata, contaminated mine waters may be subject to chemical reactions which improve their quality. The most obvious example would be flow of net-acidic mine water into a limestone aquifer, where calcite dissolution can neutralise the acidity, and provoke precipitation of iron as a hydroxide coating on the limestone. In Section 3.3.2, the case of Mainsforth Colliery was discussed, in which mine water rebound in the Coal Measures from 1975 to 1983 caused a rise in groundwater levels in the overlying Magnesian Limestone aquifer, accompanied by a migration of mine water into the aquifer. Table 6.2 records the resultant changes in groundwater chemistry in the Magnesian Limestone aquifer. As expected, buffering of pH by limestone dissolution neutralised the acidity of the mine water, and also precluded persistence of elevated iron concentrations in the limestone groundwater. However, limestone dissolution alone does nothing to prevent an overall rise in total mineralisation of the limestone groundwater, which is sufficient to render the water unsuitable for public supply use in that area.

Table 6.2. Changes in selected chemical parameters in the Magnesian Limestone aquifer near Mainsforth, County Durham, 1975 to 1983. (Data from EA archives).

Year:	**1975**	**1976**	**1977**	**1978**	**1979**	**1980**	**1981**	**1982**	**1983**
SO4 (mg/l):	50	40	80	197	270	340	377	463	491
Cond. (μS/cm):	536	715	705	818	910	970	1150	1250	1400
pH:	7.9	7.7	7.6	7.5	7.7	7.6	7.7	7.6	7.7

6.2. Predicting Mine Water Quality

The approach to mine water quality prediction presented below draws upon the concepts outlined above, supported by interpretation of water quality records and geological data for more than one hundred existing mine water discharges around Britain. The details of these interpretations are presented in a series of reports by Younger and Adams (1996a,b, 1997a,b, 1998). More elaborate, site-specific assessments can be made using one or other of the "acid-base accounting" (ABA) techniques which have been developed in North America in connection with surface mining permit applications (see, for instance, Morrison *et al.*, 1990). These methods are applicable only where rock samples are available. The meaning and reliability of many ABA techniques are the subjects of sustained controversy in the US literature. However, they are not really suited to predicting quality in complicated mine water rebound situations in deep mines, and are thus not considered further in this report.

Here, the aim is to briefly present a practical protocol for predicting mine water quality which represents our current best judgement, as informed by 6 years of intensive research. The focus of the predictive methods outlined below is total iron concentration. While concentrations of other parameters (such as Al concentrations, pH etc) will certainly be of interest in *some* cases, experience shows that iron concentrations will be a major concern in virtually *all* cases. Hence the focus on iron is justified in a general work such as this.

In predicting iron concentrations, it is important to remember that we are concerned here with flooded deep mines, not spoil heaps or shallow drift mines which never flood in the manner of deep workings. Since active pyrite oxidation is far more common in spoil heaps and shallow drift mines than in flooded deep mines, long-term iron concentrations may be higher in spoil and drift mines than in flooded deep workings. On the other hand, because it is only in deep mines that rebound can take acid-generating salts from such a large volume of workings into solution at once, the initial peak concentrations of iron in deep mine discharges are often considerably higher than those associated with spoil and drift mines.

Supposing that the typical trend in water quality for a problematic mine water discharge will be similar to those shown in Figures 6.2 and 6.3 (and Younger, 1997a, provides abundant evidence to suggest that this supposition is reasonable), then it would be helpful to be able to predict at least the following parameters:

- peak iron concentration
- half-life of decline in iron concentration
- long-term iron concentration after the decline is complete.

Let us deal with each of these in turn.

(i) Predicting peak iron concentration.

A review of deep coal mine water quality data world-wide reveals that the maximum iron concentration ever recorded after rebound is around 1500 mg/l. This was the peak concentration recorded at, for instance, Dalquharran Colliery in Scotland (the UK's worst recorded deep mine water discharge) and at the Omega Mine, Morgantown, West Virginia (which the US Bureau of Mines state is the worst post-rebound quality they have ever encountered in the Appalachian coalfields; W. Aljoe, personal communication, 1995). This rather suggests that a maximum of 1500 mg/l can be reliably argued to provide the absolute upper-bound estimate for total iron in a newly-rebounded coal mine discharge.

Most metals mines will not produce such high iron concentrations, save where (as at Wheal Jane) they worked particularly pyritic ore bodies. The case of Wheal Jane provides a possible upper-bound estimate for post-rebound peak iron concentrations in a mine working a highly pyritic ore body. Concentrations as high as 2500 mg/l were recorded immediately after rebound was complete.

Most mines of either type will not produce peak iron concentrations anything like as high as these upper limits. From the limited data available, it would appear that the peak iron concentration in most abandoned mine discharges is around 10 times greater than the long-term iron concentration. Thus a mine which has around 40 mg/l in the long-term probably displayed peaks up to 400 mg/l in the first few years after emergence. As the prediction of long-term quality is covered below, it may be feasible to estimate peak iron concentration by multiplying the figure obtained using the methods described below by 10.

Further guidance on possible peak iron concentrations is provided by consideration of the total sulphur contents of the worked strata. Although not thoroughly supported by statistical analysis, consideration of the available data suggests the rough relationships between total sulphur content of the principal worked seams and peak post-rebound iron concentration shown in Table 6.3.

Table 6.3. Peak post-rebound iron concentration versus seam total sulphur content.

Total sulphur content (wt %)	Peak iron post-rebound (mg/l)
0 - 1	1
1 – 2	200
2 - 4	500
> 4	1500

It should be emphasised that where a multiple-seam sequence has been worked, the post-rebound quality will not simply reflect the total sulphur content of the most sulphur-rich seam, but will be a mixture reflecting the contributions of less sulphur-rich seams also.

(ii) Predicting half-life of decline in iron concentration

The half-life of the decline in iron concentration (after the peak post-rebound iron concentration has been reached) is controlled by hydrogeological factors, chief amongst which are:

- the extent of the flooded mine system
- the rate of recharge to the system
- the tortuosity of the flow-paths to the discharge point (which governs mechanical mixing, and thus dispersion)

As these parameters vary dramatically from one system to the next, general guidance is difficult to provide. However, Glover (1983) suggested that the half-life of a given mine system is generally equal to the total time period which was required for rebound. This suggestion is intuitively appealing, for the rate of rebound is clearly controlled by the extent of the floodable voids and the rate of recharge. Though few complete data-sets are publicly available to allow assessment of this suggestion, the record for Wheal Jane (Figure 6.2) suggests that the half-life was very close to the length of the total rebound period (10 months). By contrast, the Ynysarwed-Blaenant system has thus far shown a half-life on the order of 3 years, whereas the rebound period was around 2 years in length. This attenuation of the half-life beyond that predicted by the method of Glover (1983) could be attributed to the high tortuosity of the inter-connected mined system at Ynysarwed, which is far more complex than the single linear lode structure worked at Wheal Jane. Nevertheless, it would seem that the Glover (1983) method does provide an estimate which is of the right magnitude, and which would serve for most practical planning purposes.

