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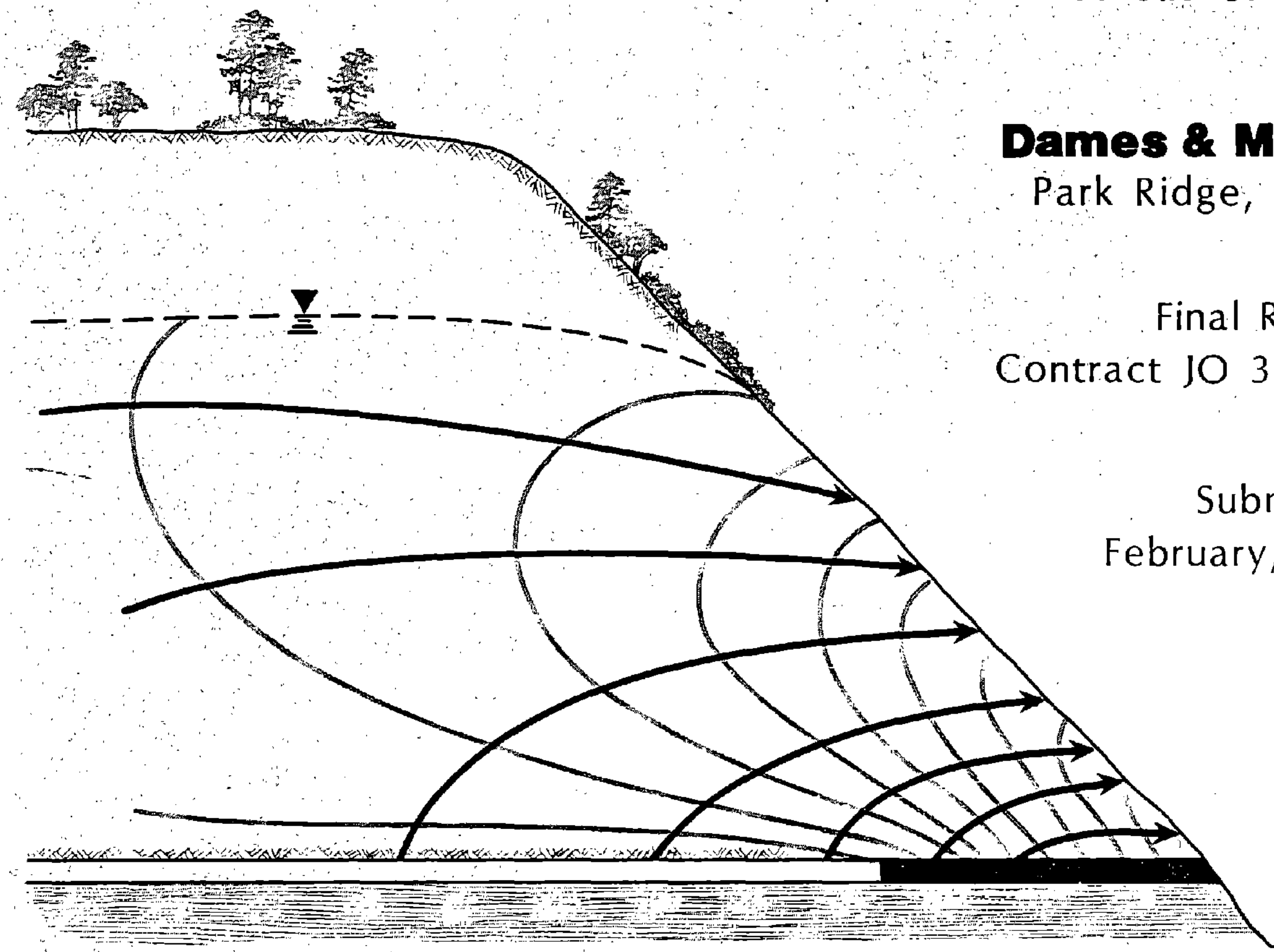
OUTCROP BARRIER DESIGN GUIDELINES FOR APPALACHIAN COAL MINES



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This report includes summaries of existing legislation pertaining to outcrop barriers and a state-of-the-art survey of outcrop barrier designs and related mine closure techniques. Six sites were inspected in the field and are presented as case histories. The data analysis presented includes a computer modeled seepage analysis, an overburden blow-out analysis, and a wedge stability analysis. The results of these analyses are interpreted to form recommendations for coal outcrop barriers that will minimize seepage and the potential for failure. Supplementary support and reinforcement alternatives are also discussed.

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FOREWORD

This report was prepared by Dames & Moore's Park Ridge, Illinois Office under USBM Contract No. J0395069. The contract was initiated under the Mining Environmental Research Program. It was administered under the technical direction of the Pittsburgh Research Center with Mr. Noel Moebs acting as Technical Project Officer. Mr. William R. Mundorf was the Contract Administrator for the Bureau of Mines. This report is a summary of the work recently completed as a part of this contract during the period July 1979 to February, 1981. This report was submitted by the authors on February, 1981.

Dames & Moore would like to acknowledge the assistance and generosity of the Pennsylvania Department of Environmental Resources, especially Messrs. Dave Hogeman, Evan Shuster, Richard Hoffman, and Dick Fisher. The MSHA offices throughout West Virginia were also very cooperative, especially Mr. Romeo and Mr. Gunter of the Pineville, West Virginia, office.

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INTRODUCTION

Mine drainage pollution has been a national concern for many years, especially in the Eastern Bituminous Coal Region where the major contributors to acid mine drainage are abandoned mines. Since presently active mines will be responsible for the treatment of any polluted discharges after abandonment, a method of effectively closing mines is needed.

A previous U.S. Bureau of Mines Contract (No. J0265044) performed by Bituminous Coal Research, Inc. (BCR) reported preliminary indications that mine flooding would be the best method of abating acid mine drainage formation and discharge after abandonment. BCR's study also recommended several research and development programs which would be required prior to implementing mine flooding as an abatement technique. Among the research and development recommendations was a program to determine the specifications for the minimum size of coal barriers, both internal and outcrop, to be left in place to implement mine flooding. The Bureau of Mines has since developed a program that is intended to provide design criteria for use in planning and designing outcrop barriers that will minimize barrier failures and mine drainage problems. This research deals with the design of outcrop barrier widths, potential problems associated with impounding large volumes of water in an abandoned bituminous coal mine, outcrop grouting, and measures necessary to prevent seepage through a coal barrier. The Bureau of Mines authorized Dames & Moore to perform this study under Contract No. J0395069. This report presents the results of the contract and includes guidelines for the design of outcrop barrier pillars that will induce post-mining inundation and maintain stability when subject to hydrostatic pressure.

At the outset of the project an intensive literature search was performed to document the existing technology of mine closure and to determine what legal requirements mine operators are faced with relating to outcrop barriers. During the course of the literature search, several cases were found that would be suitable for the field investigation, and after carefully comparing all of the possible sites, six were selected for field inspection. The case histories of these sites are presented to show a variety of conditions that actually occur within the Eastern Bituminous Coal Region.

A data analysis program was designed to utilize the information obtained in the field and literature studies as a basis for designing outcrop barriers. Both seepage and stability were analysed for a variety of different cases that might typically be found in the Appalachian Region. Based on the interpretation of these analyses, recommendations for the design of outcrop barriers are made. The report that follows describes the results of each phase of the project and presents the recommended design procedures for outcrop barriers.

OUTCROP BARRIER PILLAR LEGISLATION

The following information has been compiled in order to understand the legal requirements placed on mining companies and the enforcement philosophy of the regulatory agencies. Clearly defined limits for outcrop barriers are very seldom specified, and when they are, rules of thumb rather than scientific designs govern the barrier limits. The following sections describe both the Appalachian States' and the Federal requirements.

State Regulations and Inspection Procedures

Barrier pillar legislation has, in the past, been limited to property boundary barriers for approaching an abandoned mine. These regulations were designed to insure the safety of miners by preventing the possibility of punching into a flooded section of an abandoned mine. Pennsylvania, in particular, has become very stringent regarding discharge allowances after abandoning a mine. Although no state has a clearly defined method for computing an adequate outcrop barrier, they have other requirements which provide the states with the authority to approve or disapprove a mining company's plans and thereby control barrier design. The following section describes the regulatory programs in the Appalachian states as they apply to outcrop barrier pillar requirements and the inspection procedures which enforce them.

Pennsylvania

Act No. 729 of the Pennsylvania Bituminous Coal Mining Laws establishes a requirement for "35 feet of rock cover" for approval of proposed mining plans, and a "200 foot safety zone" adjacent to surface water bodies. The Act is included as Appendix A. Section 315 of the Clean Streams Law of Pennsylvania establishes permit requirements for all mine discharges including those that develop after mine closure. Section 315 has been included as Appendix B. Neither act directly establishes a barrier requirement.

The permit evaluation procedure in the State of Pennsylvania involves the submission of a mining plan including those plans related to closure of the mine. The Division of Water Resources evaluates the closure plans and determines whether the barrier pillars are of acceptable width. In cases where there is a difference of opinion, the mining company usually alters their plans in order to obtain their mining permit. The state's inspectors determine barrier pillar requirements by a rule-of-thumb. The most important factor is the amount of hydrostatic head which could be present on the barrier if the mine became totally inundated after abandonment. Once that head is determined, the barrier width requirement is 0.5 meters for every foot of head. In cases where the mine development is down-dip, there is essentially no head on the outcrop, but a minimum barrier of 50 feet is recommended. In cases where there is an anticipated head of over 300 feet, mining up-dip is generally discouraged.

The Office of Deep Mine Safety and the Bureau of Water Quality Management coordinate the inspection procedures after abandonment. An excerpt describing their policy is presented in Appendix C. In most instances, there is no inspection of the outcrop for seepage. The state assumes the final mine maps are reasonably correct and that the outcrop barrier conforms to the amount stated in the original permit.

West Virginia

The State of West Virginia is a member of the Ohio River Valley Water Sanitation Compact and, as such, has agreed to carry out the control measures so established. The West Virginia Administrative Regulations presented in Appendix D describe the objectives within the state. In West Virginia, as in Pennsylvania, abandonment plans must be approved prior to mining. The state evaluates the plans according to guidelines that were derived for property barriers and a general rule of thumb. The rule of thumb requires a 50 foot minimum barrier, and one additional foot for every foot of hydrostatic head that could be present on the pillar after abandonment.

The State of West Virginia generally has the attitude that water that builds up in a mine is a potential hazard, and they would rather drain the mine than sustain the danger of a catastrophic outburst. For this reason, water tight sealing techniques are not commonly found. There is presently a shift in authority from the Department of Natural Resources to the Reclamation Division which will probably lead to more requirements similar to those of Pennsylvania and Maryland.

The inspection procedures in West Virginia involve a spot check of all mine closure sites such as drifts, portals, and shafts, to make sure they meet safety standards and show no visible construction deficiencies. Surveys for outcrop seepage or weak points are usually not performed. Follow-up inspections are not made unless a complaint is registered.

Kentucky

Kentucky Revised Statutes do not address the subject of outcrops with respect to requirements for a barrier. Property boundaries and adjacent mines are discussed in Chapter 352, Section .090 and .490. Twenty-five feet of barrier is required in this instance.

Mine closures are inspected by the State of Kentucky to assure that safety standards are met. All seals with the potential to impound water are required to have a discharge pipe that will prevent any such buildup. These pipes are inspected to ensure nothing is blocking them.

Maryland

The Maryland Register Title 08, Subtitle 13, Section 02, Deep Mining of Coal regulates mine sealing and prescribes barrier requirements. Two sections of particular interest are presented in Appendix E. Outcrop barriers are required to have 50 feet of cover and barriers with hydrostatic head must be at least one foot wide for each foot of hydrostatic head. Formulas and clear definitions are specifically stated in Maryland's regulations.

Inspection procedures are carried out after mine closure. Since all mine plans and closure plans must be certified by a professional engineer, they are assumed to be correct. The stringent regulations of Maryland are enforceable because there are very few active underground mines in the State.

Virginia

The mining laws of Virginia do not address the subject of outcrop barriers in any manner. The only boundaries which are described are those that divide property owners. The subject of inundated mines is dealt with only from the standpoint of punching into them and the hazards associated with approaching them.

Ohio

The State of Ohio has its own Environmental Protection Agency (EPA) which regulates mine drainage occurrences. It has recently been given the authority to approve or deny abandonment plans which are submitted along with the "permit to mine" request. The state has not given much attention to outcrop barrier requirements because it has not had many closures. The Ohio EPA generally recommends the use of double bulkhead seals to form a watertight seal of the portal area, but no regulations specifically define outcrop barrier limits. The Ohio Division of Reclamation has recently started a program to clean up some of the abandoned mine discharges left throughout the state.

Tennessee

The Department of Labor is the regulatory agency for mines and mining in the State of Tennessee. Title 58, Section 1012 defines the requirements for abandoned workings and is presented in Appendix F.

Paragraph (c) of Section 1012 addresses the sealing of abandoned mines. It should be noted that sealing is not required and that in cases where seals are installed, there must be a capped pipe or valve to provide samples of mine atmosphere and a means of determining hydrostatic head. A post-abandonment inspection is made to assure that requirements for posting and sealing are met. No attempt is made to locate seepage.

Alabama

In the Coal Mining Laws of the State of Alabama, regulated by the Department of Industrial Relations, no mention is made of outcrop barriers or abandonment procedures. Guidelines are stated for use when approaching abandoned mines that are likely to be inundated. Outcrop barrier regulations may have little usefulness, however, due to the absence of many outcrop mines.

Federal Regulations and Inspection Procedures

Two Federal agencies are responsible for enforcing regulations regarding abandonment and mine closure procedures. These are the Mine Safety and Health Administration (MSHA) and the Office of Surface Mining Reclamation and Enforcement (OSM).

MSHA requires mines to be sealed as a protective measure to prevent entrance into abandoned mines by unauthorized persons. Specific guidelines for slope and drift mines are given in the Code of Federal Regulations Title 30 Section 75.1711-2 which is presented in Appendix G. The sealing procedure is not designed to withstand much hydrostatic pressure. From conversations held with MSHA inspectors, it became evident that they do not favor water tight seals. The opinion was that there are too many unknown variables and possibilities that could result in uncontrolable accidents or disasters. MSHA does not prescribe limits for outcrop barrier widths.

The OSM also has regulations for the closing of underground openings. In this instance, though, seals must also keep acid or other toxic drainage from entering the ground or surface water system. Section 817.15 of Title 30 Code of Federal Regulations and Section 817.50 are presented in Apendix H. By referring to these regulations it will be found that outcrop barriers are not considered, yet the intent of the regulation is to produce a water tight seal after abandonment.

Concerning enforcement and inspections of mine seals, MSHA becomes involved when there is concern about the adequacy of the seal. In some cases, the District Office will receive complaints from local residents who see an imminent danger in the seal. MSHA will then send out an inspector to check on the situation and make a recommendation to remove the hazard. MSHA does not have the staffing requirements or time necessary to actively seek out leaking mines. The result is that mines are sealed to prevent the entrance of unauthorized people rather than to control drainage.

The OSM has a staff of inspectors, but as of this writing closures of mines that opened after the effective date of the regulation have not occurred. Post-abandonment inspections will become part of their duties when the programs are implemented.

With respect to gravity drainage, decisions are made at the permit stage, and will determine whether or not treatment facilities will be required.

According to the legislation which has been presented, there is a wide variety of attitudes throughout the Appalachian Region, and there is an obvious inconsistency in the recommendation of post-mining inundation as an acid abatement measure. The development of hydraulic seals has led to regulatory recommendations for sealing, but barrier requirements are still based largely on formulas derived for use at adjacent mine boundaries underground. With the advent of OSM regulations, more inundations are likely to occur in order to comply with the zero discharge requirement, and in order to assure the safe accumulation of water in abandoned mines, engineering quidelines for outcrop barrier designs are necessary.

EXISTING TECHNOLOGY REGARDING OUTCROP BARRIER DESIGNS

The literature search that was performed under Task 1 of the project resulted in an accumulation of data relating to the design of outcrop barrier pillars. In addition, a state-of-the-art survey was performed to compile background information relating to sealing technology and mine drainage treatment. The formulas currently used to estimate barrier widths are also described. This information provides the background upon which design criteria will be formulated.

Factors Influencing Barrier Designs

The following section describes the geologic and engineering factors that influence the design of barriers and closure methods and how or why they affect barrier and closure designs. Some of the general characteristics of the Eastern Coal Province are described in each category.

Geologic Factors

The composition of the mine water and the ability to maintain a sound hydrologic system after the completion of mining is controlled by the geology of the coal measure strata. The following paragraphs describe the influences of mineralogy, stratigraphy, structural characteristics, and hydrology of the Eastern Bituminous Coal Region.

Mineralogy

The mineralogical characteristics of coal and associated strata will ultimately affect the chemical composition of the ground water that seeps into the surface water system. The sulfur content is the most important factor regarding water quality. A recent study showed that increasing sulfur content in the coal relates to increases in acidity, sulfate, and total iron concentrations.[4]

The most abundant minerals in the coal and associated strata are quartz, micas, clays, and carbonates. The presence of calcareous rocks (carbonates) has been shown to have a beneficial influence on the pH and net acidity values.[5] Calcareous rocks also tend to increase sulfate concentrations and decrease the total iron concentrations.[5]

Stratigraphy and Structure

In the Eastern Bituminous Coal region, coal seams are usually associated with a series of sandstones, siltstones, and shales, and occasionally limestones. The local structure determines the drainage paths for a given area. The Eastern Coal Region is generally characterized by nearly flatlying beds dipping gently in association with broad folds or regional tilting. Low displacement faults are common in many areas. Jointing is variable, but often associated with axes of gentle anticlines and synclines.

Hydrology

The ground-water flow in coal and associated strata is usually controlled by the rock types, fractures, and the dip of the beds. In the pre-mining, natural conditions, the ground water will flow down through the joints, and fractures and the rock members themselves until a relatively impermeable bed such as the underclay beneath the coal seam is encountered. The water will then flow laterally to the surface. In response to dynamic forces, ground water will generally flow from topographic highs to topographic lows along a subsurface parallel to the local topography. Ground water is also more abundant in synclinal structures than on the anticlinal structures due to the relative structural positions.[21] This is especially apparent where anticlines underlie topographic highs.

In post-mining systems dominated by fractures, the sandstones, limestones, and coal seams still act as secondary aquifers and probably provide most of the ground-water storage. At various locations within the region there are perched water tables which are isolated on impermeable strata and discharge as springs where the ground surface intercepts the permeable strata. It is also possible to find areas of artesian flow in the middle and lower portions of the synclines. Artesian flow is generally associated with intergranular porosity, but inclined fracture zones can also contribute to or be characterized by artesian flow. If the intersecting fracture zone is drained by a mine opening, water is released from storage.

In sandstone, flow is usually along bedding planes, joints, and other separation planes rather than through intergranular spaces. In limestone, groundwater flow is generally along joints which are most predominant along the axes of the folds. At greater depth, however, these joints appear to heal and don't yield much ground water. In shales, ground water flow paths occur primarily along bedding planes and joints. Shale has a relatively low intergranular permeability due to its fine grained and lithified nature.

Engineering Factors

Barrier pillars have traditionally been designed to provide stability and protection from high stresses. More recently, however, they are becoming responsible for impounding water as a pollution abatement measure. Some design experience may be derived from property boundary pillars which have been exposed to hydrostatic forces over the years. There are also some engineering factors that are unique to outcrop barrier design including the physical properties of the coal measure strata, the hydrostatic head, the integrity of the coal, the thickness of the overburden, the method of mine development, and the effects of other mining operations in close proximity.