(iii) Predicting the long-term iron concentration after decline is complete

At its simplest, this task might be accomplished by simply re-writing Table 6.3 as follows:

Table 6.4.: Long-term iron concentrations in mature mine water discharges versus seam total sulphur content.

Total sulphur content (wt %)	Long-term iron conc. (mg/l)
0 – 1	0.1
1 – 2	20
2 – 4	50
> 4	150

The values in the right-hand column are intuitively appealing, as they neatly span the range of values observed in mature mine water discharges around the UK.

To provide a little more guidance on long-term iron concentrations, a detailed analysis was made of the hydrogeological settings of 81 major, mature mine water discharges in Wales, north-east England and Scotland for which reliable mean iron concentrations were available. This analysis was intended to test the hypothesis that the two most important geological controls on long-term mine water quality are:

- the distance from the mine water discharge point to the closest outcrop of the shallowest seam in which extensive workings were made by the relevant mine. The reasoning behind this is that discharges distant from outcrop are less likely to be prone to active oxidation of pyrite near the discharge than discharges right on the seam outcrop, which can be expected to be located close to unsaturated voids in which pyrite oxidation is active. While it might seem necessary to consider the relative topographic positions of the discharge and the seam outcrop, it has been found in practice that very few discharges occur up-dip from the lowest point on the outcrop of the shallowest worked seam: the vast majority of discharges lie down-dip from the seam outcrop.

- the thickness of strata between the shallowest-worked seam and the nearest "marine bed" in the sequence. (This "marine bed" might be a shale with a marine fossil assemblage, a brackish-water shale containing Lingula sp., or a marine limestone). As noted in Section 6.1, number of earlier workers (eg Caruccio and Ferm, 1974; Morrison et al, 1990; Younger, 1994) have noted that the pyrite content of marine-influenced coal-bearing strata tends to be higher than that of strata which accumulated under terrestrial conditions. Consequently, oxidation of "marine beds" and adjoining coals can be expected to result in more severe pollution than would be the case with terrestrial strata (Caruccio and Ferm, 1974).

The analysis of data for the 81 existing mine water discharges largely bore out this hypothesis. On this basis, it has proved possible to proffer the following "rules":

- The long-term iron concentration is only likely to exceed 20 mg/l (and the pH is only ever likely to be lower than 6.5) where the mine water discharge lies within half of a kilometre of the outcrop of the shallowest worked seam.

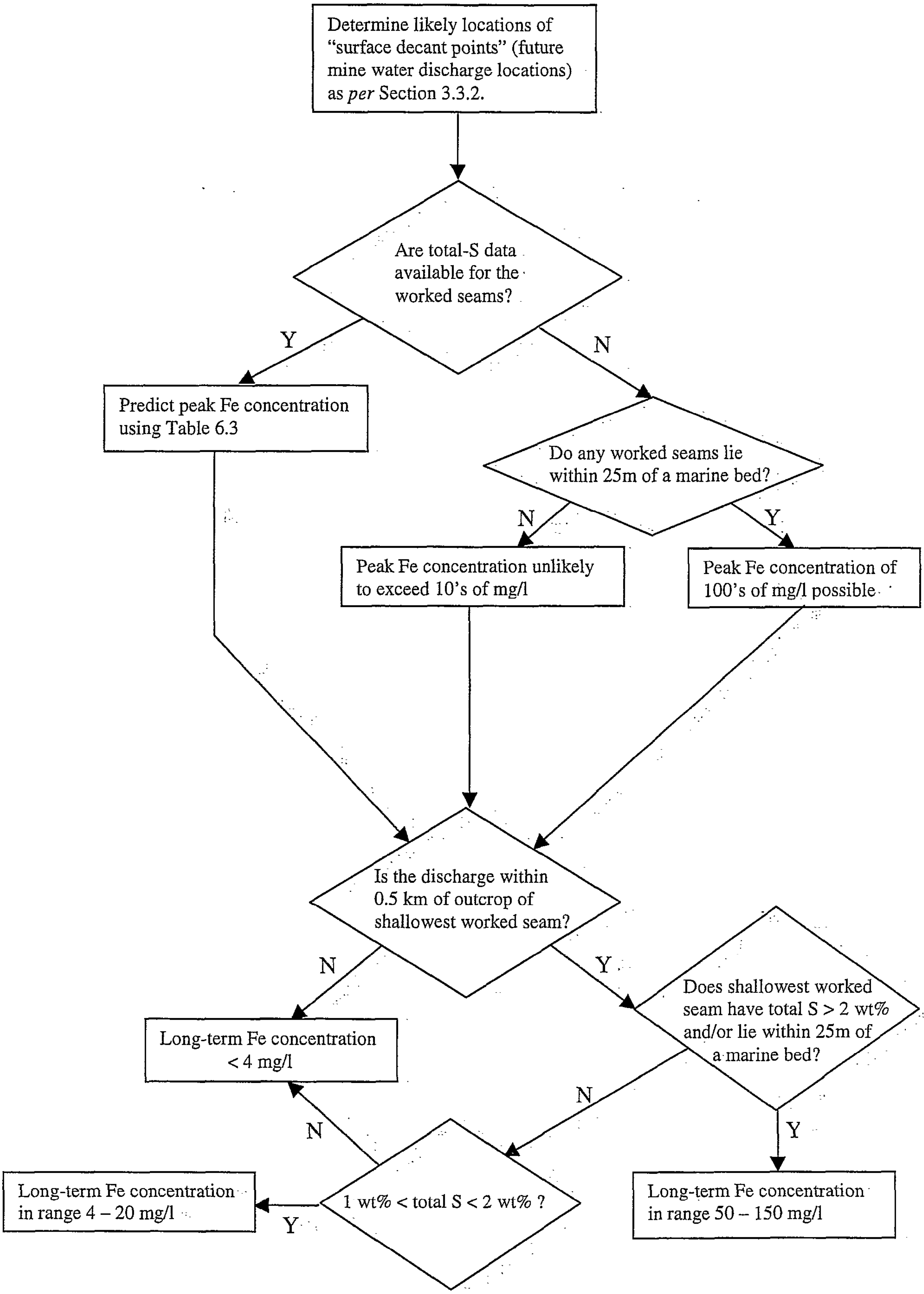
- Even within half of a kilometre of outcrop, the long-term iron will generally be less than 50 mg/l

- Long-term iron concentrations greater than 100 mg/l are only likely where the discharge is within half of a kilometre of the outcrop of the shallowest worked seam AND a marine bed lies within 25m (stratigraphically above or below) this seam.

- Where the nearest marine bed in the sequence is more than 80m stratigraphically above or below the shallowest worked seam, the total iron will not exceed 4 mg/l in the long-term, and will usually be less than 1 mg/l, no matter how close the discharge lies to the seam outcrop.

Figure 6.4 synthesises all of the above observations into a single set of guidelines in flow-chart form. It is recommended that this be used in conjunction with the methodology for predicting the likely locations of mine water discharges (i.e. "surface decant points") presented in Section 3.3.2.

Figure 6.4. Water quality prediction protocol for future mine water discharges.