Physical Properties

The physical properties that influence outcrop barrier designs are permeability and strength. The permeability of coal is an important consideration because of its role in providing a complete seal at the outcrop. In its natural state, the coal seam is very often characterized by horizontal flow due to the presence of a relatively impermeable underclay (many coal seams have been located by finding the seepage along the outcrop). A recent paper by John L. Miller and D. Richard Thompson gives a range of values for the permeability of coal and related rocks typical of the Appalachian Region.[21] Table 1 presents these values. The permeability of coal strata is normally altered as a result of underground mining due to secondary faults and fractures that are formed and enhance the vertical permeability of the overlying strata.

TABLE 1 - Permeability Values Typical of the Appalachian Region

Material Description	Tests	Depth	Average Permeability
Upper Freeport Coal	4	23'-67'	1.00 ft/day
Base of Upper Freeport Coal	3	28'-68'	3.21 ft/day
Lower Kittanning Coal and adjacent shale w/sandstone	4	54'-109'	0.75 ft/day
Shale w/sandstone bridged through a height of 44' over a lower Kittanning mine void	7	50'-99'	4.25 ft/day
Shale w/sandstone over solid coal	12	44'-95'	0.74 ft/day
Mine debris		99'-104'	1.98 ft/day

Source: Miller, J.T., and D.R. Thompson, 1974.

In addition to the permeability, it will be important for all coal pillars to remain stable if post-mining inundation occurs. There is a substantial amount of information on the strength properties of coal, but there has not been much research into the effect of inundation on the strength of coal or underclays. Table 2 gives strength values for coal and associated strata typical of the Appalachian Region. The strength properties will dictate the design of a support system that will allow the safe operation of a mine. It is important to stabilize the mine in order to achieve hydrologic equilibrium and to minimize acid formation. The support system should be able to handle periods of high ground-water infiltration without the threat of bursting through the outcrop or overburden.

Hydrostatic Head

In the design of post-mining closure plans, the maximum head that would be present on a sealed portal or outcrop barrier can be estimated by finding the difference in elevation between the portal or outcrop and the highest elevation of surface water. If seals and outcrop barriers are designed to sustain the computed head, there should be an inherent safety factor in the design due to the probability that new fractures will open as a result of mining, and the post-mining water table elevations will not reach the same height as the pre-mining elevations.

TABLE 2 - <u>Strength Properties of Coal and Associated Rocks</u>

Name of Bed	Location of Rock	Wet or Dry	Compressive Strength Perpendicular to Bedding psi		Compressive Strength Parallel to Bedding psi		Young's Modulus of Elasticity psi		Shear Strength psi		Modulus of Rupture psi	
			Strength	PE**+	Strength	PE**+	Modulus	PE**+	Strength	PE**+	Strength	PE**+
Upper Banner	Roof	Dry	17860	367	16250	388	2,980,000	135,350	4686	109	2984	104
(High Volatile)	Coal	Dry	3011	147			680,000	29,000				
	Floor	Dry	3488	105	2073	242	1,709,000	471,000	1353	74	858	86
Sewell	Roof	Damp	6331	209			2,481,714	173,000	1210	56	790	66
(Medium Volatile)	Coal	Dry	2190	110	1667	50	370,000	20,000				
C Seam	Roof	Dry	15550	528			3,670,000	371,000	6233	261	2846	108
(High Volatile)	Coal	Dry	3122	314			580,000	81,000				
		Dry	7587	1187			4,416,000	984,000*	3948	792	2178	328
	Floor	Wet	6298	273	4740	608	6,466,000	330,000*	3786	168	1733	208
Harlan	Roof	Dry	13892	700	9519	195	8,003,000	819,000*	4833	233	2922	187
(High Volatile)		Wet					8,775,000	424,000*	2615	155	2339	299
	Coal	Dry	5138	224			567,000	38,000				
		Dry	9003	529	7862	666	6,200,000	540,000*	2666	102	1216	350
	Floor	Wet	5392	183	5389	421	4,600,000		2037	177	958	127
America	Coal	Dry										
(High Volatile)		Dry	3383	128	1521	143			1270	80		
	Floor	West	1294	117	765	53	5,500,000		358	29	175	
Elkhorn No. 3		Dry	8392	513	7223	275	6,647,000	306,000*	3397	150	1690	154
(High Volatile)	Floor	West	4726	400	3250	225	5,690,000	665,000*	2242	123	975	66
Pocahontas No. 4	Roof	Dry	14741	254			5,290,000	407,000	6653	305	2167	55
(Low Volatile)		Wet					3,708,000	456,000	5106	264	1730	57
	Coal	Dry	2743	84			370,000	27,000				
Pocahontas No. 3		Dry	14296	1040	12151	810	6,103,000	449,000*	5246	262	2918	282
(Low Volatile)	Floor	Wet	8217	672	5717	616	5,787,000	436,000*	3478	201	1952	101
Pittsburgh	Roof	Damp	1863	143					390	25	275	50
(High Volatile)	Coal	Dry	2710	260	2400	150	470,000	10,000				
Mary Lee	Roof	Dry	20360	856	14607	1307			11743	387	3358	327
		Wet	13787	525	14189	975			10152	483	3845	55
	Coal	Dry	1606	53	1020	59						
		Dry	4451	235	2732	302						
	Floor	Wet	1525	86	1581	191						

*E value obtained by sonic method. These values run from 2 to 3 times higher than values computed from stress-strain measurments.
**PE = Probable Error

Source: Holland, C.T., 1962.

Coal Integrity

Another engineering factor that will influence the barrier width is the competency of the outcrop barrier pillar. There may be weak spots present in the outcrop barrier due to fractures or weathering which may fail and develop into direct openings as the hydrostatic pressure increases. The only way to prevent an incident such as this is to provide for a barrier of sufficient thickness. There is still a discrepancy as to what a sufficient thickness is.

When outcrops are disturbed by mine entries, the integrity of the coal adjacent to a seal is one of the principal problems in obtaining an effective seal. As a result of the mining operation, the coal surrounding the entry is normally fractured. Seepage may take place around the seal, and with increasing hydrostatic head, the seepage rate will increase. Therefore, installation of grout curtains adjacent to the seal is often an integral part of the sealing procedure. This has been proven to be an effective method of reducing seepage in the surrounding coal seam.

Overburden Thickness

The thickness and condition of the overburden are other factors that will dictate procedures for designing outcrop barriers and post-mining pollution control systems. When inundation is being attempted, the overburden must counteract the hydraulic uplift forces. If the overburden is shallow, it will be necessary to leave a larger coal pillar in place to avoid the possibility of seepage topping over the coal and draining out through the overburden. If the overburden is deeply weathered; a large coal barrier will be necessary to sustain excessive forces on it. In cases where air seals are going to be installed, the overburden should not allow access of oxygen to the mine workings. A recent study concluded that mines with thicker overburden are better suited for air sealing.[4].

Mine Development

Mine development planning plays a very important role in mine closure because the mine designs are determined at this stage. Post-mine development practices were usually planned such as to remove ground water from the mine as easily and cheaply as possible and to reduce safety hazards and production problems. The mining operation often used the geologic conditions to an advantage to handle the drainage, and, as a result, many mine openings were located in structural lows. The openings were then developed up-dip in order to provide natural drainage away from the working faces and towards the openings. When mining down-dip, additional pumping costs may be incurred during the operating life of the mine.

The advantage of down-dip mining to post-mine inundation is that the entries are located at topographic highs and are likely to be above the re-established water table. This gives little or no chance of discharge because of the remote chance of fracturing the seal. In cases where there is a down-dip coal barrier the subject of barrier integrity becomes extremely important because the barrier will be subject to the hydrostatic head caused by inundation. Care must be taken to leave enough coal in the barrier pillar to withstand the anticipated hydrostatic pressure. The coal pillar should be relatively sound because no openings have been punched out, but fracturing may have occurred during mining of the area close to the barrier or during pillar extraction. In an EPA study to evaluate up-dip versus down-dip mining, a water quality analysis showed that mining down-dip could be implemented as a pollution control measure.[18] However, in the mine studied, part of the abandoned workings lie above the level of discharge and are therefore not inundated. In cases similar to this an acid discharge may still be possible since total inundation is necessary to prohibit acid generation.

Proximity of Active Mines

The nature of the eastern coal region is such that mines are located in close proximity to one another. Since many mines can be interconnected by a hydrologic network, the following section describes some of the possible hazards.

An EPA study found that strip mining in close proximity to the closure site resulted in lower pH values, increased acidity concentrations, and increased total iron concentrations.[5] Spoil piles contribute the most to these results. When surface run-off comes in contact with spoil material, acid could be formed and subsequently infiltrate down into abandoned workings. In cases such as these, the attempts for pollution abatement become somewhat futile since the water entering an underground mine may be acid in nature and may therefore contaminate relatively clean mine water.

Another surface mining effect is the deterioration of outcrop barriers caused by blasting in close proximity to the abandoned workings. This could cause increased leakage out of the mine or even possibly a "blow-out" through the outcrop which could pose a very serious threat to the safety of persons or property within the path of the released water.

When two mines are located adjacent to one another and one mine is abandoned and partially inundated, a small head will be maintained across the barrier pillar between the two mines. Seepage from the up-dip mine into the down-dip mine may therefore result regardless of the barrier size. Cases have been recorded where several hundred feet of barrier exist and water still seeps through. In such instances a down-dip mine may be contaminated from acid water seeping through from the up-dip mine.

One of the largest problems in the control of mine drainage occurs in areas where multiple seam mining takes place. In order to help prevent factures and subsidence from occurring between two coal seams, it is recommended that barriers in the upper seam be underlain by solid coal barriers in the lower seam. Any passages for water between the two seams will undermine the effectiveness of the barriers in the upper seam by allowing only partial inundation of the mine. In a report by the Maryland Geological Survey, four mines exhibit the loss of water to lower workings by drainage through the intervening rock strata.[17] Another aspect of multiple seam mining is the depreciation of the amount of water in storage to a greater extent than single seam mining, especially when subsidence fractures develop surrounding the mined-out seams. This will prohibit the restoration of the ground water level and enhance acid production.

Current Guidelines For Estimating Barrier Widths

The current methods of estimating barrier widths came about largely as a result of experience with internal and property barriers. From a safety standpoint, it was considered important to leave a barrier to prevent water from a break-out from endangering the lives of men working underground. The methods described in the following evolved from the early 1900's and are still used to a varying degree by mining companies and regulatory agencies throughout the Appalachian Region. Some methods are not discussed because they do not apply when water is involved.

Mine Inspector's Formula or Ashley's Method

The State of Pennsylvania organized a commission of seven men to study the problem of barrier pillars betwen adjacent mines and to determine recommendations for interior barrier widths. The formula they derived was named after George H. Ashley, the State Geologist at that time and one of the men on the commission. After much discussion and deliberation, the following formula was derived:

$$W = 20 + 4t + 0.1D$$

where W = width of pillar; t = bed thickness; and D = thickness of overburden, or, if water is involved, the height of the hydrostatic head possible if it is greater than the vertical thickness of the overburden. The units of all symbols are in feet. The width, W, was to be divided equally on both sides of the property boundary.

This requirement is still contained in the Pennsylvania Regulations, but it is in reference to property boundaries and approaches to abandoned mines. Although Ashley's Formula was not intended as a guide to determine the width of outcrop barrier widths, it has also been used, to some degree, for this purpose. However, it does not take into account many of the factors which will affect the integrity of the outcrop barrier.

Rules of Thumb

The most commonly used method for computing an outcrop barrier pillar's width is the so called rule of thumb expression:

$$W = 50 + H$$

where W = width of pillar, and H = maximum hydrostatic head possible against barrier. It is generally found acceptable in the states of the Appalachian Region, however, the method is based only on experience and does not take any factors into account except for hydrostatic head.

Current Mine Sealing Techniques

The effects of mine inundation have become more predominant due to recent advances in hydraulic sealing techniques that provide impermeable seals. A summary of the techniques presently being used is therefore provided to form a sound background for the development of design principles for outcrop barriers presented in a subsequent section of the report. Some of the seals described are not watertight, but are mentioned because they are still in use in some states.

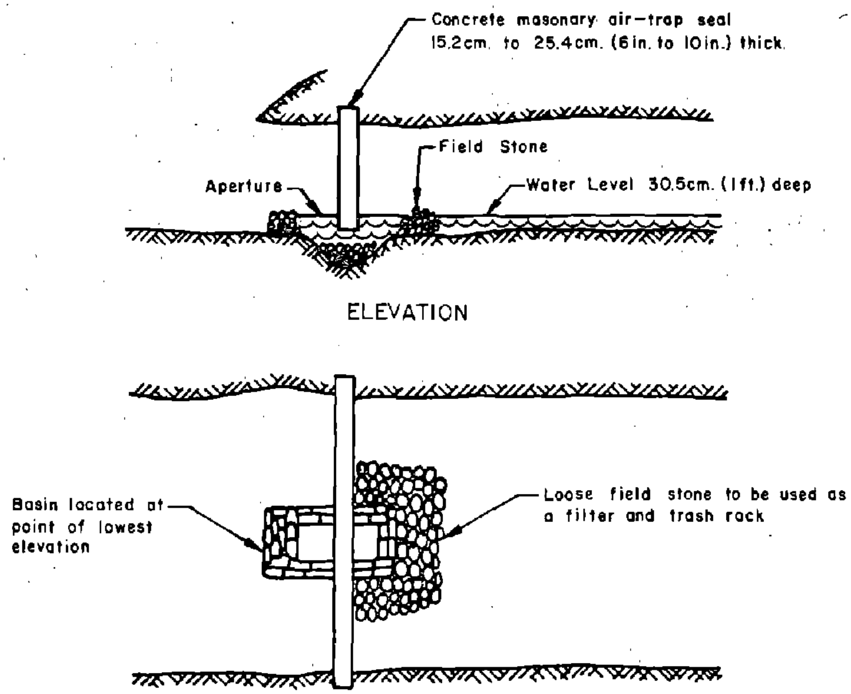
Air Seals

Air sealing is one of the first sealing techniques based on an theoretical approach to the abatement of acid formation. The sealing technique is based on the theory that by prohibiting the entrance of air into the mine at its portals and openings, the oxygen content within the mine will be reduced to a level that will inhibit the oxidation of pyrite.

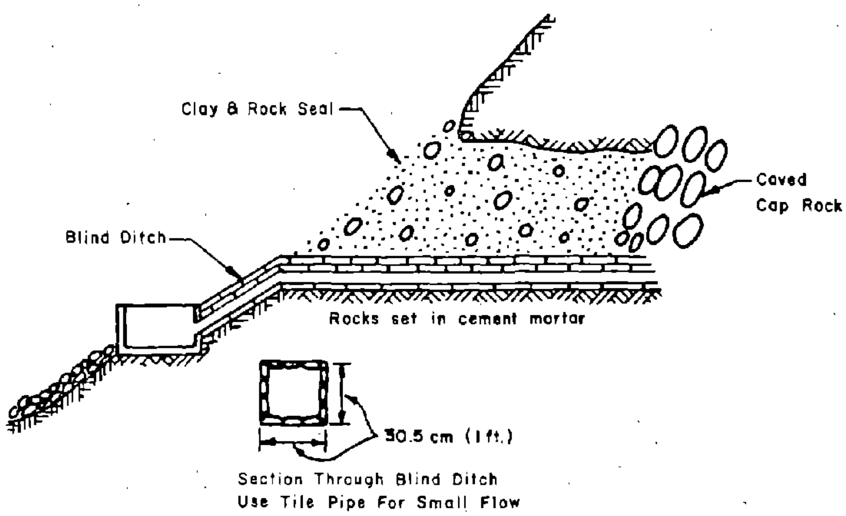
An air seal is the structure placed in a discharging underground mine opening that allows mine water to flow out of the mine without allowing the entrance of air. The air seal consists of one or more dry masonry wall seals with a water trap similar to traps in sinks and drains to prevent buildup of water, but to allow water discharge without letting air in. A typical air seal is shown in Figure 1.

The effectiveness of air seals in reducing pollution has been questioned. The average reduction of acidity by air seals constructed in mines with shallow overburden has been about 50 percent.[4] There are indications that air sealing could be more effective if implemented in small drift mines with thicker overburden, however, the success of air sealing will depend upon the ability to locate and seal all air passages to a mine.[4,23] Underground mines have many air passages such as boreholes, joints, fissures, and subsidence cracks. Even if all passages are located and sealed, porous overburden and fractured outcrops may allow the mine to "breathe" when a pressure gradient develops as a result of changes in barometric pressure.

Since the advent of OSM and the Surface Mining and Reclamation Act of 1977, the outlook on air sealing efforts has diminished rapidly. Since the air sealing technique does not reduce flow there is a constraint on their implementation. If



CONCRETE OR MASONRY AIR-TRAP SEAL



CLAY & ROCK WATER SEAL FOR CAVED PIT MOUTH

(Adapted from Ref. 23)

FIGURE 1

TYPICAL AIR SEALS

air seals are installed, the mining company will be responsible for perpetual water treatment. On the other hand, the air seal is generally more effective over a longer period of time and requires less maintenance, but there have been instances when the water passage has become blocked causing a head to develop in the mine. This will present a hazard since none of the seals are designed to sustain a head.

Dry Seals

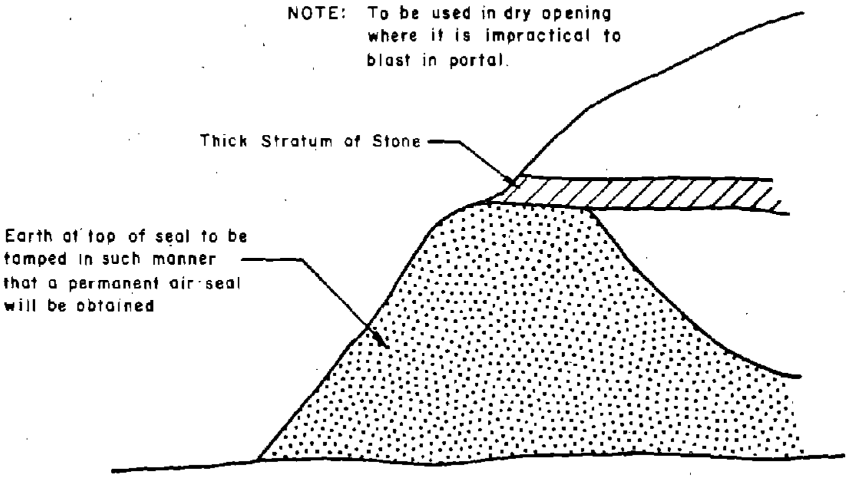
Dry seals were the first known method of mine sealing designed for the purpose of pollution abatement. The construction was sponsored by the Works Progress Administration (WPA) in the 1930's.[23] Dry seals have been used extensively in conjunction with air programs to seal openings where no flow occurred.

Dry sealing has been defined as the complete closure of mine drifts, slopes, shafts, subsidence areas, fractures, and other openings with impermeable material or structures at locations where there will be very little or no hydrostatic pressure. The seals may be constructed of masonry block, clay, soil or other suitable materials. The function of dry seals is to prevent the entrance of water and air into a mine. Figure 2 illustrates a typical dry seal construction.