REFERENCES

Adams, R., and Younger, P.L., 1997, Simulation of groundwater rebound in abandoned mines using a physically based modelling approach. *In* Veselic, M., and Norton, P.J., (Editors), Proceedings of the 6th International Mine Water Association Congress, "Minewater and the Environment", held Bled, Slovenia, 8 - 12th September 1997. Volume 2, pp 353 - 362.

Adams, R., Dunn, S.M. and O'Connell, P.E. (draft paper). Application of the NUARNO macromodel to investigate land use change.

Aldous, P.J., and Smart, P.L., 1988, Tracing ground-water movement in abandoned coal mined aquifers using fluorescent dyes. Ground Water, Vol. 26, pp 172 - 178.

Aljoe, W.W., and Hawkins, J.W., 1994, Application of aquifer testing in surface and underground coal mines. *In* Proceedings of the 5th International Mine Water Congress, Nottingham (UK), September 1994. Vol. 1, pp. 3 - 21.

Appelo, C.A.J., and Postma, D., 1993, Geochemistry, Groundwater and Pollution. A.A. Balkema Publishers, Rotterdam. 536pp.

Aston, T.R.C., and Whittaker, B.N., 1985. Undersea longwall mining subsidence with special reference to geological and water occurrence criteria in the North-East of England Coalfield. Mining Science and Technology, 2: 105 - 130.

Awberry, H.G., 1988. The protection of the Nottinghamshire Coalfield by the Bentinck Colliery minewater concentration scheme. International Journal of Mine Water, 7: 9 - 24.

Banks, D., Younger, P.L., and Dumpleton, S., 1996, The historical use of mine-drainage and pyrite-oxidation waters in central and eastern England, United Kingdom. Hydrogeology Journal, 4, (4), pp. 55 - 68.

Banks, D., Younger, P.L., Arnesen, R.-T., Iversen, E.R., and Banks, S.D., 1997, Mine-water chemistry: the good, the bad and the ugly. Environmental Geology, **32**, (3), pp. 157 - 174.

Bayless, E.R., and Olyphant, G.A., 1993, Acid-generating Salts and their Relationship to the Chemistry of Groundwater and Storm Runoff at an Abandoned Mine Site in Southwestern Indiana, USA. Journal of Contaminant Hydrology, Vol. 12, pp 313 - 328.

Bowen, G.G., Dussek, C., and Hamilton, R.M., 1998, Pollution resulting from abandonment and subsequent flooding of Wheal Jane Mine in Cornwall, UK. *In* Mather, J., Banks, D., Dumpleton, S., and Fermor, M., (editors), Groundwater contaminants and their migration. Geological Society Special Publication No 128. Geological Society, London. pp. 93 – 99.

Buckley, J.A., 1992, The Cornish Mining Industry - A Brief History. Tor Mark Press, Penryn. 48pp.

Burke, S.P., 1997, An assessment and prediction of minewater movement at the abandoned collieries Treeton, Thurcroft and Silverwood. Unpublished MSc Thesis, Department of Civil Engineering, University of Newcastle. 92pp.

Burke, S.P., and Younger, P.L., 1999, Groundwater Rebound in the South Yorkshire Coalfield: A First Approximation using the GRAM Model. Quarterly Journal of Engineering Geology. *(in press)*

Burrell, R., and Friel, S., 1996, The effect of mine closure on surface gas emissions. In Proceedings of the IBC UK Conference on "The Environmental Management of Mining Operations", held 23rd - 24th September 1996, London, UK. 15pp.

Cain, P., D.J. Forrester, and R. Cooper, 1994, Water inflows at Phalen Colliery in the Sydney Coalfield, and their relation to interaction of workings. *In:* Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 1, pp 83 - 95.

Cairney, T., and Frost, R.C., 1975, A case study of mine water quality deterioration, Mainsforth Colliery, County Durham. Journal of Hydrology , 25, pp.275 - 293.

Calder, J.W., 1973, Inrush at Lofthouse Colliery, Yorkshire. Report on the cause of, and circumstances attending, the inrush which occurred at Lofthouse Colliery, Yorkshire, on 21 March 1973. Department of Trade and Industry (Cmnd 5419). HMSO, London. 26pp. plus plans.

Calver, A. 1997. Recharge response functions. Hydrology and Earth Systems Sciences, 1, pp 47-53.

Caruccio, F.T., and Ferm, J.C., 1974, Paleoenvironment - Predictor of Acid Mine Drainage Problems. Proceedings of the 5th Coal Mine Drainage Research Symposium, National Coal Association (USA), Kentucky. pp 5 - 9.

Chadwick, A.J. and Morfett, J.C., 1993, Hydraulics in Civil Engineering. Allen & Unwin, London. 557pp

Chen, M.,. Soulsby, C., and Younger, P.L., 1997, Predicting water quality impacts from future minewater outflows in an urbanized Scottish catchment. *In* Chilton, P.J., *et al*, (Editors), Groundwater in the urban environment, Volume 1: Problems, processes and management. (Proceedings of the XXVII Congress of the International Association of Hydrogeologists, Nottingham, UK, 21 - 27 September 1997). A.A. Balkema Publishers, Rotterdam. pp 383 - 388.

Clarke, A.M., 1962. Some structural, hydrological and safety aspects of recent developments in south-east Durham. The Mining Engineer, 122: 209 - 231.

Coldewey, W.G., and L. Semrau, 1994. Mine water in the Ruhr area (Federal Republic of Germany). *In:* Proceedings of the 5th International Mine Water Congress, Nottingham, UK, 18 - 23rd September 1994. Volume 2: 613 - 629. 1994.

Colebrook, C.F. 1939. Turbulent flow in pipes, with particular reference to the transient region between the smooth and the rough pipe laws. Journal of the Institution of Civil Engineers Vol 11, pp 133-156.

Cook, N.G.W., 1982. Ground-water problems in open-pit and underground mines. In: Narasimhan, T.N., (Editor), Recent trends in hydrogeology. Geological Society of America, Special Paper 189. Boulder, Colorado. pp 397 - 405.

Cullen, J.J., and LaFleur, R.G., 1984, Theoretical considerations on simulation of karstic aquifers. *In* LaFleur, R.G., (Editor), Groundwater as a Geomorphic Agent. Allen & Unwin, Boston. pp 249 – 280.

Doyle, A., 1997, Sacrifice, achievement, gratitude. Images of the Great Northern Coalfield in decline. County Durham Books, Durham. 115pp.

Dunham, K.C., 1990, Geology of the Northern Pennine Orefield, Volume 1 Tyne to Stainmore (2nd Edition). Economic Memoir of the British Geological Survey, sheets 19 and 25, and parts of 13, 24, 26, 31, 32 (England and Wales). HMSO, London. 299pp.

Duckham, H., and Duckham, B.F., 1973, Great pit disasters: Great Britain 1700 to the present day. David and Charles, Newton Abbot. 227pp.

Dumpleton, S., & Glover, B.W., 1995, The impact of colliery closures on water resources with particular regard to the NRA Severn-Trent Region. British Geological Survey Technical Report WD/95/40. 44pp.