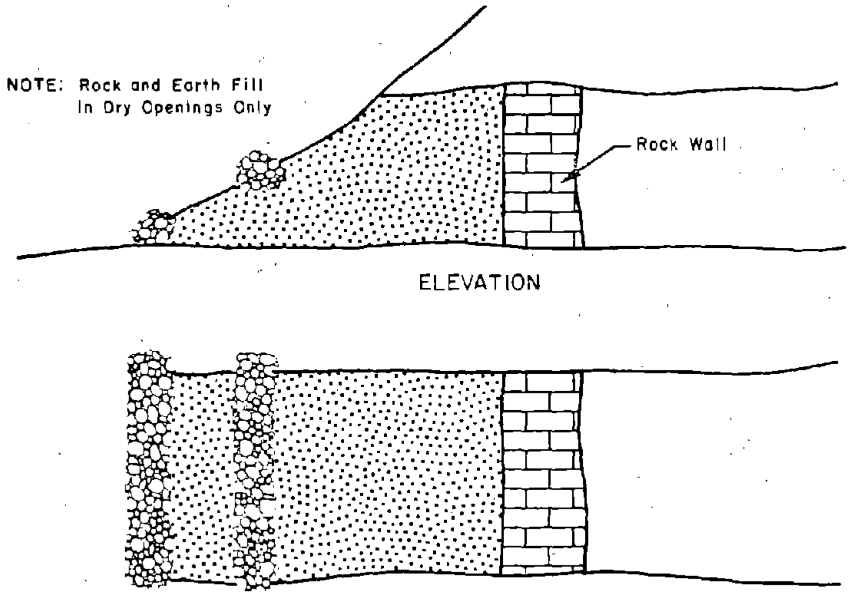
The effectiveness of dry seals has never been determined separate from an air sealing system. Most often the problem of air and water entry into sealed mines is due to cracks and fissures in overburden and along the outcrop, not to leakage at the dry sealed entries. In general, dry seals have only limited usefulness in water pollution control. However, they are useful for keeping the surface water out of a mine.

Hydraulic Seals

Hydraulic seals can be defined as the sealing of any mine openings (that is entries, drifts, slopes, shafts, boreholes, subsidence areas, etc.) where there will be hydrostatic pressure in the area of the seal. Theoretically, hydraulic seals improve water quality by reducing the generation of acid and by containing any pollutants that are formed. Thus, any amount of seepage discharged into surface waters should be small enough to be neutralized or diluted by natural means. The success of a hydraulic sealing program does not depend solely on the competence of the seal because when a mine becomes



EARTH SEAL FOR DRY OPENING



PLAN

ROCK AND EARTH FILL SEAL

(Adopted from Ref. 23)

FIGURE 2
TYPICAL DRY SEALS

undated, the mine perimeter must also be able to withstand the water pressure and able to reduce seepage rates out of the mine. In addition, the success of the program will depend upon the mine's location with respect to local ground water levels because total inundation is required to prohibit the generation of acid.

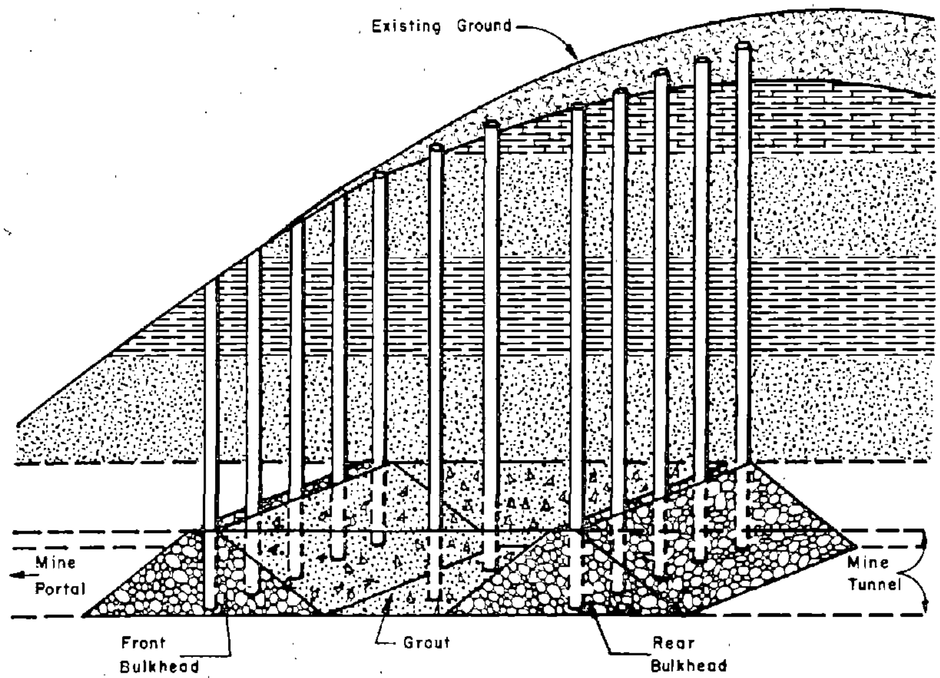
When considering a hydraulic sealing program, an extensive survey must be conducted prior to construction to identify hydraulically unsound areas and highly permeable zones. If remedial measures such as curtain grouting can improve these areas, a seal design can be started. If such remedial measures become technologically impractical or uneconomical, other pollution abatement techniques will have to be considered.

If a hydraulic seal is justified, many alternative designs are available. The most commonly used is the double-bulkhead seal. This seal is constructed by placing two retaining bulkheads in the mine entry and then placing an impermeable seal in the space between the bulkheads. The front and rear bulkheads provide a form for the center seal which is placed by injecting concrete or grout. Quick setting cement and grouted coarse aggregate may be used for the bulkheads. Curtain grouting of the adjacent strata is normally done for additonal strength and reduced permeability. Figure 3 illustrates a double bulkhead hydraulic seal design.

A single bulkhead seal is an alternative to a double bulkhead when less strength is required. The seal is constructed by placing coarse, dry aggregate in the mine entry or other openings. The aggregate is then grouted with a quick setting cement slurry to form a solid aggregate plug as shown in Figure 4. The effectiveness of single bulkhead seals is very dependent on the ability to control seepage around and especially under the seal. Curtain grouting usually supplements the seal.

Another alternate hydraulic seal is the gunite seal. This type of seal is constructed by placing layers of gunite, a pneumatically placed low slump concrete, in a mine opening until full. The roof, sides, and floor of the mine opening are cut so that a tapered seal is formed. This type of seal must be placed in an accessible entry and in areas of sound adjacent strata.

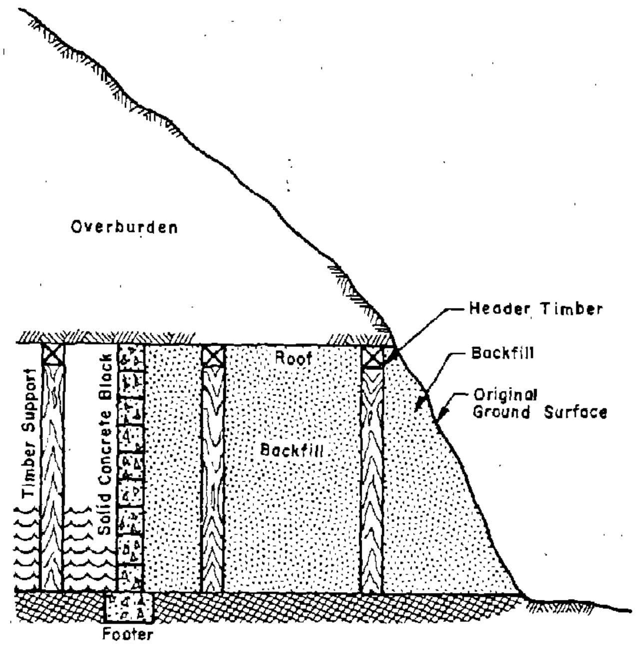
Clay seals have been used to some extent as hydraulic seals for up to 30 feet of hydrostatic head. The mine entry is



ELEVATION

(Adapted from drawing in reference No. 26)

FIGURE 3
TYPICAL DOUBLE BULKHEAD SEAL



ELEVATION

(Adapted from drawing in reference No. 26)

FIGURE 4
TYPICAL SINGLE BULKHEAD SEAL

first cleaned of all debris, then the clay material is placed inside the entry in layers and compacted. In most cases, an earth backfill material is placed over the seal and entry to hold it in place and prevent erosion.

Hydraulic seals may be placed using two different approaches. Firstly, the seal is placed in the mine openings directly, which can be done only where entries are accessible. The advantage to entry placement is that it provides for on-site inspections of construction work and mine conditions. Entry placement has a disadvantage in that the working environment is more hazardous due to the weakened condition of the roof near the outcrop. Secondly, when entries are not accessible, placement is performed remotely through boreholes. The advantage with this method is the ease of installation, however, the disadvantage with this method is that the conditions that actually exist in the entry cannot be determined and the quality of the seal can therefore not be guaranteed to the same extent where access to the mine entry is available.

There are numerous problems associated with hydraulic seals and the impoundment of water in a mine. One that has been mentioned in the seal descriptions is the difficulty encountered in anchoring the seal into the roof, ribs, and floor. The greatest problem seems to be the seal-floor interface. Leaks which develop around the seal may cause sufficient lowering of the mine pool elevation to degrade water quality. Another problem associated with hydraulic sealing is that the local ground water table elevation may be raised. In some instances, homes have become flooded as a result of an elevated water table. The greatest cause for concern, however, is the threat of an outcrop or perimeter failure and the ensuing rush of mine water into the surrounding area. An EPA Study found that with double bulkhead seals, as the water levels and hydrostatic pressures increased, the mine waters were in almost all of the studied cases diverted.[5] The diversions ranged from other mine openings which were not sealed, to weak points in the coal outcrop, to the surrounding strata, to the seal itself. In most cases, the contact between the seal and the mine floor was eroded first. In mines where only partial flooding occurred, leakage was also attributed to an increased seepage rate through the mine floor as a result of the elevated hydrostatic pressure.

Consequently, safety measures such as emergency discharge boreholes or mine pool drawdown systems, are recommended to be incorporated as part of the hydraulic seal design. An emergency discharge borehole, when drilled at the elevation of maximum sustainable head, will allow the free flow of water out of the mine in any case where the head exceeds that level. The mine pool drawdown system consists of a pipe through the lowest mine entry that is equipped with a valve that can be opened to release a dangerous accumulation of water.

Permeable Seals

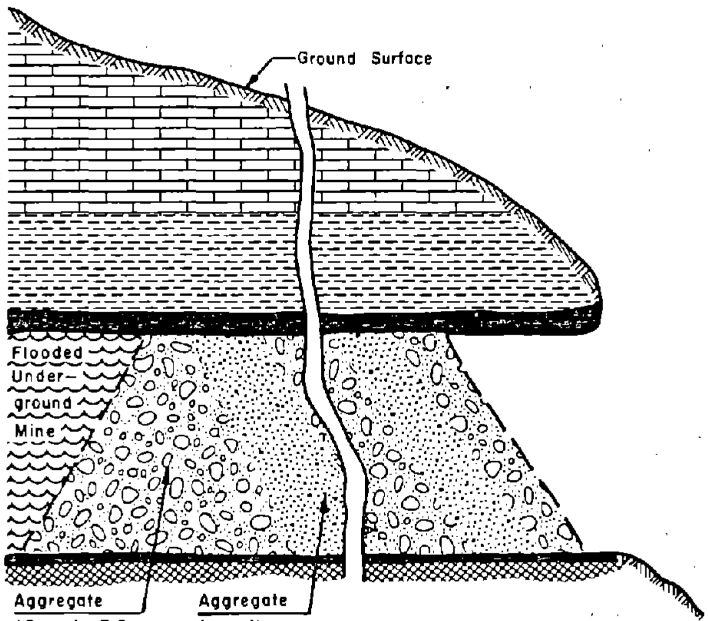
Sealing of underground mines with permeable seals involves the placement of permeable alkaline aggregate in mine openings through which acid water may pass. As the acid water passes through the alkaline materials, neutralization occurs and precipitates are formed.[23] Theoretically, the precipitates continue to form and clog the pores in the aggregate until the permeable seal actually becomes a solid single bulkhead seal of aggregate and precipitate material and flooding of the mine occurs.[26] Figure 5 illustrates a typical construction design.

The principle involved in designing a permeable seal of alkaline aggregate is in-place treatment of acid mine drainage as it passes through the plug. The aggregate must be so graded that acid mine water flowing through the plug has sufficient retention time to be partially or completely neutralized.[10] If graded limestone is used as the aggregate, iron hydroxide and possibly calcium sulfate are precipitated and eventually the void spaces in the aggregate are filled.

Limestone aggregate seals have been demonstrated by the EPA in West Virginia. Various degrees of mine inundation have been attained at the demonstration sites, although the seals still continue to leak through the aggregate indicating that either the precipitates have not completely clogged the pores, or the precipitates are unable to withstand the water pressure. Increases in pH and alkalinity and decreases in acidity showed the neutralizing ability of the seal. Another problem with these seals has been slumping of the aggregate causing an opening at the mine roof interface.[26] This problem could be solved by grouting the opening. In practice, permeable seals have not yet demonstrated their ability to form an impermeable seal at the mine opening and remain to be proven effective.

Curtain Grouting

Although curtain grouting in itself is not a mine sealing technique, it is used so widely as a supplementary technique that it will be discussed further, here.



ELEVATION

(Adapted from drawing
in reference No. 26)

FIGURE 5

TYPICAL PERMEABLE LIMESTONE AGGREGATE SEAL

Grouting is the process of injecting fluid materials into permeable rock and/or soil formations to fill pore spaces and reduce permeability. Curtain grouting is commonly performed in conjunction with other types of mine seals to control leakage around seals and other permeable zones. Grouting is only applicable where void areas are small. The grout mixtures are pressure injected through vertical boreholes. The injected material sets to form a stiff gel or hardened cement-type material that creates an impermeable barrier in the grouted medium.[22] Grout is commonly used around, and extending away, from mine seals. It tends to fill voids between a mine seal and the mine entryway, and to decrease permeability in adjacent rock. This will reduce seepage bypassing a seal area. Grout curtains can also be placed in areas of permeable or weak outcrop barriers during mine sealing. This serves to decrease leakage rates and strengthens the outcrop barrier to decrease failure possibilities.

The effectiveness of grout curtains depends upon the method of injection, the grout material applied, and the type and condition of the geological formation being treated. Grout packers may be utilized to isolate portions of the grout hole and allow grouting of individual zones. Changing the grout mixture and viscosity will further improve the efficiency of grout injection. A limited subsurface investigation should be performed to obtain information on the character of the strata to be grouted and assist in the estimate of grouting requirements. Grout holes must be properly spaced to ensure that the total area between holes receives grout treatment.

Curtain grouting is a convenient and generally effective method of reducing the flow of water through fissures, fractures, and permeable strata. However, grouting operations are expensive and require skilled personnel having knowledge of the available grout materials, the equipment used, and the various grouting techniques.

The sealing methods which have been presented are a brief summary of their characteristics and applications, and are only intended to provide background information for closure designs. Further references and information are provided in the bibliography.

Current Methods of Treating Mine Drainage

The following brief discussion of water treatment technology is presented since it is likely to be part of the mining and post-mining operations. However, with the implementation of effective sealing techniques, treatment may eventually be limited to the active mining stage.

Treatment techniques can be categorized into two basic groups: Those based on chemical reaction processes and those based on physical processes. Mine drainage can also be treated by combinations of various chemical and physical processes to produce water of almost any desired quailty. Most often, mine drainage is treated to remove those chemical compounds considered to be pollutants that will threaten the aquatic life or restrict other uses of the receiving stream. In some locations, mine drainage is being treated for use as public and industrial water supplies where it is the only source of water available.

The following methods are the basic treatment techniques that are most commonly found in use.

Chemical Processes

Neutralization of acid is the principal chemical reaction process used in the treatment of acid drainage. In the neutralization process an alkali is mixed with acid mine waters to neutralize the acid and to precipitate the contaminating metal salts, which can then be separated by sedimentation and/or filtration. The metal salts commonly found in acid mine drainage can be separated because they become less soluble with increasing pH.

Hydrated lime and limestone are the two primary sources of alkaline materials used because they are readily available and relatively inexpensive and require no pretreatment of the acid polluted waters. The cost of hydrated lime is nearly twice that for limestone and becomes an important consideration when plant size becomes large.[1]

Neutralization with either hydrated lime or limestone creates a sludge, requiring disposal in a manner that will prohibit future water pollution. The amount of sludge is dependent on quantities of pollutants for both processes, however, the relative amounts of sludge produced by the two processes differ.[1]

The water quality resulting from neutralization with hydrated lime or limestone will reflect a reduction in acidity, a significant decrease in iron and other metals, and generally some decrease in sulfates. However, the hardness of the water increases in proportion to the amount of lime added. This may or may not be important depending upon the intended use of the treated water.[1]

Many alkaline reagents are available in addition to limestone such as soda ash, caustic soda, and ammonia. Selection will be based on cost, ease of handling, and the required composition of treated water.

Other chemical processes have been developed and tested, but most cannot compete with the economic advantages of neutralization. Many publications have been devoted to their descriptions and are readily available.[14]

Physical Processes

The physical processes of treating acid mine drainage have evolved from salt water treatment technology. Reverse osmosis is one of the techniques that has been successfully applied to acid drainage treatment.

The reverse osmosis processs produces a high quality water that is fairly independent of the input water quality. However, it also produces a pollutant concentrate which ranges from 20 to 40 percent of the volume of treated water depending on the initial acid concentration of the mine drainage. Disposal of this concentrate presents a much greater problem than sludge disposal in neutralization because the pollutants have not been chemically altered and the concentrate requires either further treatment or other disposal, such as by deep well. Since the process does not produce potable water, consideration of the desired water use will have considerable influence on this choice of treatment method.

Other physical processes have been developed on experimental levels. These include flash distillation, ion exchange, and electrodialysis. Developments and demonstrations have not shown economic feasibility comparable with reverse osmosis, but water quality produced is in some instances better.

Many additional publications are available for further details on treatment methods. Reference 14 provides sources for additional information.

FIELD RECONNAISSANCE OF EXISTING OUTCROP BARRIERS

A field program was implemented to observe conditions which actually exist in the Appalachian Region. A comparison could then be made between the theoretical and actual conditions which occur following a mine closure and inundation attempt. Several Appalachian state agencies and MSHA district offices were contacted to obtain suggestions for possible field sites. In Pennsylvania, the DER allowed Dames & Moore to review their mine closure inspection and abandonment records. This review provided Dames & Moore with a list of recent mine closures in Pennsylvania. From the West Virginia Department of Natural Resources, some specific sites that had seepage problems were obtained. The MSHA district offices were helpful in obtaining information about blowout incidents.

After compiling a list of all the possible sites, they were reviewed for their suitability to the study. Many sites were eliminated because they did not have an exposed outcrop while others were eliminated because they were not closed in a manner that would induce flooding of the mine.

After all unsuitable sites were eliminated, the owners of the remaining sites were contacted to request their cooperation and approval of the inspection. Among those that were approved, the final six sites were selected. The following section describes the site selection considerations, field inspection procedures, and also gives a brief case history of the six sites that were inspected.

Site Selection Considerations

In designing the field program, various factors were considered to have an important effect on outcrop barriers. Among these parameters were barrier width, hydrostatic head behind the barrier, overburden depth above the barrier, disturbance of a coal barrier by an entry, adjacent active mining, level of inundation, and the effect of multiple coal seams. The following section discusses each of these parameters, and describes how the selected sites exhibited the qualities associated with them.

TABLE 3 - Description Of Site Parameters

PARAMETERS	SITE 1	SITE 2	SITE 3	SITE 4	SITE 5	SITE 6
Width of barrier[a]	50' - 300'	100'	100' - 300'	50' - 250'	30' - 100'	15' - 20'
Hydrostatic head	15' - 25'	5' - 15'	6' - 10'	10' - 36'	20' - 30'	approx. 50' at time of blowout
Overburden depth	0' - 250'	0' - 250'	0' - 150'	0' - 150'	80' average	0' - 300'
No. of entries	4	3	2	5	2	0
Active Mining in proximity	No	No	outcrop was stripped	No	outcrop was stripped	Yes
Level of inundation	100%	30%	40% w/ seasonal fluctuation	100% w/ seasonal fluctuation	100%	presently 15%
Multiple seams	Yes	Yes	Yes	Yes	Yes	Yes
Sulfur Content	2.7%	2.7%	2.8%	2.8%	1.5%	1.5%
Dip of Coal Seam	3°	3°	2°	1°	2°	1°
Coal Seam Thickness	40"	40"	36"	36"	35"	30"
Slope of Overburden (H:V)	3.3:1	1.5:1	4.5:1	5:1	3.5:1	1.5:1
Years Since Mine Closure	5	8	10	6	8	7
Discharge Flow Rate[b] (ave.)	9630 ft^3/day	7125 ft^3/day	2890 ft^3/day	4632 ft^3/day	9630 ft^3/day	unknown
Approximate Length of Outcrop	3000'	1000'	750'	1200'	1500'	1500'

[a]This width is that of the coal outcrop barrier.

[b]Discharges are measured from ineffective portal seals.

Barrier Width

A range in barrier widths was sought in order to observe differences in seepage rates. As shown in Table 3, the barrier widths for the six selected sites varied from 15 feet to 400 feet. Variation in barrier widths was also observed within each mine, particularly at Site 1 where the barrier width ranged from 50 to 300 feet. The barrier widths were determined from mine maps. However, occasionally the outcrop was not clearly defined, in which case the barrier widths were estimated based on structure contour maps.

Hydrostatic Head

Since both seepage and outcrop stability are dependent on the level of hydrostatic head, mine barriers exposed to different hydrostatic heads were pursued. Table 3 shows the values for the six sites. In cases where seasonal fluctuations have been recorded, the range of values are given. The level of head at Site No. 6 is an estimate based on the elevation difference across the mine. A blow-out that occurred in 1975 drained much of the mine but the precise level of head is not known. All other heads were determined from observation wells which were installed to monitor such levels as those which developed subsequent to closure.

Overburden Depth

Since the amount of overburden will affect barrier stability and design, a range of depths were sought. For the six sites, the amount of overburden present is shown on Table 3. The depths ranged from zero at the outcrop to a maximum of 300 feet at Site 6. The overburden depths were determined by overlaying USGS topography maps over the mine maps. The maximum depths represent the greatest amount of overburden found anywhere in the mine.

Barriers Disturbed by Entries

Sites were selected that exhibited both undisturbed and disturbed outcrop barriers. The number of entries in the coal barriers for each site are given in Table 3. Mining at all sites was updip, except Site 6. However, at this site a blow-out had occurred through the overburden immediately above the coal barrier. At sites 1 through 5 where entries were present, seals had been installed to prohibit flow out of the mines.

Driving entries normally causes the surrounding coal to weaken and, hence, permits water to escape from the mine. Disturbances such as this were studied to determine what reinforcement would be necessary to prevent seepage.

Active Mining Operations Nearby

The presence of active mining in close proximity to an abandoned and sealed mine could expose the outcrop barrier to abnormal stresses. At Site 6, active underground mines were operating adjacent to the site. At Sites 3 and 5, surface mining operations were active nearby. Table 3 shows that sites 1, 2, and 4 were not influenced by active mining operations.

Level of Inundation

Mines with various inundation levels were sought to establish a relationship with water quality and to determine the ease with which inundation could be achieved. Table 3 shows inundation levels as a percentage with 100 percent meaning that the mine void is completely filled with water. Based on recorded observations, the level of inundation normally fluctuates according to seasonal changes.

Multiple Coal Seams

Multiple coals seams are present at all six sites as shown on Table 3. Site 1, however, was the only site that showed a noticeable influence. Another coal seam had been mined 30 feet below it and contained an artesian aquifer which was discharging at the surface of Site No. 1.

Several other parameters are listed in Table 3 that describe the sites but were not used to select sites. Each of these and the six sites will be further described in a section titled Case Histories.

Field Inspection Procedures

The procedures discussed in the following section were used during the field inspections. In addition, permission from surface owners was secured prior to scheduling field visits at all sites. At the five sites in Pennsylvania, a representative of the Department of Environmental Resources (DER) was present, and an MSHA representative was present at the West Virginia site inspection.

Investigate Outcrop

The first priority upon reaching a site was to locate and inspect the outcrop for any signs of seepage or flow. In the case seepage was present, its location and elevation were plotted on a USGS topographic quadrangle map. This information was later used to determine levels of head existing at the outcrop by comparison with a mine map. The mine map was also used in the determination of the width of the outcrop barrier pillar at the point of seepage.

Water Quality Sampling

Water samples were obtained at seepage locations and shipped to a laboratory for analysis. The samples were analyzed for total iron, total manganese, suspended solids, sulfides, and pH. Samples were collected at seepage locations and mine discharges. In cases where water quality data were made available from state agencies, samples were not taken. All of the Pennsylvania sites had observation wells installed, hence in-mine water quality data were obtained. A summary of the water quality data is presented in a later section describing case histories.

Inspection of Seals

At mines where the entries had been sealed, additional time was devoted to inspecting the seal and surrounding areas. In most instances, the seals had been installed through boreholes from a pad above the portal. The inspection of these areas was to find seepage either coming through the seals or around them. Details of the seal type and corresponding construction methods were obtained for comparison with mine maps so that excavated or backfilled areas could be identified.

Case Histories

The following case histories describe the six field sites and the conditions that exist at each of them. The material is presented with the site background information, followed by the present status as determined by the field inspection and comments on the abandonment program.

Site No. 1

Site No. 1 is located in Venango Township, Butler County, Pennsylvania. The site consists of two drift mines, 1A and 1B, on opposite sides of Seaton Creek (refer to Figure 6). The two mines were producing at the turn of the century but were abandoned shortly thereafter. The site was found to be one of the leading contributors of acid mine drainage in the Slippery Rock Creek Watershed when a survey was performed in 1969. At that time the State of Pennsylvania was developing "Operation Scarlift" to abate some of the acid mine drainage problems within the state.[22] Site No. 1 was determined to have a first

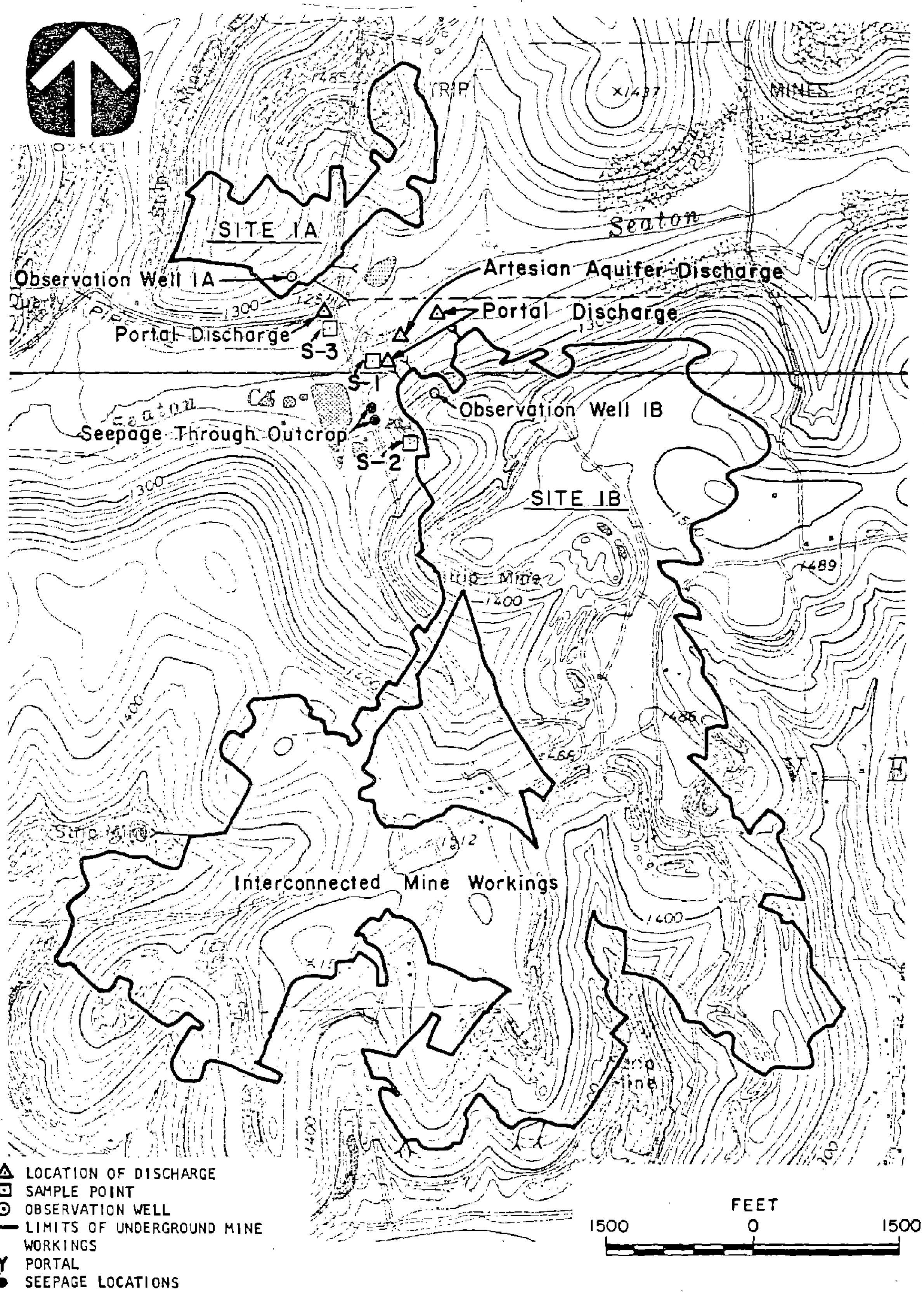


FIGURE 6 LOCATION OF DISCHARGES, SAMPLE POINTS, AND OBSERVATION WELLS FOR SITE NO. 1

priority rating with respect to the urgency of abatement needs. The survey in 1969 found an average of 639 pounds per day of acid being produced by the two drift mines.

Before a sealing program could be designed, a site description was prepared. The two mines at Site No. 1 lie in the Clarion or Brookville Seam where the average coal thickness is 40 inches and the sulfur content averages 2.7 percent. The overburden is 57 to 72 percent shale with 11.5 feet of sandstone and 29.5 feet of coal and calcareous material for a total of 95 to 144 feet. The local dip is approximately 3° SW. Since abandonment, the entries had deterioated and become filled with fallen rock and debris. Mine water was flowing out of portals 1 and 2 in Mine 1A, and portals 4 and 5 in Mine 1B (refer to Figure 6). In general, the mines were either above, or very near drainage.

It was determined by consultants to the DER that double-bulkhead hydraulic seals would be remotely installed, and an observation well would be drilled for periodic samples to be taken. This was performed in March of 1975. After sealing, the acid load in Seaton Creek increased 115 percent. This was attributed to the initial flushing of the mine, leaks that developed in the seals, and a nearby surface mine area that was draining into the Seaton Creek. Both mines were evaluated to determine what remedial measures were necessary.

At mine 1A the flow was reduced by 44 percent. The mine was successfully inundated as of December 1975. A measurement in November 1975 recorded 65 pounds per day of acid in a flow of 50 gallons per minute. It was decided that no remedial work would be performed. No seepage or discharge was observed from anywhere other than the portals.

At mine 1B, flow was reduced by 70 percent and acid load was reduced to 324 pounds per day. The mine has been inundated since March 1975, but the amount of flow still discharging from the mine was reason for concern by state officials. After the seals were installed, discharge was observed for the first time from portal 3. An adjacent property owner reported acidic water in a well.

To remedy the situation, new boreholes extending to 5 feet below the coal seam were drilled in the seals. After resealing, the entire area was inspected for any leakage at weak points in the system.

Discharges continued out of portals 3, 4, 5, and the adjacent property owner's well even after resealing attempts. Another investigation indicated that an artesian aquifer lies 30 to 40 feet below the mine and is the cause of the discharge. At this elevation is also a mined-out portion of the Mercer coal seam which may contribute to the discharge through the sealed portals above.

The present condition of Site No. 1 was determined during a site investigation performed on March 11, 1980. The temperature was near 40° F, and the ground was lightly covered with snow. The first view of the site was a large swampy area accumulating mine drainage as illustrated in Figure 7. Upon walking to Mine 1B, the discharge from portal 3 was encountered. Sample No. 1 was taken from this discharge flow. Officials from the Pennsylvania DER, who were also at the site, made available their records from samples collected at this location. Between portal 3 and portal 5, a pipe was encountered which was discharging water out of the artesian aquifer believed to be 30 feet below the mine level. A large quantity of discharge was emanating from portal 5, but no sample was taken since the DER also had records available for this location. Above portal 5 was observation well 1B. Records of water elevations through this well are kept by the DER and were made available

The outcrop area was further examined to locate any seepage areas or weak points. The second sample was obtained from a discharge point above the outcrop (see Figure 6). Two discharges were noticed along highway 504 that the DER would make records of water quality analyses available. No further discharges were found at Mine 1B.

Inspection of Mine 1A on the opposite side of Seaton Creek, indicated that the discharge from portal 1 is running along highway 504. Figure 8 illustrates this discharge. Sample no. 3 was obtained from this flow. Observation well 1A, which is located behind the seal in portal 1 was also inspected. Water was present in the hole the day of the visit. The last recorded DER measurement indicated 16 feet of water in the observation well. No other discharge or seepage was found at this mine.

The results of the water quality analysis performed on samples 1, 2, and 3 are presented in Table 4 along with all other samples collected. Figure 6 illustrates the location of discharges, sample points, and observation holes.

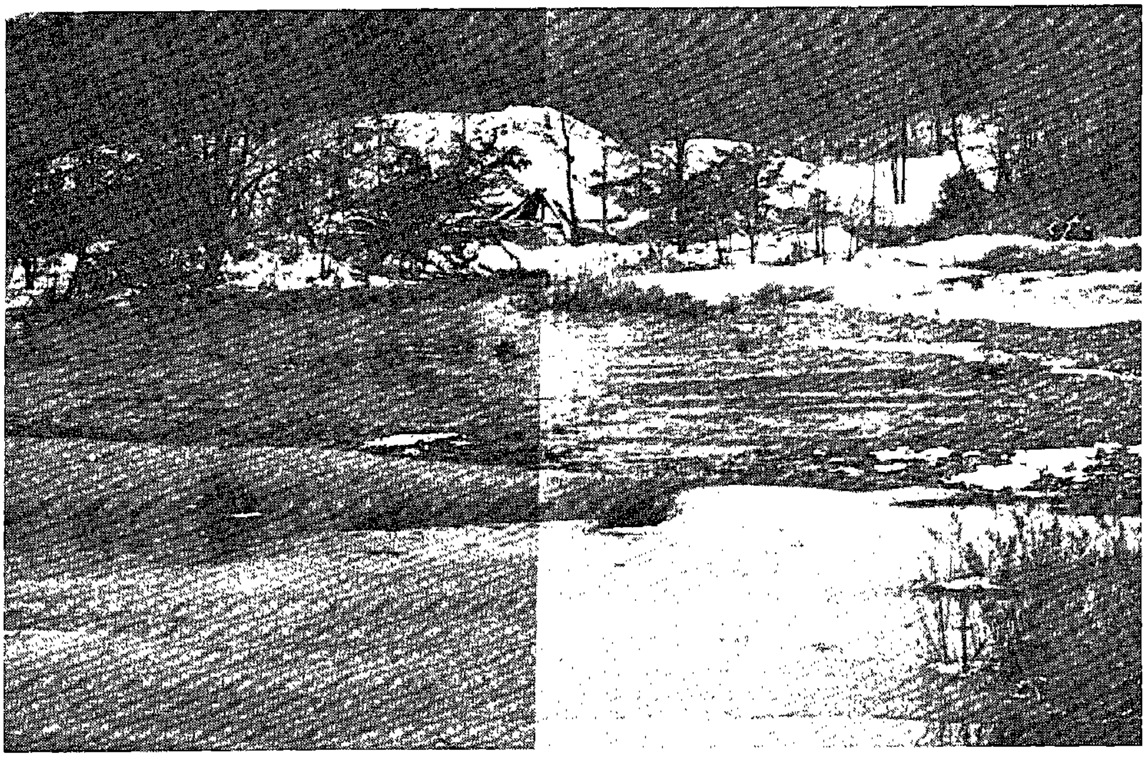


FIGURE 7 SITE NO.1 DRAINAGE
BASIN BETWEEN MINE 1A AND MINE 1B

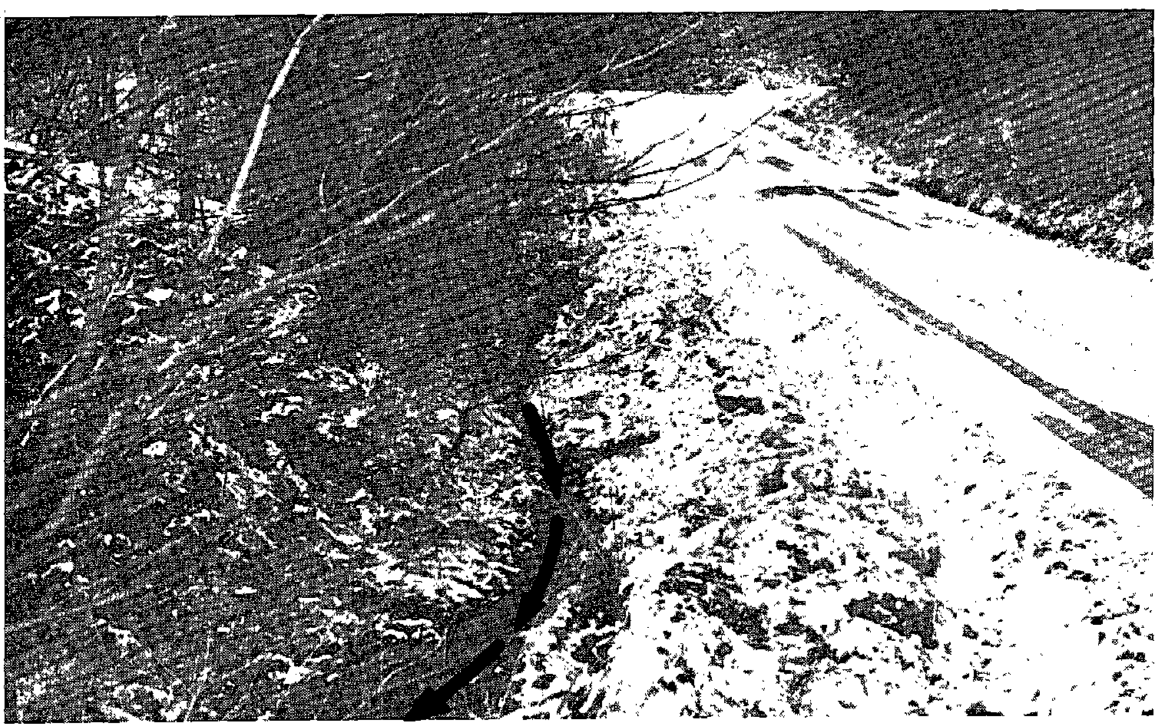


FIGURE 8 DISCHARGE
FROM MINE 1A AT SITE NO.1

TABLE 4 - Summary Of Water Sample Analyses

SAMPLE NUMBER	TOTAL IRON mg/l	MANGANESE mg/l	SUSPENDED SOLID mg/l	SULFIDES mg/l	pH @ 25°C
Site No. 1					
Sample 1	42.0	2.78	22.	0.01	3.49
Sample 2	11.0	0.47	764.	0.06	5.55
Sample 3	14.0	2.47	5.	0.05	3.20
Site No. 2					
Sample 4	0.4	0.98	12.	0.09	3.52
Sample 5	53.0	3.09	4.	0.03	2.97
Sample 6	58.0	3.89	4.	0.01	2.72
Site No. 3					
Sample 7	16.0	4.77	2.	0.06	3.13
Site No. 4					
Sample 8	45.0	6.07	3.	0.03	2.72
Sample 9	62.0	6.68	1.	0.01	2.83
Site No. 5					
Sample 10	12.0	10.90	1.	0.05	2.87
Sample 11	26.0	12.40	1.	0.05	2.81
Sample 12	16.0	9.20	24.	0.06	2.90
Site No. 6					
Sample 13	6.6	0.69	8.	0.02	3.25
Sample 14	18.0	1.03	52.	<0.01	3.24
Sample 15	5.9	0.73	28.	0.02	3.36
Sample 16	3.8	0.61	14.	0.02	3.41

The closure attempts for the two mines at Site No. 1 were successful in inundating the mines, however, seeps and leaks developed which have prohibited heads from exceeding 25 feet. The structural low point of mine 1B is where the seepage was noted along highway 504. This seepage is passing through an approximately 250 feet wide coal barrier pillar. Portal discharges have also been occurring since the closing of the mines. The portal discharges average 63 and 95 gallons per minute for mines 1A and 1B respectively. State inspectors believe the seals were not sufficiently keyed into the floor. Also, these seals were some of the first to be installed and were not constructed to the standards required at present.