Ewen, J., Parkin, G. and O'Connell, P.E. (draft paper). SHETRAN: a coupled surface/subsurface modelling system for 3D water flow and sediment and contaminant transport in river basins.

Featherstone, R.E., and Nalluri, C., 1982 Civil engineering hydraulics: essential theory with worked examples. Granada, London, 371pp.

Ferry, J., Jeffery, R.I., Lloyd, J.W., and Stopher, A., 1994, Hydrogeological and hydrochemical problems relating to the Smotherfly opencast coal site: An integrated approach to the granting of a discharge licence. Proceedings of the 5th International Minewater Congress, Nottingham, UK. Vol. 1, pp 401 - 427.

Ford, D.C., and Williams, P.W., 1989, Karst geomorphology and hydrology. Chapman and Hall, London. 601pp.

Freeze, R.A. and Cherry, J.A., 1979, Groundwater. Prentice-Hall, Englewood Cliffs, NJ. 604pp

Frost, R.C., 1979, Evaluation of the rate of decrease in the iron content of water pumped from a flooded shaft mine in County Durham, England. Journal of Hydrology, 40, pp 101 - 111.

Glover, H.G., 1983, Mine water pollution -- an overview of problems and control strategies in the United Kingdom. Water Science and Technology, 15, pp 59 - 70.

Gupta, P.K., and Singh, T.N., 1994. Water resource characterization of underground coal working - a case study for part of the Jharia Coal Field :In: Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 1: 97 - 104.

Grmela, A., and Tylcer, J., 1997, Determination of coal mines water balance. *In* Veselic, M., and Norton, P.J., (Editors), Mine Water and the Environment. Proceedings of the 6th International Mine Water Association Congress, held in Bled, Slovenia, 8 - 12 September 1997. Volume 1, pp 35 - 42.

Hamilton, R.M., Bowen, G.G., Postlethwaite, N.A., and Dussek, D.J., 1994, The abandonment of Wheal Jane, a tin mine in SW England. . In: Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 2: pp 543 - 551.

Hanna, R.B., and Rajaram, H., 1998, Influence of aperture variability on dissolutional growth of fissures in karst formations. Water Resources Research, 34, pp 2843 – 2853.

Harrison, R., Scott, W.B., and Smith, T., 1989, A Note on the Distribution, Levels and Temperatures of Minewaters in the Northumberland and Durham Coalfield. Quarterly Journal of Engineering Geology, Vol. 22, pp 355 - 358.

Hartman, H.L., 1987, Introductory mining engineering. Wiley, New York. 633pp.

Hedin, R.S., Nairn, R.W., and Kleinmann, R.L.P., 1994, Passive Treatment of Coal Mine Drainage. US Bureau of Mines Information Circular 9389. US Department of the Interior, Bureau of Mines, Pittsburgh, PA. 35 pp.

Henton, M.P., 1979, Abandoned Coalfields: Problem of Pollution. The Surveyor, Vol. 153, pp 9 - 11.

Henton, M.P., 1981, The problem of water table rebound after mining activity and its effects on ground and surface water quality. In van Duijvenbooden, W., Glasbergen, P., and van Lelyveld, H., (Editors), Quality of Groundwater. (Proceedings of an International Symposium, Noordwijkerhout, The Netherlands, held 23 - 27th March 1981). Elsevier, The Netherlands. pp 111 - 116.

Jarvis, A.P., and Younger, P.L., 1997, Dominating chemical factors in mine water induced impoverishment of the invertebrate fauna of two streams in the Durham Coalfield, UK. Chemistry and Ecology, 13, pp 249 - 270.

Kesserû, Z., 1995. New approaches and results on the assessment of risks due to undermining for mines safety and for protection of water resources. In: Proceedings of the International Conference on "Water Resources at Risk", held by the American Institute of Hydrology and the International Mine Water Association, Denver, Colorado, May 14th - 18th 1995, pp IMWA 53 - IMWA 72.

Knight Piésold and Partners, 1995, Wheal Jane Minewater Study. Environmental Appraisal and Treatment Strategy. Report to the National Rivers Authority South Western Region, Exeter.

Knight Piésold and Partners, 1997, South Crofty Strategic Closure Study. Provisional Report to the Environment Agency, Bodmin. December 1997. 39pp. Plus figures.

Laine, D.M., 1997, The treatment of the pumped minewater discharge at Woolley Colliery, West Yorkshire. *In* Younger, P.L., (Editor), Minewater treatment using wetlands. Proceedings of a National Conference held 5th September 1997, at the University of Newcastle, UK. Chartered Institution of Water and Environmental Management, London. pp 83 - 103..

Lancaster, D., 1995, Delineating the catchment area and flow system of a major polluted spring in County Durham. Unpublished MSc thesis, Department of Civil Engineering University of Newcastle.

Lemon, R., 1991. Pumping and disposal of deep strata mine water. Mining Technology, March 1991: 69 - 76.

Lerner, D.N., Issar, A.S., and Simmers, I., 1990, Groundwater recharge. A guide to understanding and estimating natural recharge. International Contributions to Hydrogeology Volume 8. International Association of Hydrogeologists. Verlag Heinz Heise, Hannover. 345pp.

Llewellyn, K., 1992, Disaster at Tynewydd. An account of a Rhondda mine disaster in 1877. (Second Edition). Church in Wales Publications, Penarth. 92pp.

McDonald, M.G., and A.W. Harbaugh, A modular three-dimensional finite-difference ground-water flow model., United States Geological Survey, Techniques of Water Resources Investigations Report O6-A1. 567pp. 1988.

Middleton, J., and Waltham, T., 1986, The Underground Atlas. A Gazetteer of the World's Cave Regions. Robert Hale Ltd, London. 239pp.

Miller, J.T., and Thompson, D.R., 1974, Seepage and mine barrier width. 5th Symposium – Coal Mine Drainage Research. National Coal Association, Kentucky. pp 103 – 127.

Milne, S., 1994, The Enemy Within: MI5, Maxwell and the Scargill Affair. Verso, London. 344pp.

Minett, S.T., Blythe, D.A., Hallam, G.D., and Hughes, D.B., 1986, Analysis of an Advanced Dewatering Scheme at an Opencast Coal Site in Northumberland. In Cripps, J.C., Bell, F.G., and Culshaw, M.G., (editors), Groundwater in Engineering Geology. Engineering Geology Special Publication No 3, pp 347 - 352. Geological Society, London.

Minett, S.T., 1987, The hydrogeology of parts of the Northumberland and Durham Coalfield related to opencast mining operation. Unpublished PhD Thesis, Newcastle Upon Tyne.

Morrison, J.L., Scheetz, B.E., Strickler, D.W., Williams, E.G., Rose, A.W., Davis, A., and Parizek, R.R., 1990, Predicting the occurrence of acid mine drainage in the Alleghenian coal-bearing strata of western Pennsylvania; an assessment by simulated weathering (leaching) experiments and overburden characterization. In Chyi, L.L., and Chou, C.L., (editors), Recent advances in coal geochemistry. Geological Society of America, Special Paper 248. pp 87 - 99.