A summary of the water quality data for Site No. 1 is presented in Table 5. This information shows that the best quality water is found in the mine pool. When the pool water leaves the mine either as seepage, or a leak, the water quality degrades evidently from contact with pyrites in the outcrop or overburden. Seepage through the outcrop seams to have slightly better quality than discharges through the sealed portals and associated gob. The artesian well discharge does not meet the OSM maximum allowable effluent limitations of 7 mg/l for iron, 4 mg/l for manganese, 70 mg/l for suspended solids, and 6.0 to 9.0 for pH. There may be some influence from the Mercer coal seam that lies 30 feet below the discharge level. Sample No. 2 which was at an elevation above the mine does not meet OSM effluent limitations either.

TABLE 5 - Water Quality Summary For Site No. 1[a]

LOCATION OF DATA COLLECTION[b]	MINE 1A					MINE 1B				
	pH	TOTAL IRON		SULFATES		pH	TOTAL IRON		SULFATES	
		TOP	BOTTOM	TOP	BOTTOM		TOP	BOTTOM	TOP	BOTTOM
Mine Pool via Observation Wells	7.0	8.2 mg/l	30.0 mg/l	135.0 mg/l	175.0 mg/l	6.7	6.5 mg/l	25.7 mg/l	510 mg/l	460 mg/l
Portal Discharges	3.0	32.0 mg/l		330 mg/l		4.0	135 mg/l		870 mg/l	
Seepage through Outcrop						4.4	33.4 mg/l		280 mg/l	
Artesian Well Discharge						5.0	34.2 mg/l		497 mg/l	
Ground-Water Discharge above mine level						5.55	11.0 mg/l		not analyzed	

[a]Data presented represents an average of available records.

[b]Location of collection points is shown in Figure 6.

Site No. 2

Site No. 2 is located in northern Butler County, Pennsylvania. This site includes the largest deep mined area in the Slippery Rock Creek Watershed (see Figure 9). The site consists of 5 large deep mines which had a total production of over 4 million tons of coal from the Clarion seam. Discharges from these mines flow into Slippery Rock Creek. During 1969 these discharges had a combined average acid load of 469 pounds per day.

This site was administered by the State of Pennsylvania as part of Operation Scarlift. The design of the seals was based on the characteristics of the mine and the surrounding

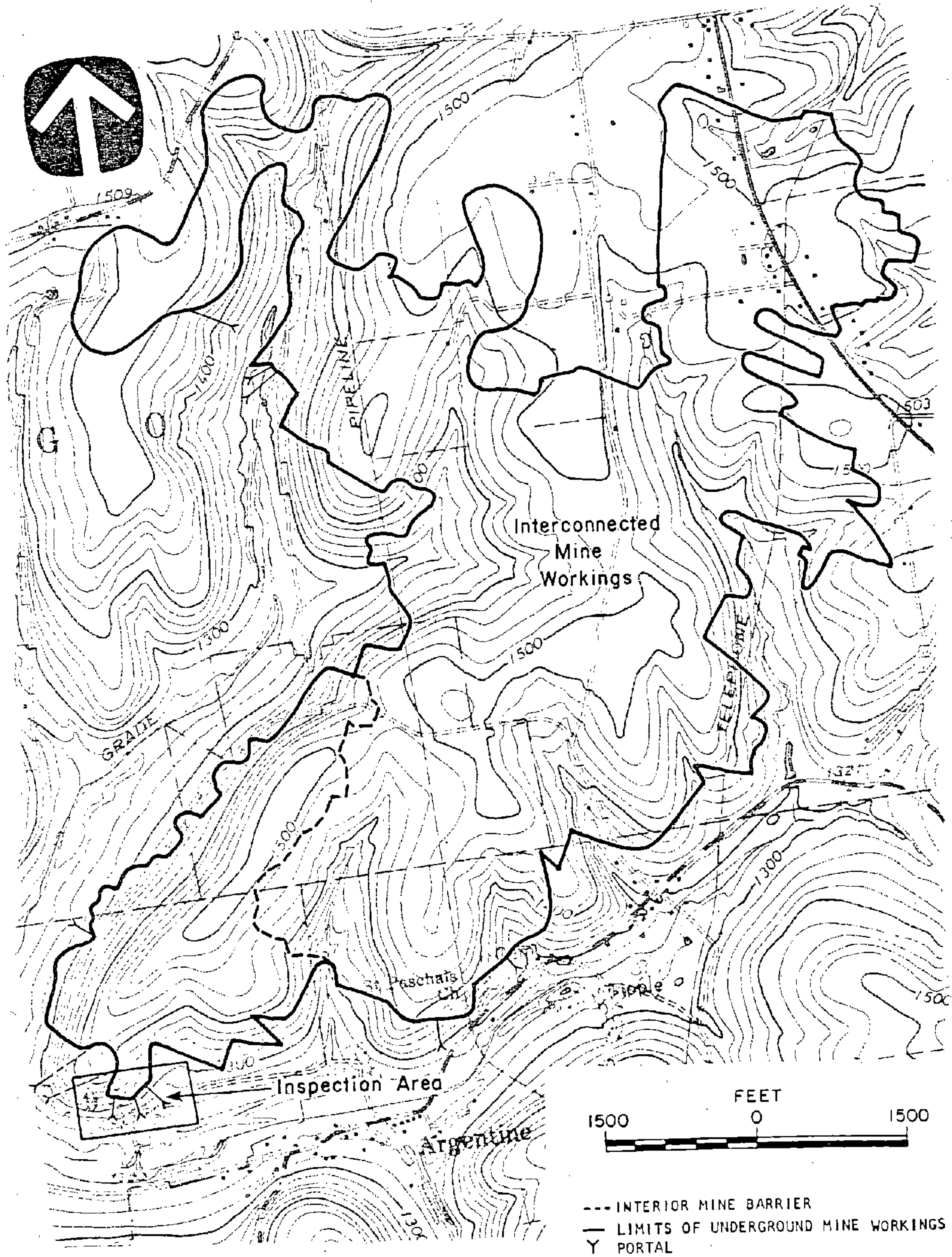


FIGURE 9 SITE NO. 2 INSPECTION AREA

environment. The overburden at this site averages 145 feet. Of this, 105 feet is shale, 13 feet is sandstone, and 26 feet is calcareous rock and coal. The thickness of the Clarion seam is approximately 40 inches and its sulfur content averages 2.7 percent. With regard to the water table, most of the mine is beneath the ground-water table, but some portals are above. The dip of the coal seam in this area is approximately 3° SW.

The DER employed consultants to design the seals and manage the installation operations. Remotely installed hydraulic seals were chosen as the best closure technique. Grout curtains supplemented the seals to prevent leakage at weak spots. For observation and sampling purposes, boreholes were drilled behind five seals. Periodic samples and measurements have been taken since installation.

From January 1972 to January 1973, the water quality information of the observation well samples indicated alkaline water impounded in the mine. The mine discharges indicated a 70 percent reduction in flow and an 85 percent reduction in net acidity.

During this period of time, the mine reached a fairly stable level of inundation having obtained a hydraulic head of about 20 feet. In January of 1973, a wooden mine drain approximately 12 inches square buried in the underclay three and one half feet below the mine seal broke loose from the debris which had apparently previously plugged it. The nature and location of this water course was determined in March, 1973 by excavating near the discharge. As a result of this discharge the mine pool elevation dropped 17 feet in only two months. Despite the drop in elevation only the observation hole near the discharge indicated acidic water in the mine pool. Four other observation holes on the opposite side of the mine remained fairly alkaline. Plans were approved and carried out in 1974 to plug and seal this drainage course. No other remedial work has been performed since then even though discharges remain.

The present condition of Site No. 2 was determined during a site investigation performed on March 11, 1980. The site was lightly covered with snow that was beginning to melt. The inspection only covered the area indicated on Figure 9. Upon arriving at the site, the pad where the seals were installed was inspected. An area directly above a seal was accumulating water. Figure 10 presents a view behind the seal and an area where water is seeping out of the mine pool. This may be caused

FIGURE 10 SEEPAGE THROUGH OVERBURDEN BEHIND HYDRAULIC SEAL AT SITE NO. 2

FIGURE 11 DISCHARGE THROUGH PORTAL AT SITE NO. 2

FIGURE 12 FLOW FROM SITE NO. 2 PRIOR TO ENTERING SLIPPERY ROCK CREEK

by surface runoff and lack of infiltration due to saturated ground. However, based on the appearance of the vegetation, it is felt that the origin is mine water. The elevation of the pad at this point is approximately 1,260 feet, hence only 17 feet of overburden exists at the seepage location.

Above the sealing pad, an observation well is present and has been monitored by the Pennsylvania DER periodically. Water was present in the observation hole at the time of the site investigation. DER records provided the level of inundation.

The abandoned portals could be seen below the sealing pad. Figure 11 illustrates one of the discharging portals where Sample 5 was obtained. The water quality analysis is given in Table 4. What appeared to be two additional mine portals were also discharging. Sample 6 was taken from one of these portals. Mine maps reveal only two portals in this vicinity. State officials assume that discharge is also coming through the outcrop coal which was grouted with only a single line of grout curtain. Sample 4 was obtained near one of the DER's weirs at the location of the third portal. This portal contains the underdrain which caused the discharge in 1973 and the ensuing drop in the mine pool level. Table 4 lists the water quality analysis.

Figure 12 illustrates the area where flow from the four discharge areas accumulates before draining into Slippery Rock Creek. The weir where sample 4 was collected is also shown. The vegetation around these outflows is very poor as a result of the water quality. A portion of the area had been backfilled and graded. The vegetation on these areas is better established. Figure 13 illustrates the location of sampling points, discharges and observation wells for Site No. 2.

The closure attempt at Site No. 2 was also one of the early mine sealing attempts. Inundation was achieved for a short time prior to the blow-out of the underdrain. During that time the head behind the barrier reached 20 feet. After remedial work was performed to seal the drain, the water level has fluctuated between 5 and 15 feet above the mine floor. At present, the mine is only partially inundated.

Three portals at the structural low point of the mine have continued to discharge after they were sealed. This again, is thought to be from flow beneath the seal. The portal

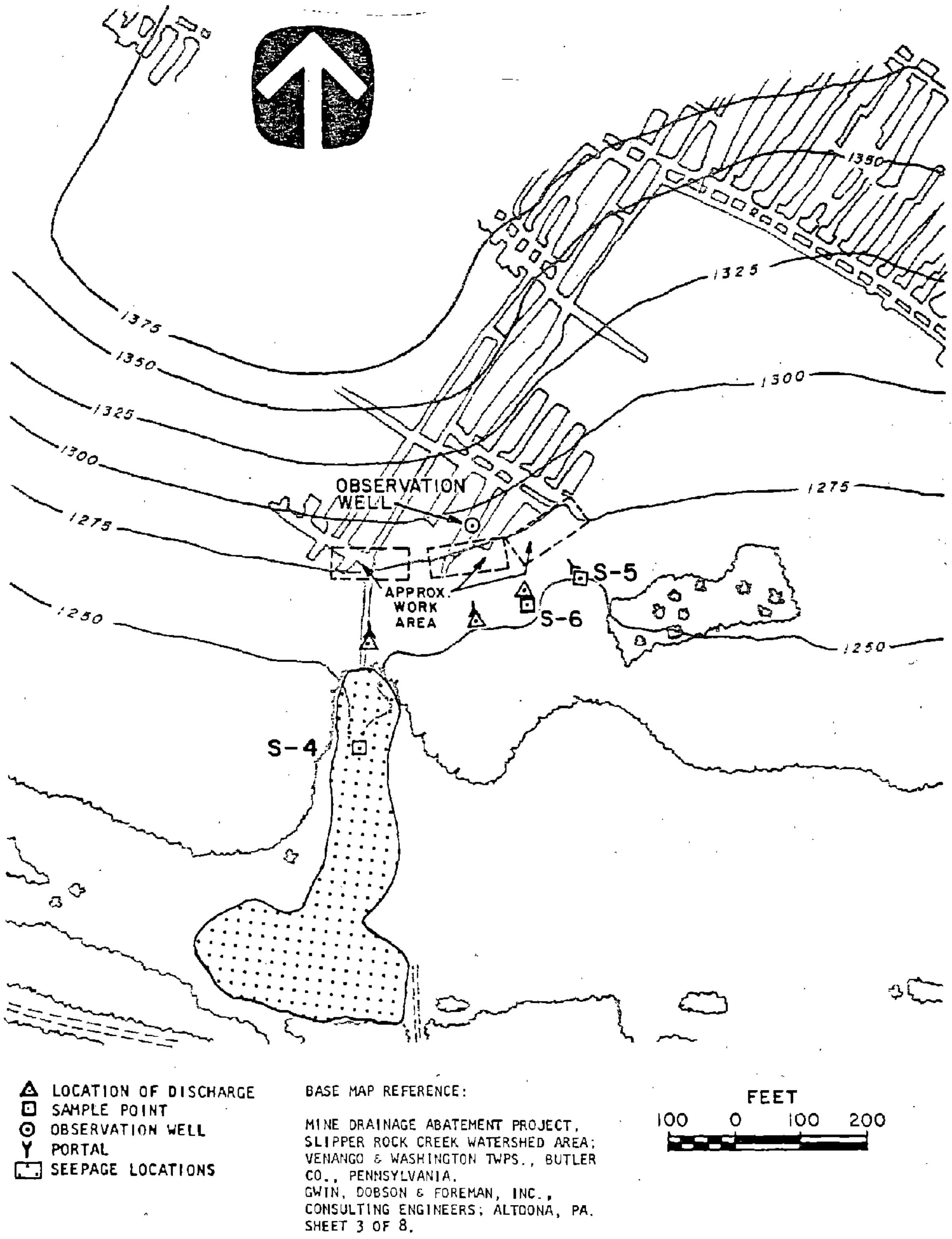


FIGURE 13 LOCATION OF DISCHARGES, SAMPLING POINTS, AND OBSERVATION WELL FOR SITE NO. 2

discharges average 37 gallons per minute. Discharge is also coming through the outcrop despite a single line of grout curtain between two portal seals. The DER now recommends at least two rows of curtain grouting for better effectiveness. Seepage is present behind the sealing pad where there is only 17 feet of overburden.

Determination of the width of outcrop barriers was difficult at this site because the mine map does not agree with the estimate obtained from the sealing construction drawings. The mine map indicates a 200 to 300 foot outcrop barrier pillar, while the seal construction drawings indicate only 75 to 100 feet of barrier.

A summary of the water quality data for Site No. 2 is presented in Table 6. Here, as at Site No. 1, the mine pool water shows the best quality. Portal discharges show slightly better quality than the discharge through the outcrop. The quality of the water flowing through the overburden and outcrop is not known, since the quality of flow was too small to sample.

TABLE 6 - Water Quality Summary For Site No. 2[a]

LOCATION OF DATA COLLECTION[b]	pH	TOTAL IRON	SULFATES
Mine Pool via Observation Well	6.45	0.4 ppm	21. ppm
Portal Discharges	3.3	29. ppm	372 ppm
Seepage through Outcrop and Grout Curtain	2.85	55.5 ppm	not analyzed

[a]Data presented represents an average of available records.

[b]Location of collection points is shown in Figure 13.

Site No. 3

Site No. 3 which is located in Brady Township, Butler County, Pennsylvania, is a small drift mine mined in the early 1900's from the Middle Kittanning Coal Seam. Most of the deep mines in this area were abandoned by 1930. This mine is now part of Morraine State Park and borders Lake Arthur (see Figure 14).

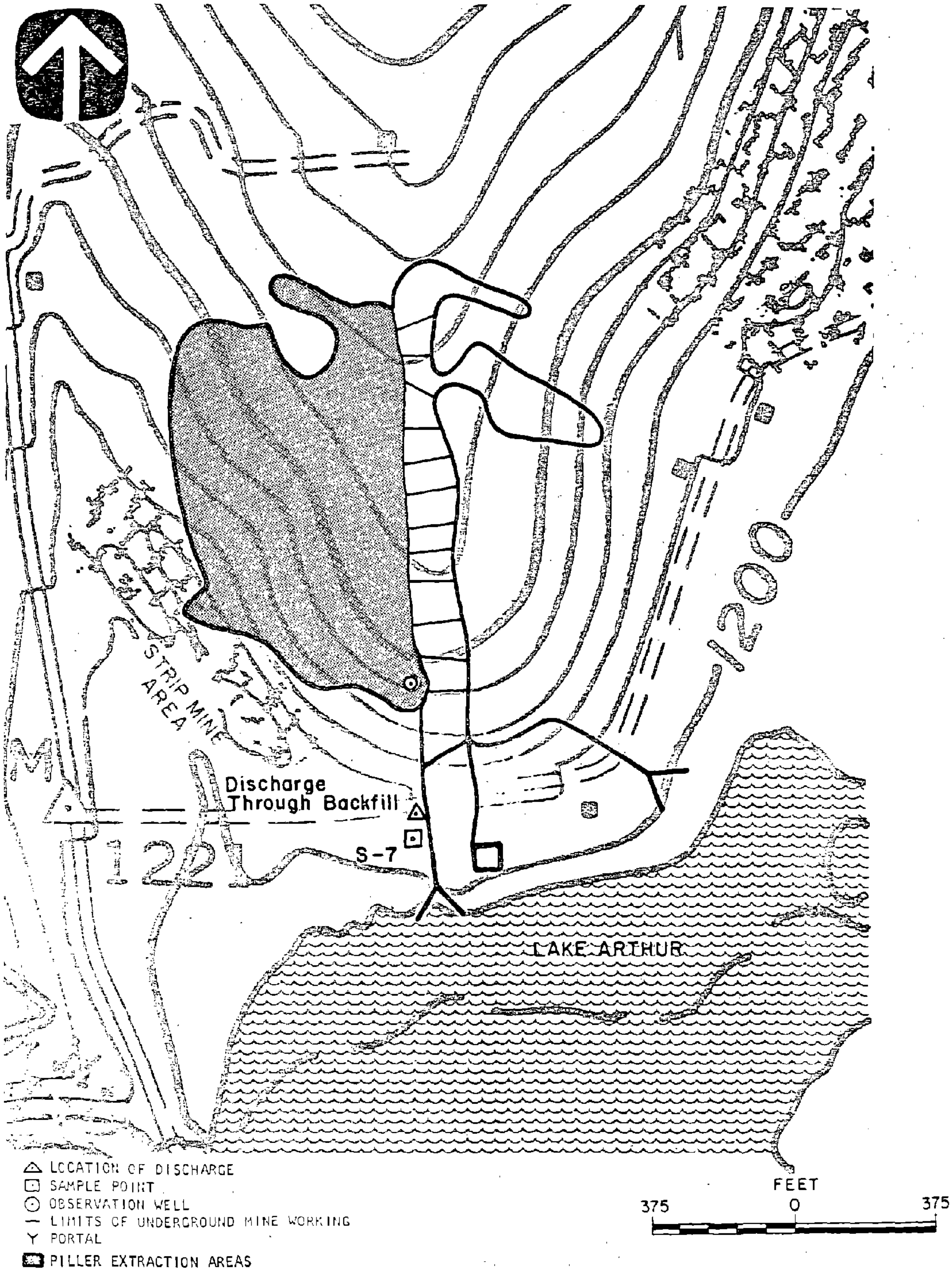


FIGURE 14 LOCATION OF DISCHARGE, OBSERVATION WELL, AND SAMPLING POINT AT SITE NO. 3

Prior to creating the State Park, an extensive survey was performed to determine what pollution abatement procedures were necessary. Mine drainage data were collected over a period of three years. The acid generated in the mine was 102 pounds per day. Site No. 3 ranked second in acid generation of all the deep mines in Morraine State Park.

The mine is located along the northern shore of Lake Arthur. Mine sealing work began in February 1969, and was completed in November of 1970, however, additional construction was necessary to obtain satisfactory results. The initial construction consisted of sealing four entries, an airshaft, and installing several hundred feet of grout curtain.

Since sealing, a fairly steady discharge has produced nearly 30 pounds per day of acid on the average compared to 102 pounds per day prior to sealing. The flow out of the mine was reduced from 25 to 15 gallons per minute as a result of sealing. Iron concentrations were reduced from 8 to 7 pounds per day after sealing.

This mine is only partially inundated and shows seasonal fluctuations. The mine pool water quality has shown variations between acid and alkaline levels. The variations in levels of inundation indicate a 6 to 10 foot range in hydrostatic head on the outcrop.

The water quality of the mine pool determined from an observation well gave results shown in Table 7 for the first two years after sealing.

TABLE 7 - Observation Well Data for Site No. 3 Showing Seasonal Fluctuation

QUARTER	WATER HEIGHT	pH	ALKALINITY mg/l	ACIDITY mg/l	TOTAL IRON mg/l	MANGANESE mg/l
2	3.9'	3.6	0	136	29.3	7.9
3	0'	-	-	-	-	-
4	5.5'	6.1	28	0	30.6	6.5
1	8.0'	4.1	0	124	76	11.5
2	4.8'	6.9	134	0	-	5.1
4	4.2'	7.9	62	0	72	0

Source: Pennsylvania Department of Environmental Resources.

The present condition of Site No. 3 was determined during a field visit on March 11, 1980. The site had been backfilled and regraded because some of the outcrop had been removed by stripping operations. The only source of seepage that was found is shown in Figure 15. Water sample 7 was obtained from this discharge. The analysis is presented in Table 4.

Remedial work was still being performed. A slurry trench system had been surveyed (note stakes on Figure 15) and construction was scheduled for later in the spring of 1980. Figure 14 illustrates the location of the discharge, sample point, and observation well.

The closure of Site No. 3 did not result in the complete inundation of the mine. Discharges resulted which prevented sufficient build-up of head within the mine. The maximum head that has been attained is 10 feet.

After underground mining ceased, the outcrop area around Site No. 3 was strip mined, and has since been backfilled and graded. For this reason, it is impossible to determine how much of an outcrop barrier exists between the backfilled material and the mine void. The only mine map available indicates 100 to 300 feet of outcrop barrier.

Seepage is emanating from the backfill material at the structural low point and drains into Lake Arthur. Additional work is in the design stage to construct a slurry trench to control this seepage. Table 8 summarizes the water quality data for Site No. 3. The mine pool again seems to have the best water quality although not as clearly as in Sites 1 and 2. The seepage water has a lower pH, but not very high concentrations of metals. None of the discharges meet OSM effluent limitation standards.

TABLE 8 - Water Quality Summary for Site No. 3[a]

LOCATION OF DATA COLLECTION[b]	pH	TOTAL IRON mg/l	MANGANESE mg/l	ACIDITY mg/l
Mine Pool via Observation Well	5.7	52	7.75	52
Mine Discharges	-	39	-	166
Seepage through Backfill	3.13	16	4.77	-

[a]Data presented represents an average of available records.

[b]Location of data collection points is shown in Figure 14.

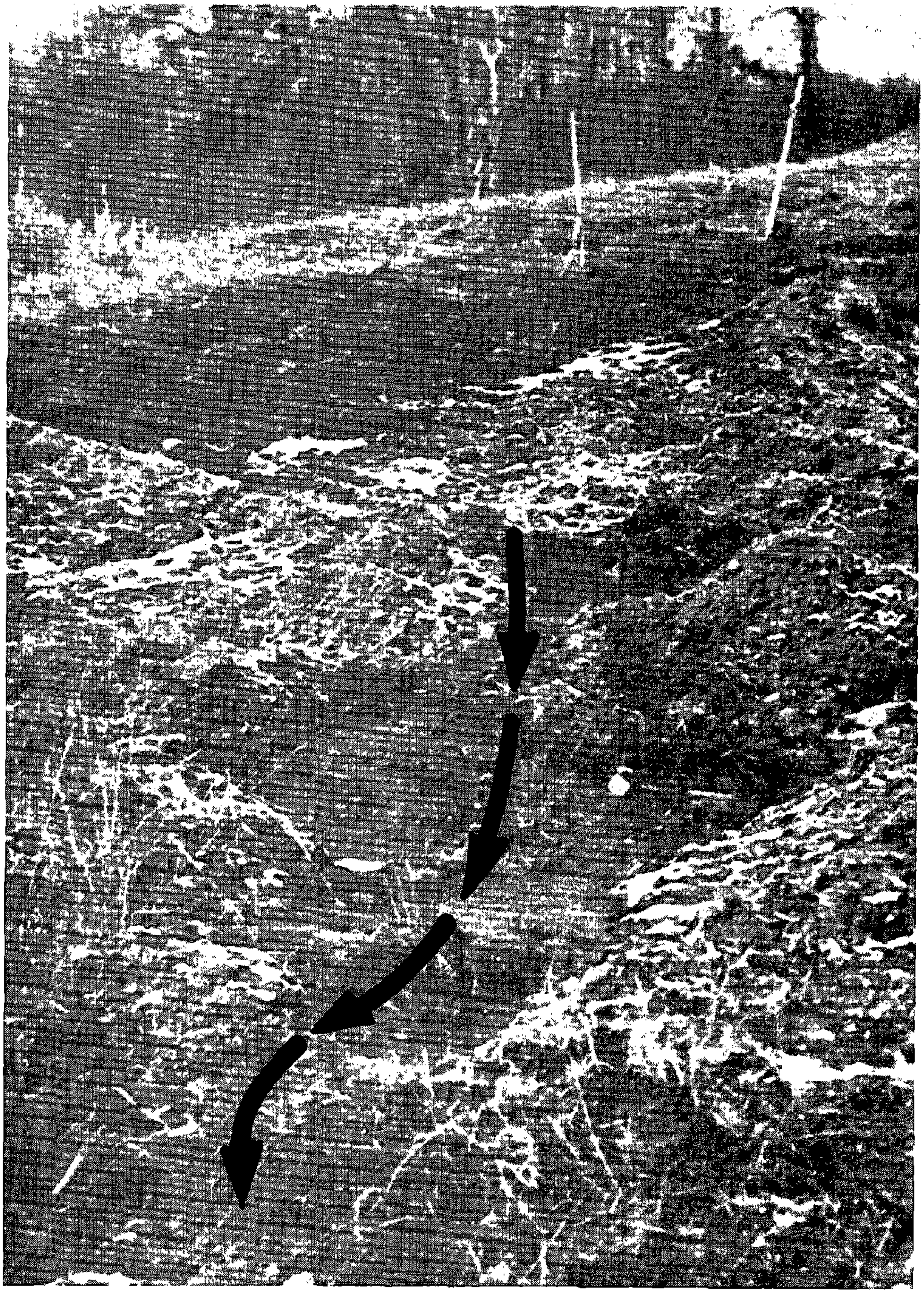


FIGURE 15 SEEPAGE AT SITE NO. 3

Site No. 4

Site No. 4 is located in Brady Township, Butler County, Pennsylvania (see Figure 16). Mine drainage flows into Glade Run and Big Run. Nearby recreation areas are run by the Pennsylvania Historical Society and the Western Pennsylvania Conservancy.

In 1969, discharges from the site produced an average of 640 pounds per day of acid. This site was sealed as part of Operation Scarlift and was funded early as a "Quick Start" Project. The abatement plan consisted of 10 deep mine hydraulic seals, several hundred lineal feet of grout curtain, and 15,000 cubic yards of refuse pile removal.

Site No. 4 lies in the Middle Kittanning Coal Seam. The local dip is approximately 1° toward 249° determined from Works Progress Administration (WPA) maps with structure contours. The overburden thickness varies from 0 to 150 feet. The composition of the overburden is predominantly shale with small amounts of sandstone.

The sealing program consisted of installing double bulkhead, hydraulic seals. Nine entries were sealed in 1974, and a refuse pile was removed. A comparison of water quality data prior to and after sealing shows an increase in flow, sulfates, total iron and acidity. This data may be attributed to the initial flushing of the mine. Discharges have continued since sealing but no remedial measures were deemed necessary.

An observation well is located behind the third entry (see Figure 16). Periodic sampling has been performed to monitor the level of inundation in the mine and the water quality of the mine pool.

The present status of Site No. 4 was determined during a field investigation performed on March 12, 1980. Two drainage paths occur near the pad above portal 2. One was emanating from the abandoned portal 2, and the other appeared to be coming from portal 3 (see Figure 16). Figures 17 and 18 show the discharges coming from portals 2 and 3, respectively. Portal 3 was not as clearly visible as portal 2 where timbers still remained. A water sample was obtained from both portal discharges.

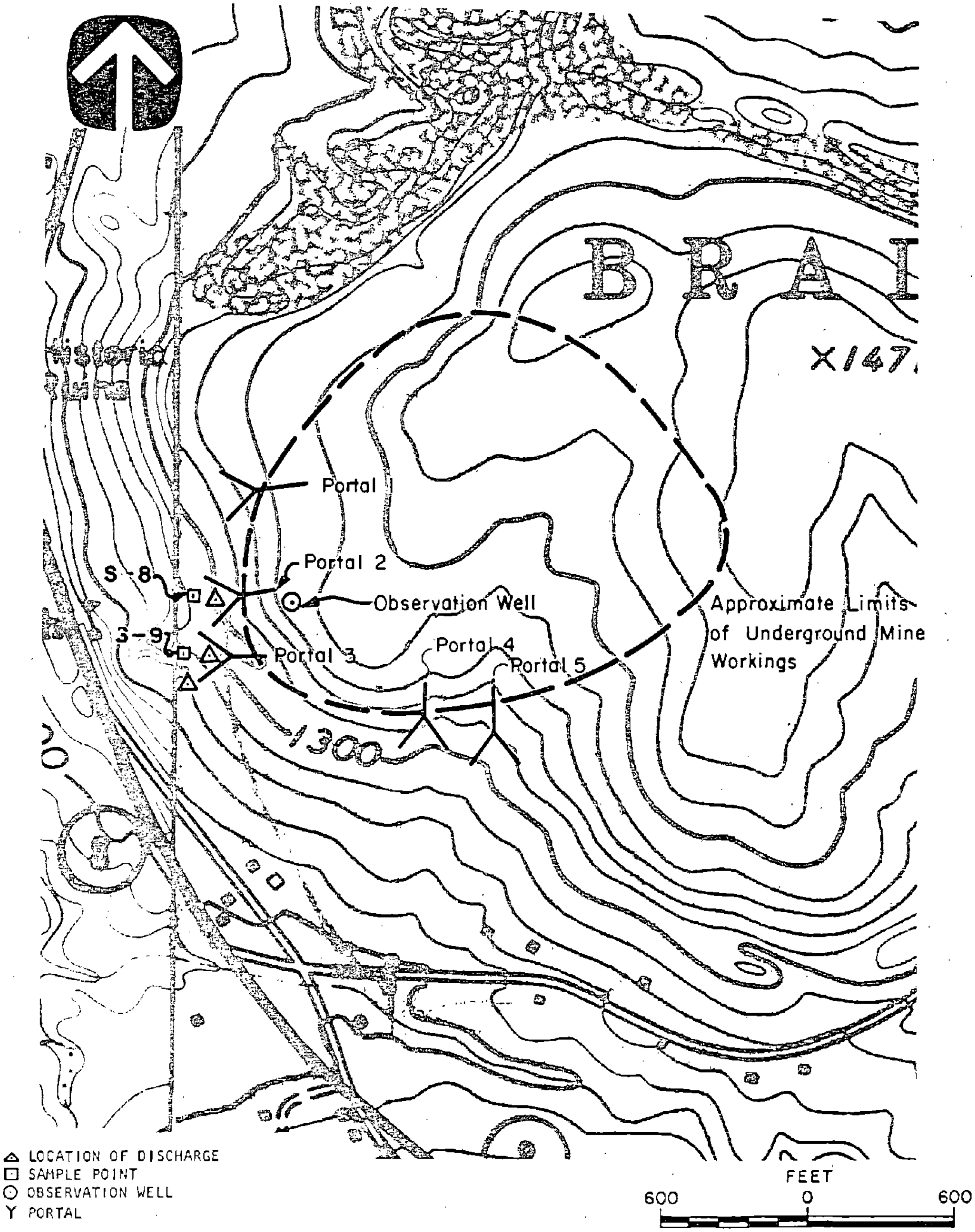


FIGURE 16 LOCATION OF DISCHARGES, SAMPLING POINTS, AND THE OBSERVATION WELL FOR SITE NO. 4

FIGURE 17 DISCHARGE
FROM PORTAL 2 AT SITE NO. 4

FIGURE 18 DISCHARGE
FROM PORTAL 3 AT SITE NO. 4

The discharge analysis results are presented in Table 4 as samples 8 and 9, respectively. The water samples may be diluted to some extent due to melted water from ice still covering the flow paths. Records pertaining to the observation well behind portal 3 were obtained from the DER, but were incomplete regarding the mine pool water quality.

No visible seepage from the outcrop was found at the site. Figure 16 illustrates the location of discharges, sampling points, and the observation well.

The mine sealing attempt at Site No. 4 was almost successful in inundating the mine. Observation well data shows that the pool elevation fluctuates seasonally. During the spring of 1979, the pool elevation reached a maximum of 36.5 feet above the mine floor. Two months later, however, it returned to only 10 feet. The average pool elevation is 13 feet above the mine floor.

Three of the ten mine portals still continue to discharge. These three portals are at the structural low point of the mine as is the coal outcrop. Seepage was not observed along the outcrop area.

The outcrop barrier width is very difficult to determine since a mine map could not be located. Construction drawings from the DER give an approximate outline of the mining limits, but these do not adequately represent all the mine workings. The barriers could be anywhere from 50 feet to 250 feet in width.

A summary of the water quality data for Site No. 4 is presented in Table 9. A comparison cannot be made since the mine pool water quality could not be obtained.

TABLE 9 - Water Quality Summary for Site No. 4[a]

LOCATION OF DATA COLLECTION[b]	pH	TOTAL IRON mg/l	MANGANESE mg/l	SULFIDE mg/l
Mine Pool	- U	N A V A I A	B L E-	
Mine Discharges	2.8	53.9	-	6.4

[a]Data presented represents an average of available records.

[b]Location of data collection points is shown in Figure 16.

Site No. 5

Site No. 5 is located in Jefferson County, Pennsylvania (see Figure 19). Mine water discharged into Rattlesnake Creek Watershed prior to abatement measures. The site is a 0.3 square mile drift mine that produced coal during the early 1900's and was abandoned by 1936. Subsequent strip mining operations removed the outcrop coal and closely approached the deep mine workings. The State of Pennsylvania undertook a drainage abatement project at this site under the Operation Scarlift Program.

Data collection programs were implemented to determine the characteristics of the site. The entire area was inspected to locate seepage and water flow from the site. Preliminary water quality sampling by the DER in 1972 indicated 430 gallons per minute with an average pH of 3.4 discharging from the site. A drilling program by the DER revealed that the underground workings were closely approached by the stripping operations. The overburden, which averages 80 feet in thickness, is predominantly shale with some calcareous material and another coal seam also present.

The seal construction program involved two operations. First, a continuous clay seal was placed to stop flow discharging from the deep mine, and secondly a slurry trench cutoff wall was dug to collect the discharge draining from the highwall. Construction was completed by November 1974. Three observation wells were installed to monitor water levels, acidity, and flow. An evaluation of the abatement program was made in December, 1974 after the first water quality data after sealing had been obtained. The locations of sampling points and the associated water quality data are shown in Figure 20 and Table 10, respectively. A water monitoring program was continued after construction operations ceased. The observation well records have also been presented in Table 11. It is important to note the blowout which occurred on February 21, 1976. The blowout occurred at an elevation 20 feet above the mine with only 10 feet of head in the overburden above it. The blowout resulted in a 3.7 foot loss in head in the deep mine. The head on the bottom of the coal seam was 29.5 feet prior to bursting. By backfilling, the seepage was able to be controlled, but only after a period of one year of discharging water with a pH of 4.