NCB, 1975, The subsidence engineers' handbook. National Coal Board, Mining Department, London. 111pp.

NCB, 1977, Surveying Practice. Codes and Rules. (First issued 1973, amended 1975 and 1977). National Coal Board, Mining Department, London. 41pp.

NRA, 1994, Abandoned mines and the water environment. Report of the National Rivers Authority. Water Quality Series No 14. London, HMSO. 46pp.

Orchard, R.J., 1975. Working under bodies of water. The Mining Engineer, 170: 261 - 270.

Parkin, G. 1996, A three-dimensional variably-saturated subsurface modelling system for river basins. PhD Thesis, Department of Civil Engineering, University of Newcastle Upon Tyne, UK. 1996

Paulino, J., F. Pendás & E.M. Casares (1997), Modelling of Groundwater Flow in a Partially Abandoned Mine with MODFLOW. Abstract in: Supplement II to Vol. 15, Annales Geophysicae, Pt. II, Hydrology, Oceans, Atmosphere & Nonlinear Geophysics, 22nd General Assembly of EGS, Vienna. pp C340.

Raistrick, A., and Jennings, B., 1965, A history of lead mining in the Pennines. Longmans, Green and Co Ltd, London. (Reprinted 1983 and 1989 by Davis Books Ltd, Newcastle, and George Kelsall Publishing, Littleborough). 347pp.

Peters, T.W., 1978. Mine drainage problems in north Derbyshire. Mining Engineer, 137: 463 - 474.

Ranson, C.M., and Edwards, P.J., 1997, The Ynysarwed experience: active intervention, passive treatment and wider aspects. *In* Younger, P.L., (Editor), Minewater treatment using wetlands. Proceedings of a National Conference held 5th September 1997, at the University of Newcastle, UK. Chartered Institution of Water and Environmental Management, London. pp 151 - 164.

Ranson, C.M., Reynolds, N., and Smith, A.C., 1998, Minewater treatment in Neath and Port Talbot. *In* Fox, H.R., Moore, H.M., and McIntosh, A.D, (editors), Land Reclamation: Achieving Sustainable Benefits. Balkema, Rotterdam. pp 497 – 507.

Reddish, D.J., Yao, X.L., Benbia, A., Cain, P., and Forrester, D.J., 1994. Modelling of caving over the Lingan and Phalen mines in the Sydney Coalfield Cape Breton. In: Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 1, pp 105 - 124.

Robins, N.S., 1990, Hydrogeology of Scotland. HMSO for British Geological Survey, London. 90pp.

Robins, N.S., and Younger, P.L., 1996, Coal abandonment - mine water in surface and near-surface environment: Some historical evidence from the United Kingdom. Proceedings of the Conference "Minerals, Metals and the Environment II", Prague, Czechoslovakia, 3 - 6 September 1996. Institution of Mining and Metallurgy, London. pp 253 - 262.

Rogoz, M., 1994, Computer simulation of the process of flooding up a group of mines. Proceedings of the 5th International Mine Water Congress, Nottingham, UK. Volume 1, pp 369 – 377.

Różkowski, A., and Różkowski, J., 1994. Impact of mine waters on river water quality in the Upper Silesian Coal Basin. In: Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 2: 811 - 821.

Ross, S.M.M., 1998, Remediation of Minewater Drainage Pollution at Eston and New Marske, Cleveland. Unpublished MSc Thesis, Department of Civil Engineering, University of Newcastle. 86pp. plus appendices.

Sadler, P.& Rees, A., 1998, Use of probabilistic methods for sizing a wetland to treat mine water discharging from the former Polkemmet Colliery, West Lothian, Scotland. *In* Proceedings of the International Mine Water Association Symposium on "Mine Water and Environmental Impacts", Johannesburg, South Africa, 7th - 13th September 1998. (Volume II). pp. 401 – 411.

Salgado, R., E. Todini, P.E. O'Connell 1989. A Comparison of the Gradient Method with some Traditional Methods for the Analysis of Water Supply Distribution Networks. In: Computer Applications in Water Supply. Vol 1. Systems Analysis and Simulation. pp 38-57.

Saul, H. 1936. Outcrop water in the South Yorkshire Coalfield. Transactions of the Institution of Mining Engineers, 93: 64 - 94

Saul, H. 1948. Mine water. Transactions of the Institution of Mining Engineers, 107: 294 - 310.

Saul, H., 1949. Mines drainage. Transactions of the Institution of Mining Engineers, 108: 359 - 370.

Saul, H., 1959. Water problems in the coalfields of Great Britain. Colliery Guardian, 199: 191 - 199; 229 - 234.

Saul, H., 1970. Current mine drainage problems. Transactions of the Institution of Mining and Metallurgy (Section A), 79: A63 - A80.

Sherwood, J.M., 1997, Modelling minewater flow and quality changes after coalfield closure. Unpublished PhD Thesis, Department of Civil Engineering, University of Newcastle. 241pp.

Sherwood, J.M., and Younger, P.L., 1994, Modelling Groundwater Rebound after Coalfield Closure: An Example from County Durham, UK. Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 2, pp 769 - 777.

Sherwood, J.M., and Younger, P.L., 1997, Modelling groundwater rebound after coalfield closure. *In* Chilton, P.J., *et al*, (Editors), Groundwater in the urban environment. Volume 1: Problems, processes and management. (Proceedings of the XXVII Congress of the International Association of Hydrogeologists, Nottingham, UK, 21 - 27 September 1997). A.A. Balkema Publishers, Rotterdam. pp 165 - 170.

Singh, R.N., 1986. Mine water inundations. International Journal of Mine Water, 5 (2): 1 - 28.

Singh, R.N., and Atkins, A.S., 1983. Design considerations for mine workings under accumulations of water. International Journal of Mine Water, 4: 35 - 56.

Smith, J.A., and Colls, J.J., 1996, Groundwater rebound in the Leicestershire Coalfield. Journal of the Chartered Institution of Water and Environmental Management, **10**, pp 280 - 289.

Smith, D.I., Atkinson, T.C., and Drew, D.P., 1976, The hydrology of limestone terrains. *In* Ford, T.D., and Cullingford, C.H.D., (Editors), The Science of Speleology. Academic Press, London. pp 179 – 212.

Sopwith, T., 1833. An Account of the Mining District of Alston Moor, Weardale and Teesdale, in Cumberland and Durham. W Davison, Alnwick, Northumberland. (Republished 1984 by Davis Books, Newcastle upon Tyne). 183pp.

Taylor, T.J., 1857. Suggestions towards a less local system of draining coal mines. Transactions of the North of England Institute of Mining Engineers, 5: 135 - 141.

Temple, J., 1972, Mining. An international history. Ernest Benn Ltd, London. 144pp.

Teutsch, G.A., and Sauter, M., 1990, Groundwater modelling in karst terranes: scale effects, data acquisition and field validation. Proceedings of the Third Conference on Hydrogeology, Ecology, Monitoring and Management of Ground Water in Karst Terranes. Nashville, TN.

Thewsey, M., 1998, South Lancashire Coalfield closure: Minewater rebound investigation. Tender Documentation, Tender Number T573. Environment Agency, Warrington.

Todini,E. & S.Pilati, 1989, A Gradient Algorithm for the Analysis of Pipe Networks. In: Computer Applications in Water Supply. Vol 1. Systems Analysis and Simulation. pp 1-20

Toran, L. and K.R. Bradbury, Groundwater flow model of drawdown and recovery near an underground mine. Groundwater, 26, pp 724-733. 1988.

Touma, J.P., 1996, Evaluation of groundwater recharge rates for the Northumberland and Durham Coalfields. Chalmers University of Technology Publication B439. Göteborg, Sweden. 64pp. plus appendices.

van Genuchten, M. Th., 1980. A closed-form equation for predicting the hydraulic conductivity of unsaturated soils. Soil Sci. Am. J., 44, 892-898.

Vivian, J., 1990, Tales of the Cornish Miners. Tor Mark Press, Penryn. 32pp.

Walker, J., 1998, The environmental effects of minewater discharge due to progressive abandonment of the North Derbyshire – North Nottinghamshire Coalfield. Unpublished MSc Thesis, University of Sheffield. 67pp.

Ward, R.C. and Robinson, M., 1990. Principles of Hydrology. McGraw - Hill, London & New York, NY. 365pp.

Wardell Armstrong, 1993. A study of the regional hydrogeology of the Durham Coalfield and the potential impact of ceasing mine dewatering (5 volumes). Report to the National Rivers Authority, Northumbria Region, Newcastle Upon Tyne.

Whittaker, B.N., Singh, R.N., and Neate, C.J., 1979. Effect of longwall mining on ground permeability and subsurface drainage. *In*: Argall, G.O., and Brawner, C.O., (editors), Mine Drainage. (Proceedings of the First International Mine Drainage Symposium, Denver, Colorado, May 1979). M Freeman Publications, San Francisco. pp 161 - 183.

Whittaker, B.N., and Reddish, D.J., 1989, Subsidence - Occurrence, prediction and control. Developments in Geotechnical Engineering, 56.

Wood, S.C., Younger, P.L., and Robins, N.S., 1999, Long-term changes in the quality of polluted minewater discharges from abandoned underground coal workings in Scotland. Quarterly Journal of Engineering Geology, 32, pp 69-79.

Younger, P.L., 1993, Possible Environmental Impact of the Closure of Two Collieries in County Durham. Journal of the Institution of Water and Environmental Management, 7, 521 - 531.

Younger, P.L., 1994, Minewater Pollution: The Revenge of Old King Coal. Geoscientist, 4, (5), pp 6 - 8.

Younger, P.L., 1995a, Hydrogeochemistry of minewaters flowing from abandoned coal workings in the Durham coalfield. Quarterly Journal of Engineering Geology, 28, (4), pp S101 - S113.

Younger, P.L., 1995b, Minewater Pollution in Britain: Past, Present and Future. Mineral Planning, 65, pp 38 - 41.

Younger, P.L., 1995c, Hydrogeology. Chapter 11. In Johnson, G.A.L., (Editor). Robson's Geology of North East England. (The Geology of North East England, Second Edition). Transactions of the Natural History Society of Northumbria, Vol 56, (5). pp 353 - 359.

Younger, P.L., 1995d, Polluted groundwater in abandoned coalfields: Some hydrogeological and geochemical idiosyncrasies. Proceedings of the Fourth Annual Conference on Groundwater Pollution, IBC Technical Services Ltd and Royal Holloway College, University of London. London, 15th - 16th March 1995.

Younger, P.L., 1997a, The Longevity of Minewater Pollution: A Basis for Decision-Making. Science of the Total Environment, 194/195, pp 457 - 466.

Younger, P.L., (Editor), 1997b, Minewater Treatment Using Wetlands. Proceedings of a National Conference held 5th September 1997, at the University of Newcastle, UK. Chartered Institution of Water and Environmental Management, London. 189pp.

Younger, P.L., 1997c, "The past we inherit, the future we build": Minewater research at the University of Newcastle. In Bird, L., (Editor), Proceedings of the UK Environment Agency Conference on "Abandoned Mines: Problems and Solutions". Held at Tapton Hall, University of Sheffield, 20th - 21st March 1997. pp 49 - 69.

Younger, P.L., 1998a, Adit hydrology in the long-term: observations from the Pb-Zn mines of northern England. *In* Proceedings of the International Mine Water Association Symposium on "Mine Water and Environmental Impacts", Johannesburg, South Africa, 7th - 13th September 1998. (Volume II). pp. 347 - 356.

Younger, P.L., 1998b, Environmental Implications of the Closure of Frazer's Grove Fluorspar Mine, County Durham. Report to the Environment Agency, Northumbria Area. Department of Civil Engineering, University of Newcastle. 51pp.

Younger, P.L., 1999, Mine Water Pollution In Scotland: Nature, Extent And Preventative Strategies. Science of the Total Environment. (*in press*).

Younger, P.L., and Adams, R., 1996a, Improved Modelling of Abandoned Coalfields. First Six-Monthly Progress Report to The Environment Agency and Northumbrian Water Ltd (EA R&D Contract No B04(95)2). Department of Civil Engineering, University of Newcastle, June 1996. 59pp.

Younger, P.L., and Adams, R., 1996b, Improved Modelling of Abandoned Coalfields. Second Six-Monthly Progress Report to The Environment Agency and Northumbrian Water Ltd (EA R&D Contract No B04(95)2). Department of Civil Engineering, University of Newcastle, December 1996. 26pp.

Younger, P.L., and Adams, R., 1997a, Improved Modelling of Abandoned Coalfields. Third Six-Monthly Progress Report to The Environment Agency and Northumbrian Water Ltd (EA R&D Contract No B04(95)2). Department of Civil Engineering, University of Newcastle, June 1997. 30pp.

Younger, P.L., and Adams, R., 1997b, Improved Modelling of Abandoned Coalfields. Fourth Six-Monthly Progress Report to The Environment Agency and Northumbrian Water Ltd (EA R&D Contract No B04(95)2). Department of Civil Engineering, University of Newcastle, December 1997. 31pp.

Younger, P.L., and Adams, R., 1998, Improved Modelling of Abandoned Coalfields. Fifth Six-Monthly Progress Report to The Environment Agency and Northumbrian Water Ltd (EA R&D Contract No B04(95)2). Department of Civil Engineering, University of Newcastle, July 1998. 54pp.

Younger, P.L. & Adams R. 1999 - Improved Modelling of Abandoned Coalfields, Project Record. Report to The Environment Agency and Northumbrian Water No. W6/i614/7 Department of Civil Engineering, University of Newcastle.

Younger, P.L., and Bradley, K.M., 1994, Application of Geochemical Mineral Exploration Techniques to the Cataloguing of Problematic Discharges from Abandoned Mines in North-East England. Proceedings of the 5th International Minewater Congress, Nottingham, UK. Volume 2, pp 857 - 871.

Younger, P.L., and Harbourne, K.J., 1995, "To pump or not to pump": Cost-benefit analysis of future environmental management options for the abandoned Durham Coalfield. Journal of the Chartered Institution of Water and Environmental Management, 9, (4), pp 405 - 415.