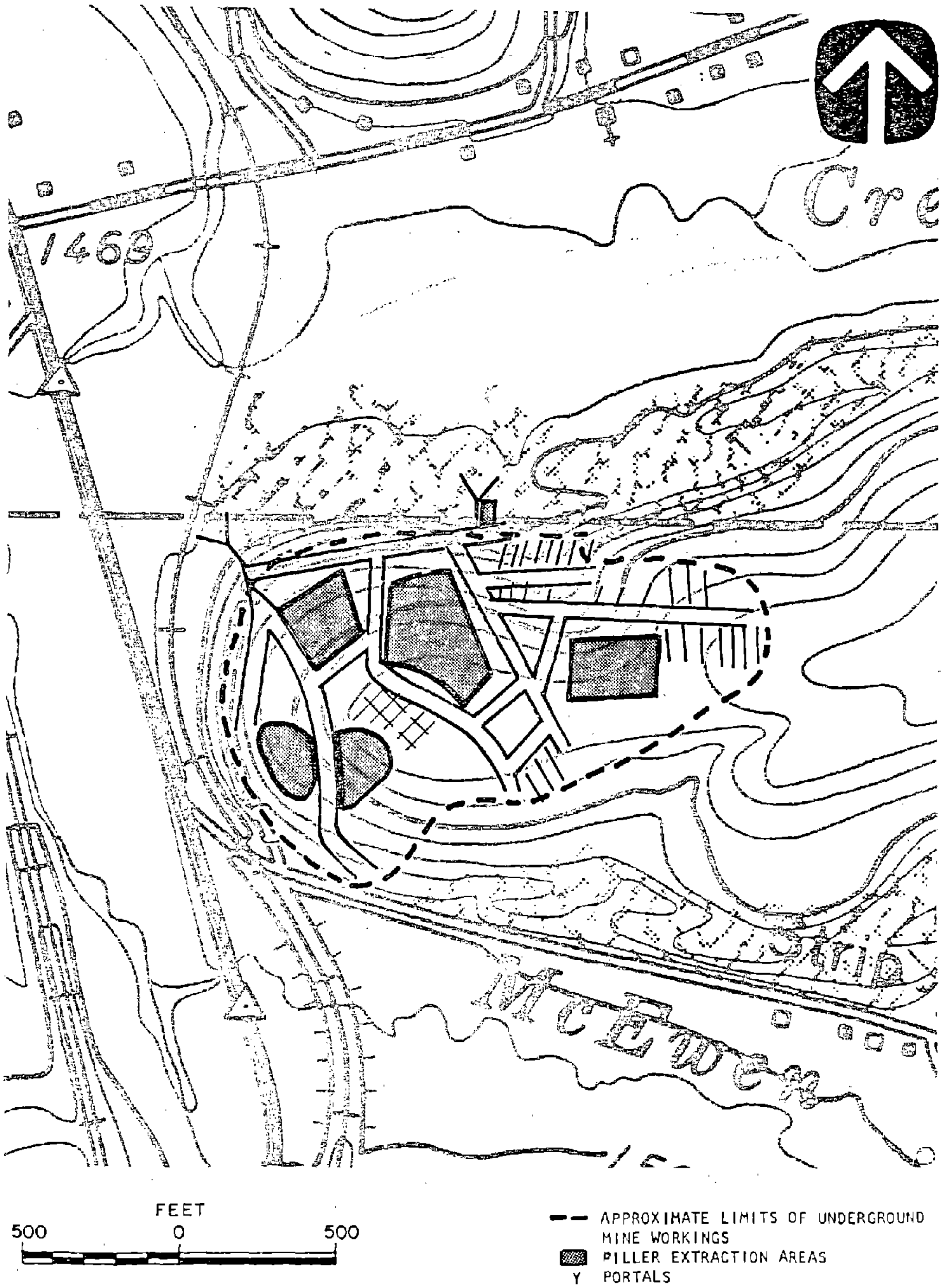


FIGURE 19 UNDERGROUND MINE WORKINGS AT SITE NO. 5

TABLE 10 - Stream Quality Data for Site No. 5[a]

SAMPLING SITE NUMBER	DATE	pH	FLOW CFS	ACIDITY		ALKALINITY		FERROUS IRON		TOTAL IRON		SULFATES	
				PPM	#/DAY	PPM	#/DAY	PPM	#/DAY	PPM	#/DAY	PPM	#/DAY
1	5-19-72	6.6	18.85	3	305	8	813	0.10	10	0.30	30	21	2,134
	7-06-72	8.0	35.65	3	577	14	2,691	0.12	23	0.59	113	32	6,150
	4-24-74	5.5	22.15	6	717	12	1,433	0	0	0.30	36	200	23,884
	11-11-74	5.0	12.08	4	261	6	391	0	0	0	0	300	19,538
2	5-19-72	7.5	1.04	6	34	93	521	0.08	0	0.27	2	86	482
	7-06-72	7.7	1.21	6	39	95	620	0.13	1	1.03	7	63	411
3	5-29-72	6.9	0.15	3	2	34	27	0.03	0	0.32	0	82	66
	7-06-72	7.0	0.30	7	11	35	57	0.08	0	0.65	1	56	91
4	5-19-72	6.2	18.61	4	401	6	602	0.12	12	0.39	39	24	2,408
	7-06-72	6.5	37.72	9	1,830	13	2,644	0.16	33	0.75	153	30	6,101
	4-24-74	5.6	23.44	6	758	12	1,516	0	0	0.35	44	45	5,687
	11-11-74	5.3	17.60	4	380	6	569	0	0	0	0	325	30,839
5	5-19-72	7.4	20.83	3	337	29	3,257	0.20	22	0.44	49	140	15,722
	7-06-72	6.7	48.46	13	3,396	32	8,360	0.13	34	1.18	308	170	44,415
6	5-19-72	7.1	28.36	4	612	40	6,116	0.02	3	0.45	69	176	26,910
	7-07-72	7.6	31.03	3	502	39	6,524	0.06	10	1.60	268	210	35,172
7	5-19-72	6.0	5.96	4	129	4	129	0.08	3	0.23	7	33	1,060
	7-07-72	6.3	16.24	9	788	8	700	0.13	11	0.55	48	63	5,516
7A	4-24-74	5.5	15.40	2	166	8	664	0	0	0.30	25	20	1,661
	11-11-74	5.1	10.85	4	234	4	234	0	0	0	0	250	14,624
8	5-19-72	7.3	56.80	3	919	21	6,431	0.06	18	0.26	80	78	23,886
	7-07-72	6.5	88.33	6	2,857	21	10,001	0.22	105	0.96	457	92	43,816
	4-24-74	6.0	58.12	2	627	40	12,534	0	0	0.50	157	70	21,934
	11-11-74	5.8	41.17	4	888	22	4,883	0	0	0.50	111	275	61,040
9	5-19-72	3.2	0.36	108	210	0	0	0.22	0	7.5	15	66	128
	7-06-72	3.8	1.20	166	1,074	0	0	0.42	3	9.6	62	410	2,653
	4-24-74	3.2	1.20	86	556	0	0	0	0	6.60	43	280	1,811
9A	11-11-74	3.0	0.11[b]	244	145	0	0	7.84	5	8.1	5	950	563
10	7-07-72	6.3	0[c]	137	—	24	—	43.2	—	67.7	—	200	-

[a]Refer to Figure 20 for Locations of Sampling Sites.
[b]Estimated flow from drain outlet pipe.
[c]Stagnant pool.
Source: Pennsylvania Department of Environmental Resources.

TABLE 11 - Observation Well Data for Site No. 5

	NO. 1 BACK OF DRIFT T.C. 1499.48			NO. 2 BETWEEN TWO TRENCHES T.C. 1494.9			NO. 3 OUT BY TRENCH T.C. 1482.2			
DATE	FEET TO H_2O	H_2O ELEVATION	pH	FEET TO H_2O	H_2O ELEVATION	pH	FEET TO H_2O	H_2O ELEVATION	pH	REMARKS
10/29/75	8.0	1,491.48	4.0	3.8	1,491.1	4.0	6.1	1,476.1	5.5	
11/21/75	6.7	1,492.78	4.5	2.5	1,492.4	4.0	5.8	1,476.4	5.5	
2/13/76	5.0	1,494.48	4.5	0.0	1,494.9	4.0	5.9	1,476.3	5.0	H20 from #2 hole 125 gpm
2/20/76	3.0	1,496.48	4.5	0.0	1,494.9	4.5	4.8	1,477.4	4.5	H20 from #2 hole 175 gpm
2/21/76	Mine blowout on side of hill from project - Flow approximately 400 gpm, elevation approximately 1,487.89 feet.									
2/22/76	Blowout flow approximately 300 gpm.									
2/23/76	6.7	1,492.78	4.5	2.7	1,492.7	4.0	6.35	1,475.9	4.5	Blowout flow approximately 250 gpm
2/25/76	8.0	1,491.48		3.6	1,491.3		6.7	1,475.5		Blowout flow approximately 200 gpm
2/27/76	9.0	1,490.48	4.5	4.65	1,489.3	4.5	6.95	1,475.25	5.5	Blowout flow approximately 300 gpm
3/05/76	10.1	1,489.38	4.5	5.6		4.5	6.95	1,475.25	5.5	Blowout flow approximately 250 gpm
3/09/77	9.8		4.0	5.6		4.0	7.0		5.5	Blowout flow approximately 300 gpm pH 4.0
3/21/77	8.5		4.0	1.5		4.0	6.4		5.5	Blowout flow approximately 200 gpm ph 4.0
4/04/77	2.7		4.0	0.0		4.0	5.3		5.5	Flow from #2 = 50 gpm Flow at blowout = 250
4/27/77	10.05		4.0	5.75		4.0	7.45		5.0	No seepage at blowout 90% backfilled
5/04/77	9.6		4.5	5.7		4.0	7.5		6.0	No seepage at blowout 100% backfilled
5/13/77	8.75		4.0	4.55		4.0	7.55		6.0	No seepage at blowout 100% backfilled
5/23/77	8.15		4.0	4.25		4.0	7.45		5.5	No seepage at blowout 100% backfilled
6/08/77	8.3		4.0	4.36		4.0	7.33		6.0	No seepage at blowout 100% backfilled
6/16/77	8.7		4.0	4.7		4.0	7.5		6.0	No seepage at blowout 100% backfilled
7/11/77	9.5		4.0	5.6		4.0	7.2		5.5	No seepage at blowout 100% backfilled
8/03/77	9.8		4.0	5.7		4.5	7.4		6.0	No seepage at blowout 100% backfilled
8/24/77	9.5		4.0	5.8		4.5	7.6		6.0	No seepage at blowout 100% backfilled
9/30/77	9.7		4.0	5.9		4.5	7.8		6.0	No seepage at blowout 100% backfilled
12/15/77	3.65		4.0	0.0		4.5	5.35		4.5	Flow from #2 hole approximately 40 gpm
4/18/78	2.2		4.0	0.0		4.0	6.8		4.0	Flow from #2 hole

NOTE: Bottom coal elevation 1,467 (Drift), Seepage area elevation 1,473 ±
SL-132-2-101.1

Source: Pennsylvania Department of Environmental Resources.

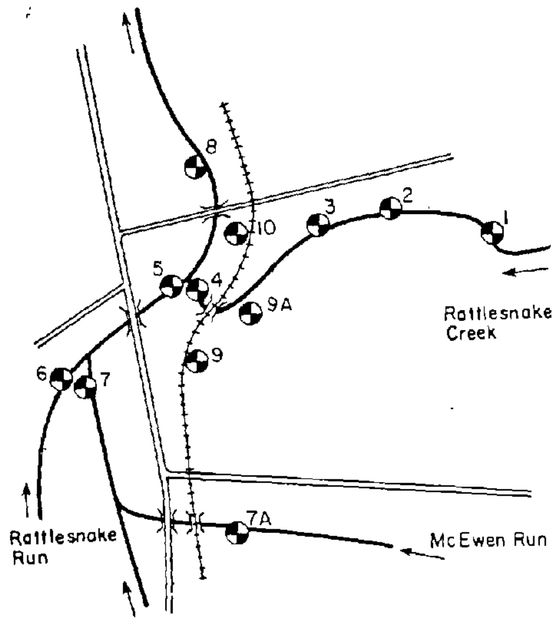


FIGURE 20 LOCATION MAP OF STREAM QUALITY SAMPLING POINTS FOR SITE NO. 5
(REFER TO TABLE 10 FOR ANALYSES)

FIGURE 21 FRENCH DRAIN DISCHARGE INTO RATTLESNAKE CREEK AT SITE NO. 5

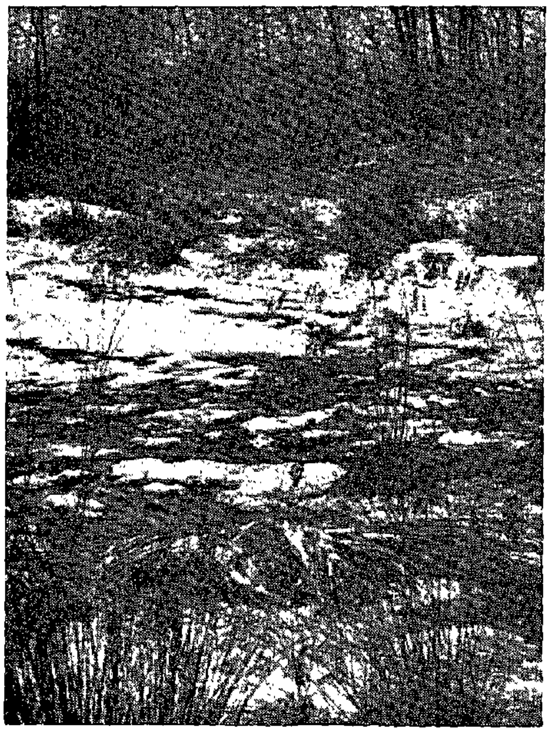


FIGURE 22 DEPRESSION WITH SEEPAGE AT SITE NO. 5

FIGURE 23 SEEPAGE AREA IN FRONT OF SEALED PORTAL AT SITE NO. 5 VIEWED FROM ABOVE THE SEAL

The present status of Site No. 5 was determined during a site investigation on March 13, 1980. The site was lightly covered with snow and the temperature was approximately 40°F. The first sample was taken from a pipe discharging into the creek. This is the discharge being collected by the French Drain System. Figure 20 shows this discharge. Closer to the highwall, a depression was encountered where water was collecting (see Figure 21). A second sample was obtained from this location. The third sample was obtained from the seepage area in front of the sealed portal shown in Figure 22. The three observation wells were located and water was present in all three. Vegetation on the site was very limited, but due to the season, it was difficult to determine the cause. The location of sample points is shown in Figure 23. Water quality analyses for the three samples are shown in Table 4. The samples taken indicate approximately the same quality of water from all three locations, with the exception of suspended solids in sample 12. This may be misleading, however, since the flow was so small that debris from the bottom may have been obtained with the sample.

The mine closure attempt at Site No. 5 was successful in inundating the mine, however, a blowout did occur in the overburden after a head of 30 feet developed. The blowout was closed by backfilling and the head has since returned to the 30 foot level.

Since much of the outcrop was removed by a stripping operation after the underground mine was abandoned, the mine map does not accurately depict the amount of outcrop barrier remaining (see Figure 19). The amount of barrier before strip mining appears to be approximately 100 feet according to the underground mine map (Figure 19). There is only a slight dip over the mine, but the structural low point is at the sealed portal and along the outcrop.

Seepage is still emanating from the underground workings around the portal area but the flow is not high. The field inspection revealed water only 3 feet below the surface at observation hole 1 which indicates a head of 30 feet on the outcrop barrier and portal seal (Figure 24).

A summary of the water quality data for Site No. 5 is shown in Table 12. Unfortunately, not all samples were tested for the same parameters. According to the pH values, the least acidic water is found in front of the slurry trench. The site does not exhibit an unusually high quality of water in the mine

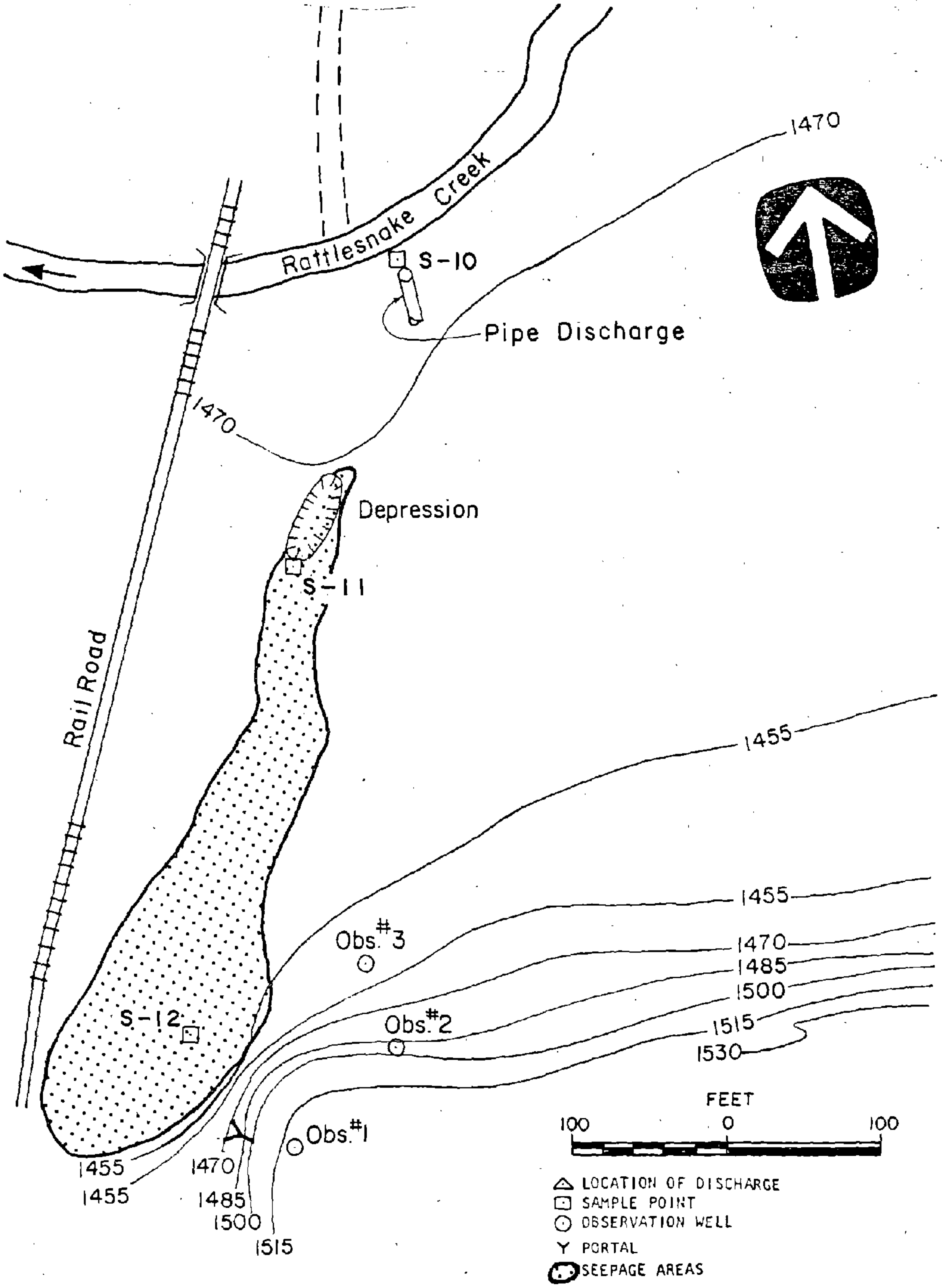


FIGURE 24 LOCATION OF DISCHARGES, SAMPLING POINTS, AND OBSERVATION WELLS FOR SITE NO. 5

pool as the other sites studied. It is interesting to note that the lowest pH is found in the depression that is at the same elevation as the bottom of the coal seam. High iron levels are also found here. The only sulfate analysis available is for the French Drain discharge. The drain is not an abatement measure, but rather a means of diverting surface water from the spoil piles and drying the area so that vegetation could become established. 950 mg/l of sulfates are shown for this discharge which averages 50 gallons per minute.