Younger, P.L., and Sherwood, J.M., 1993, The Cost of Decommissioning a Coalfield: Potential Environmental Problems in County Durham. Mineral Planning, 57, pp 26 - 29.

Younger, P.L., Barbour, M.H., and Sherwood, J.M., 1995, Predicting the Consequences of Ceasing Pumping from the Frances and Michael Collieries, Fife. In Black, A.R., and Johnson, R.C., (Editors), Proceedings of the Fifth National Hydrology Symposium, British Hydrological Society. Edinburgh, 4-7th September 1995. pp 2.25 - 2.33

Younger, P.L., Teutsch, G., Custodio, E., Elliot, T., Sauter, M., Manzano, M., Liedl, R., Clemens, T., Hückinghaus, D., Tore, C.S., Lambán, J., and Cardoso da Silva, G., 1997, Groundwater Resources and Climate change Effects - GRACE: Final Report. EC Framework III Environment and Climate Programme, Project EV5V - CT94 - 0471. 141 pp.

APPENDIX I - MATHEMATICAL FORMULATION OF THE GRAM MODEL.

Water level rise in an individual pond.

For each time-step in a rebound simulation, GRAM calculates the water budget for each pond as follows:

$R - O - Q_p = \Delta S$

Where R = recharge in the time-step (= recharge rate x pond area (A))
O = inter-pond flow through overflow points (positive for flow leaving the pond, negative for flow entering it) - also covers surface discharges
Q_p = amount of water (if any) pumped from the pond during the time-step
ΔS = change in volume of water stored in the pond during the time-step

The change in water level (ΔL) in the pond over the time-step is related to ΔS by:

$\Delta L = \Delta S / (A.S_y)$

where S_y is the specific yield of the pond at the relevant horizon.

Given that the level at the start of the time-step (L_{t-1}) is always known, then the water level at the end of each time-step (L_t) is given by:

$L_t = L_{t-1} + \Delta L$

It is advisable to use short time-steps when using GRAM to avoid possible numerical instability problems and/or computational inaccuracies. Generally, a time-step of one day has been used without any problems.

Two terms in the above equations demand further discussion: recharge and interpond overflow.

Calculation of Recharge in the standard GRAM package.

Precipitation is applied over the surface catchment area of each pond. Effective rainfall is calculated by subtracting the evaporation rate for the time-step (generally days) by the precipitation for the same time-step. Should evaporation exceed precipitation, effective rainfall is set to zero. In lieu of suitable data the daily evaporation rate can be estimated as the annual rate divided by the number of days in the year. As a consequence of this simplification, the summer daily evaporation rate will be an underestimate and during the winter there will be an overestimate. Since both calibration and predictive simulation are likely to be run for a minimum of a decade, this will only have a negligible effect on the overall water balance.

The percentage of the effective rainfall which is run-off is then subtracted from the total volume of effective rainfall to give a value of recharge. The run-off percentage can be established using a water balance or by estimation of river flow is there is an appropriate river in the catchment. Ward and Robinson (1990) estimate that globally 34% of total precipitation runs-off. Marsh and Littlewood (1978) found that for nine basins in England and Wales over the period 1960 - 1976 the mean run-off was 428 mm (the corresponding rainfall was 897 mm).

The recharge can be attenuated over a number of time-steps, using a method similar to the simple unsaturated zone transfer function applied to the Chalk and Permo-Triassic sandstones by Calver (1997). The user can thereby specify a set of values which define how long it takes recharge (or its pulse) to reach the water table.

Recharge is then applied over the whole area of each pond. The removal of water by pumping (and addition of it by inflow from adjacent mines or aquifers, or from the sea) is also applied over the area of the entire pond. This is necessary because of the assumptions of a flat hydraulic gradient and a relatively fast rate of flow within each pond.

Modelling inter-pond overflow using pipe-flow equations

The transfer of water between ponds and discharge to the surface is modelled using the Bernoulli equation for head loss between two reservoirs connected by a pipe, flowing full. Hence in the model, flow along pipes can only occur when the pipe is completely submerged in at least one pond. The Bernoulli equation has been applied to the head loss between two ponds (Equation 3.1). The total head loss is the sum of the friction head loss and the minor losses (Featherstone and Nalluri, 1982). The minor losses consist of entry loss and velocity head. The friction head loss is calculated using the Darcy-Weisbach equation (Chadwick and Morfett, 1993).

$$H = \frac{0.5V^2}{2g} + \frac{V^2}{2g} + \frac{\lambda L V^2}{2gD} \qquad \textbf{Eqn: 3.1}$$

Gross Head	Entry Loss	Velocity Head	Friction Head Loss

Rearranging Equation 3.1 to make velocity the subject gives Equation 3.2.

$$V = \sqrt{\frac{2g\Delta H}{\left(1.5 + \frac{\lambda L}{D}\right)}} \qquad \textbf{Eqn: 3.2}$$

Where:

V is the velocity of flow in the pipe (m/s),
ΔH is the head difference between the ponds (m),
g is acceleration due to gravity (m/s^2),
D is the pipe diameter (m),
L is the pipe length (m) and
λ is a non-dimensional coefficient which is a function of the roughness and diameter of the pipe.

In GRAM the value of λ can be defined using either the Prandtl-Nikuradse or the Colebrook-White equation.

Prandtl and Nikuradse Equation

Prandtl and Nikuradse divided turbulent flow into three zones (Colebrook, 1939). In the rough turbulent zone λ is only a function of the relative roughness k/D. Equation 3.3 represents the relationship for λ in the rough turbulent zone.

$$\frac{1}{\sqrt{\lambda}} = 2\log\frac{3.7D}{k}$$ **Eqn: 3.3**

Where:
k is the surface roughness (m).
This equation is limited by the assumption that flow is in the rough-turbulent zone. However, it has the major advantage that it does not need iteration to solve. As a consequence the code is simple and keeps calculation time to a minimum. This is important because the Bernoulli equation is a steady state equation, so the time step must be kept as small as is practical.

Colebrook-White Equation

The most commonly used equation for turbulent pipeflow is the Colebrook-White. Colebrook and White combined the von Kármán-Prandtl rough and smooth pipe flow equations (Colebrook, 1939). This results in Equation 3.4.

$$\frac{1}{\sqrt{\lambda}} = -2\log\left[\frac{k}{3.7D} + \frac{2.51}{\mathrm{Re}\sqrt{\lambda}}\right]$$ **Eqn: 3.4**

Where:
Re is the Reynolds number. Which is given by Equation 3.5.

$$VD/\nu$$ **Eqn: 3.5**

Where:
ν is the kinematic viscosity (m^2/s).

Kinematic viscosity is the ratio of dynamic viscosity (μ) to mass density (ρ). The variation in kinematic viscosity is small for the range of pressures and temperatures that occur in most groundwaters, it is therefore common to consider it constant. At 15.5 °c, ρ is 1000 kg/m^3 and μ is 1.124 cP, therefore ν is $1.124 \times 10^{-6} m^2/s$ (Freeze and Cherry, 1979).

In the Bernoulli equation (see Equation 3.1) entry loss and velocity head can be termed the minor head losses. Equation 3.6 (the Darcy-Weisbach equation for friction head loss) can be obtained by neglecting the minor losses and assuming that the total head can be given by the friction head loss.

$$H = h_f = \frac{\lambda L V^2}{2gD}$$ **Eqn: 3.6**

Where:
h_f is the friction head loss (m).
By rearranging and substituting for λ in the Colebrook-White (Equation 3.4), the complex Equation 3.7 is produced.

$$V = -2\sqrt{2gD \, h_f/L} \, \log\left[\frac{k}{3.7D} + \frac{2.51\nu}{D\sqrt{2gD \, h_f/L}}\right]$$ **Eqn: 3.7**

This can be used to get a value for velocity which can in turn be used to estimate the minor losses. As in Equation 3.8.

$$h_m = \frac{0.5V^2}{2g} + \frac{V^2}{2g}$$ **Eqn: 3.8**

Where:
h_m is the minor head loss (m).

Minor losses can then be taken from the total head to obtain a better estimate of friction head loss. Solution is achieved by iteration.

A study was done comparing the volume of flow calculated by the Colebrook-White and the Prandtl and Nikuradse equations. Table A1 shows that the difference in volume of the flow is 0 - 1×10^{-3} % of the total volume, where flow is in the rough-turbulent zone. Therefore, where the assumption of a rough-turbulent zone is valid, the simplicity and speed of operation of the Prandtl and Nikuradse equation makes it preferable to the Colebrook-White.

Random number generator for Monte Carlo simulations.

The FORTRAN implementation of the GRAM code incorporates a random number generator written by Professor G Pegram (University of Natal) to generate a number between 0 and 1, the probability of which is evenly distributed. This number is multiplied by 1000, truncated to an integer and used to identify which element from the data distribution is to be read into GRAM. The data distribution is a random data set produced by Minitab forming a normal distribution from a given mean and standard deviation.

Table A1 Comparison of Volumes of Flow Calculated using the Prandtl and Nikuradse and the Colebrook-White Equations.

Water level in upper pond (m)	Water level in lower pond (m)	Roughness Coefficient (mm)	Length of Pipe (m)	Discharge calculated by the Prandtl and Nikuradse equation (m^3/day)	Discharge calculated by the Colebrook-White equation (m^3/day)	Difference (m^3/day)
-204.06	-297.79	20	1000	99.38644	99.30962	0.07682
36.45	35.89	20	1000	7.682133	7.606090	0.076043
-204.06	-297.79	31	850	1150.498	1150.213	0.285
36.45	35.89	31	850	88.92842	88.64416	0.28426
-204.06	-297.79	4	22.86	11.74609	11.73077	0.01532
36.45	35.89	4	22.86	0.9079212	0.8928590	0.0150622
-204.06	-297.79	31	182.88	2475.246	2474.962	0.284
36.45	35.89	31	182.88	191.3256	191.0419	0.2837
-204.06	-297.79	31	30.1752	6014.504	6014.228	0.276
36.45	35.89	31	30.1752	464.8947	464.6179	0.2768
36.45	33.72	11.633	2000	175.1574	174.4024	0.755
-37.07	-297.79	0.0002	1000	2586.904	1624.197	962.707
40.15	36.45	0.0002	1000	1141.178	652.8956	488.2824
-32.94	-297.79	3×10^{-5}	1000	1726.220	908.6748	817.5452
42.03	36.45	3×10^{-5}	1000	1577.230	821.6074	755.6226
40.15	33.72	6×10^{-9}	1000	2507.120	888.5166	1618.6034
-32.94	-37.07	0.03	1000	798.6786	673.9743	124.7043
42.03	40.15	0.03	1000	538.8600	436.9419	101.9181
-32.94	-37.07	0.03	1000	798.6786	673.9743	124.7043
42.03	40.15	0.03	1000	538.8600	436.9419	101.9181

Appendix II - Equations used in VSS-NET component of SHETRAN

The equation of three - dimensional groundwater flow (5.1), which is employed by SHETRAN, is formed by combining the Darcy equation and the continuity equations, assumes laminar flow, and states that.

$$\eta\frac{\partial\psi}{\partial t}=\frac{\partial}{\partial x}\left[K_xk_r\frac{\partial\psi}{\partial x}\right]+\frac{\partial}{\partial y}\left[K_yk_r\frac{\partial\psi}{\partial y}\right]+\frac{\partial}{\partial z}\left[K_zk_r\frac{\partial\psi}{\partial z}\right]+\frac{\partial(k_rK_z)}{\partial z}-q \qquad \textbf{Eqn. 5.1}$$

where x,y and z are Cartesian co-ordinates, q is a specific volumetric flow rate out of the medium representing sources or sinks, ψ pressure potential, K_x, K_y and K_z the saturated hydraulic conductivity tensor and k_r the relative conductivity which defines the unsaturated hydraulic conductivities as a fraction of the saturated value, and is therefore a function of ψ.

η is the *storage coefficient* defined by Eqn. 5.2. as

$$\eta=\frac{\theta S_s}{n}+\frac{d\theta}{d\psi} \qquad \textbf{Eqn 5.2.}$$

Where, θ is the volumetric moisture content, S_S the specific storage, and n the porosity. In the saturated zone, dtheta/dpsi = zero, theta=n so that nu = Ss. In the unsaturated zone the main contributor to the storage coefficient is the second term.

The model solves for a new value of the pressure potential ψ at each timestep using an iterative finite - difference matrix scheme.

In the NET component, a pipe resistance formula is used to calculate the flow in the pipes between nodes. Flows in the pipes are calculated using either the Colebrook - White empirical pipe resistance formula in conjunction with the Darcy - Weisbach flow formula (see Appendix I), or the simpler, exponential Hazen - Williams formula (Eqn. 5.3).

$$q_{ji}=0.27746C_{ji}D_{ji}^{2.63}\left[\frac{h_i-h_j}{L_{ji}}\right]^{0.54} \qquad \textbf{Eqn. 5.3}$$

where q_{ji} is the flow from node i to node j, C_{ji} the Hazen - Williams flow coefficient for pipe ji, D_{ji} the pipe diameter and h_j, h_i the respective heads at nodes j and i.
The Hazen - Williams formula is generally preferred for most pipe analysis problems when pipes are operating in the turbulent zone. The Colebrook-White formula includes viscosity, which is temperature-dependent, so is applicable over a wider range of conditions.

The equation for the exchange flow between the pipe and the aquifer used in VSS-NET is based on the Darcy equation, assuming laminar flow and states that:

$$Q_p^{n+1}=\beta_p k_r^n\left(z_p^n-\psi^n\right) \qquad \textbf{Eqn. 5.4}$$

Where Q_p^{n+1} is the exchange flow at the current timestep between pipe and aquifer, β_p is the conductance, k_r^n is the relative conductivity (< 1.0 if the column is unsaturated), z_p^n, the head at the pipe, and ψ^n the head at the column node, the superscripts n indicating the value at the previous timestep.