TABLE 12 - WATER QUALITY SUMMARY FOR SITE NO. 5[a]

LOCATION OF DATA COLLECTION[b]	pH	TOTAL IRON mg/l	SULFATES mg/l
Mine Pool (determined from Obs. 1)	3.58	27.7	-
Seepage from Portal	3.2	12.3	-
Depression Seepage (at coal bottom elevation)	2.81	26.0	-
French Drain Discharge	3.0	8.1	950
Obs. 2	4.2	-	-
Obs. 3	5.4	-	-

[a]Data presented represents an average of available records.

[b]Location of Data Collection points is shown in Figure 24.

Site No. 6

Site No. 6 is located in Wyoming County, West Virginia (Figure 25). The mine opened in 1969 in the Ben's Creek Coal Seam which has a local dip of 1° to the west. Mining was developed for a distance of 2,600 feet from the entry followed by partial and total pillar extraction. One section of the mine on the downdip side, was driven all the way to the outcrop and then pillared back. Abandonment took place on February 15, 1973 after which the entries were sealed. The seals were placed

FIGURE 25 _ LOCATION OF BLOWOUT AND SAMPLING POINTS AT SITE NO. 6

at an updip location and were not designed to sustain any head. Since there was no monitoring system or observation well installed, there was no way of predicting or warning of an accumulation of water.

In March, 1975 a blowout occurred through the overburden at the location where the workings were driven to the downdip outcrop. A contributing factor to the blowout was probably overburden fracturing due to the pillar extraction and roof cave-in. An inspection by MSHA revealed that a 15 to 20 barrier of coal had been left at the outcrop. The maximum amount of head which was possible by the geometry of the mine is roughly 50 feet within the mine workings alone. However, an adjacent mine is thought to have interconnected workings that may allow drainage to flow into the Site No. 6 mine. According to the MSHA report, the level of groundwater head probably exceeded the 50 foot level due to two months of unusually high precipitation.

When the overburden washed free, the resulting outburst of water fell from a vertical height of 200 feet down a hillside sloping roughly 30 degrees from the horizontal. The force of the water washed out a stream channel 12 feet down to the bedrock.

The present status of Site No. 6 was determined during a field investigation on March 18, 1980. The site of the blowout is now a large opening in the hillside as shown in Figure 26. Since the original blowout, subsequent deterioration and weathering of the exposed rock has caused additional rock falls. From the opening, mine timbers are visible only 10 or 15 feet from the surface. Figure 27 illustrates the mine pool.

Several water quality samples were obtained. The first was taken in the mine pool shown in Figure 27. There is some ground-water influence as shown by the drops of water falling into the pool. The water analysis from this location showed a pH of 3.25, and 6.6 mg/l of total iron. Another sample was taken at a location outside of the mine pool where the discharge starts flowing down the hill (see Figure 28). The pH is unchanged, however, total iron and suspended solids are noticeably higher. Two additional samples were also obtained along the drainage path as it proceeded down the hillside (Figure 29). No significant change in pH was noted, however, iron levels at this point are approaching the OSM effluent limitations. Figure 25 illustrates the location of the blowout and the sampling points for the site.

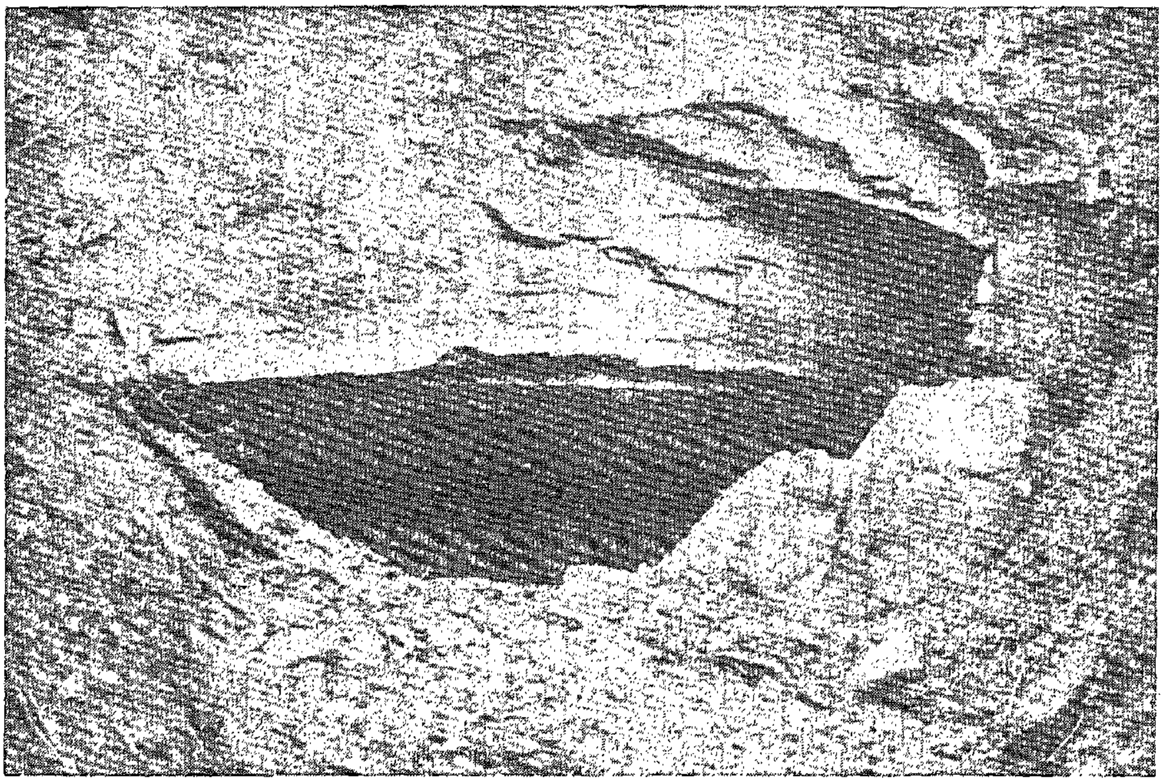


FIGURE 26 LOCATION OF BLOWOUT AT SITE NO. 6

FIGURE 27 MINE POOL AT SITE NO. 6

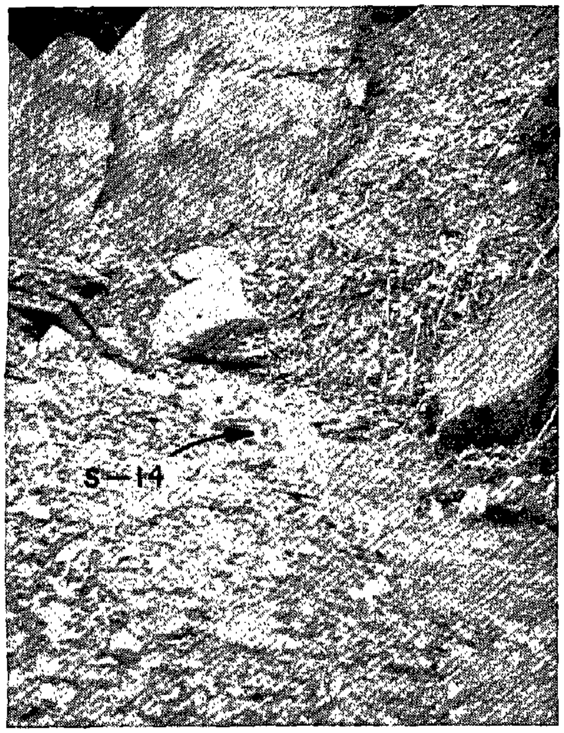


FIGURE 28 LOCATION OF SAMPLE 14 AT SITE NO. 6

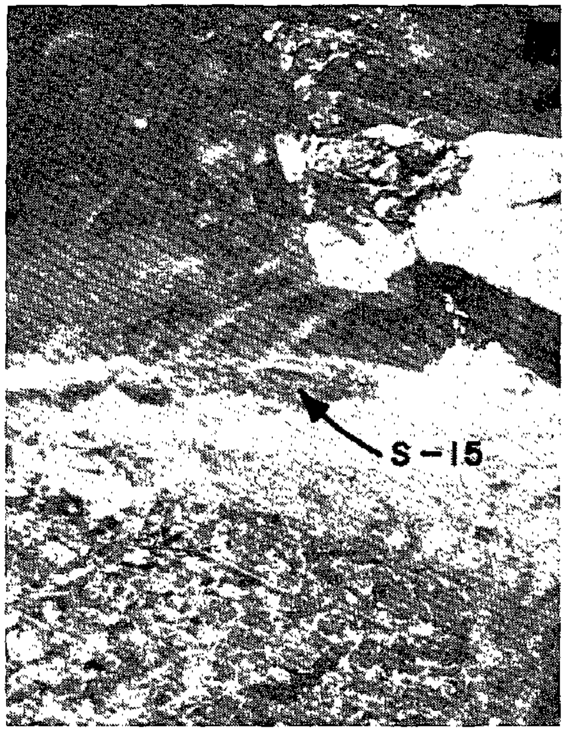


FIGURE 29 LOCATION OF SAMPLE 15 AT SITE NO. 6

The closure attempt at Site No. 6 was not designed to inundate the mine workings, but shows that this type of mine may be a potential hazard. The structural low point of the mine is along the west outcrop and was undisturbed by mine entries. The hydrostatic head that resulted in a blowout cannot be determined due to large quantities of precipitation prior to the burst, however, it has been estimated that 50 feet of head was likely . The barrier pillar remained stable with only a 15 to 20 feet width supporting the head. The overburden, however, was so weakened by fracturing and rock falls, that a blowout occurred through it.

A summary of the water quality data for Site No. 6 is presented in Table 13. Only slight differences are noted between the mine pool and the mine discharge. This is reasonable since there is no barrier. Some contamination from surface water may have affected the water quality at the time of sampling, however.

TABLE 13 - WATER QUALITY SUMMARY FOR SITE NO. 6[a]

LOCATION OF DATA COLLECTION[b]	TOTAL IRON mg/l	MANGANESE mg/l	SULFIDES mg/l
Mine pool	3.25	6.6	0.02
Mine Discharges	3.3	9.2	.016

[a]Data presented represents an average of available records.

[b]Location of data collection points is shown in Figure 25.

ANALYSIS OF DESIGN PARAMETERS

Based upon the results of the literature search and field reconnaissance, engineering analyses were performed to generate recommendations for designing outcrop barriers. Three types of analyses were made; the first was a seepage analysis, the second an overburden blowout analysis, and the third a wedge stability analysis. Where possible, actual field data are presented for comparison with analytical results.

Seepage Analysis

In order to establish a general relationship between barrier width and flow rates, a simplified analysis based on Darcy's Law and a computer seepage analysis were performed. The computer analysis generated flow lines, discharge velocities, and phreatic surfaces that provided input into the stability analysis as well.

The selection of the computer model was based on its ability to satisfactorily represent a steady state condition in which post-mining inundation and restoration of the water table was achieved. A porous media model was chosen to represent the steady state condition of the saturated rock mass.

The nature of the Eastern Bituminous Region is such that flow in coal and rock are dominated by fractures. The pre-mining conditions are described by primary fractures, while the post-mining conditions include both primary and secondary fracture systems. In order to account for fracture flow, the permeability values utilized in the analyses reflect a range consisting of primary permeability, secondary permeability, and also including weathered rock permeability. Fourteen cases were modeled to show the influence of various permeability combinations and boundary conditions on the resulting flow patterns.

Program Description and Applications

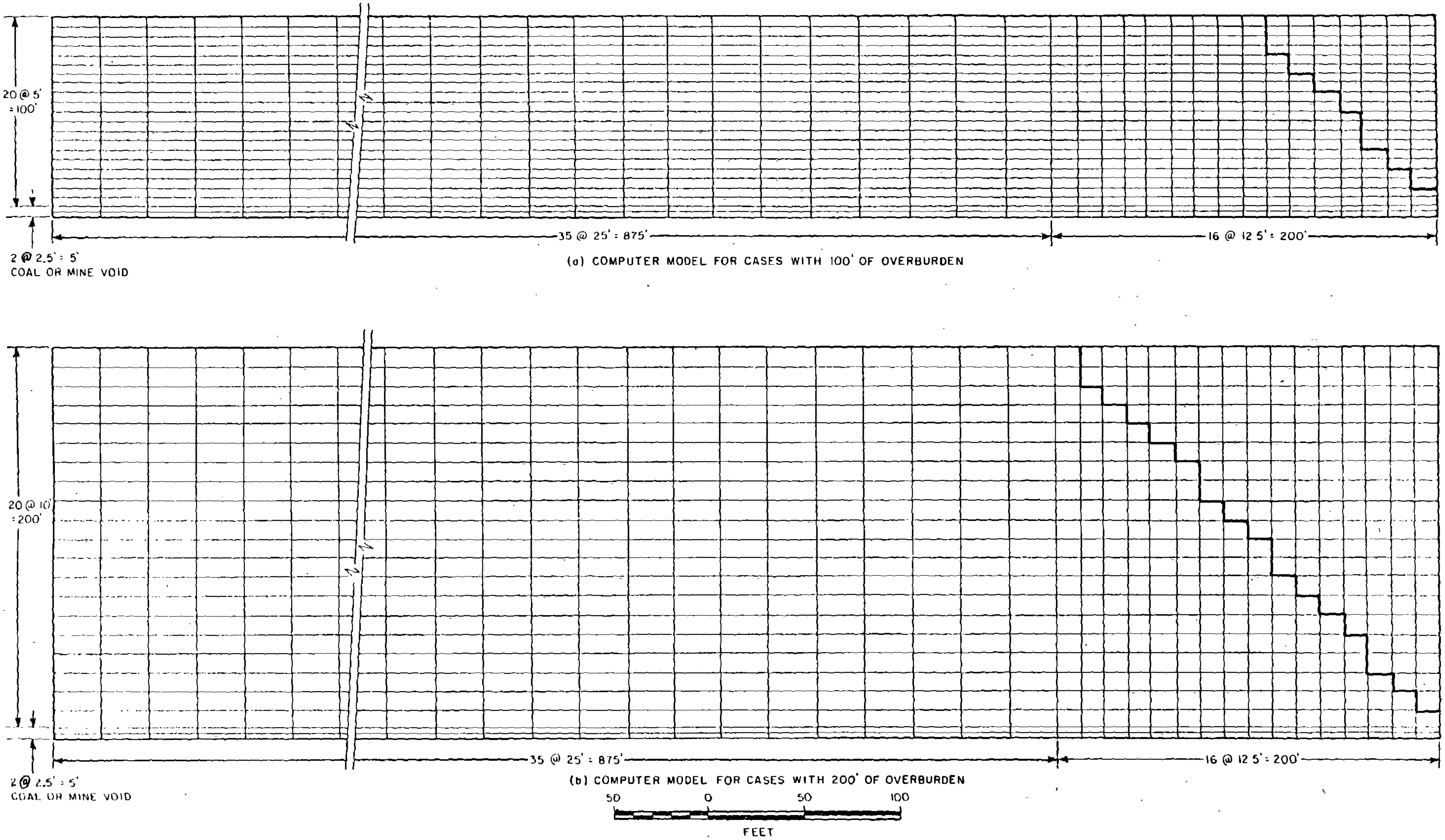
The steady-state seepage of ground water through a coal seam and surrounding host rock was calculated using the program TARGET (Transient Analyser of Ground-water Flow and Effluent Transport); a finite difference code that can predict coupled ground-water flow in saturated porous media.

A brief description of TARGET is presented here; however, a more detailed version is presented as Appendix J. A combination of the experimentally justified Darcy's law and the mass balance of a small volume yields a partial differential equation for 2-dimensional ground-water flow through porous media (Appendix J). TARGET utilizes this equation in an implicit finite-difference scheme to predict head distributions in time and space. The region being studied is divided into control volumes within which the hydraulic properties of permeability and storage coefficient are assumed to be homogeneous and are specified. The permeability can be different in the x and y directions within each volume. The hydraulic properties of the materials together with the appropriate boundary conditions (in the cases studied these are zero flux boundaries and fixed head boundaries) define the ground water flow problem uniquely and allow a numerical solution of the flow equation. The program achieves this by solving a matrix set of equations each of which describes mass balance within a zone for a particular small time step. The matrix of heads is then updated and the process is repeated for subsequent time steps. In the cases considered, only the steady-state conditions were required and the program iterated until minimal changes in head occurred.

The finite difference grid consists of a system of orthogonal intersecting lines. Figure 30 illustrates the two grids used in this analysis. The two grids have the same mesh, but with different dimensions so that two overburden thicknesses have been modeled. The spacing between grid lines (which determine the size of the control volumes), as well as the small finite time steps themselves, influence the convergence of the procedure and the accuracy of the resulting solutions. However, the implicit nature of the scheme permits the grid to be nearly unconditionally stable for steady-state solutions. The discretization permits the geometry of particular regions of interest to be modelled more carefully (finer discretization) and allows hydraulic properties to vary from place to place. To correctly model composite materials, the properties across boundaries of dissimilar materials are averaged geometrically. Fine details in areas where the heads may be changing rapidly are obtained by solving more local equations.

Generation of a Grid

The finite difference grid is generated according to the conditions which define the model. For this particular analysis, the model consisted of 5 feet of coal covered by 2 different overburden depths at a sidehill slope of 45°. The coal was assumed to have no dip. Figure 30 shows the resulting grids. There are 102 cells in the coal seam each having a thickness of 2.5 feet and 899 cells in the overburden each having a thickness of either 10 feet or 5 feet. Around the entire grid there is a dimensionless cell representing the outer boundaries of the grid.



(a) COMPUTER MODEL FOR CASES WITH 100' OF OVERBURDEN

(b) COMPUTER MODEL FOR CASES WITH 200' OF OVERBURDEN

FIGURE 30 FINITE DIFFERENCE GRIDS

Computer Model Assumptions

The following assumptions were made for the numerical models considered:

- The hydraulic properties of the overburden and coal are homogeneous within each material and remain constant for all heads considered.
- The seepage is due only to incompressible Darcian flow through the rock.
- The total head along the left-hand boundary of the grid (Figure 30) remains constant and equals 105 feet or 205 feet depending on the case under consideration. (The datum is taken to be the base of the coal seam).
- The total head in the mined-out region remains constant and equals 105 feet or 205 feet depending on the case under consideration.
- The clay seam underlying the coal seam is considered to be an impervious boundary.

Input Requirements

A complete list of TARGET input requirements is presented in Appendix K; the TARGET Input Guide. Since the program TARGET is also capable of analysing heat transfer and mass transfer, many of the variables were not utilized. The basic site parameters which are needed to run the program are horizontal and vertical permeabilities, storage coefficient, porosity, the value of maximum hydraulic head, and the geometry of the strata.

Since this analysis is based on data obtained in the literature search rather than actual field data, the most realistic and representative data was sought. The geometry of the model is simplistic as shown in Figure 30. A 5 foot coal seam thickness is assumed with a 45° sidehill slope. These two factors remain constant in all fourteen analyses. In order to find representative values of permeability and porosity, existing documentation was relied upon. A compilation of recorded values in the Eastern Bituminous Region resulted in the permeability ranges given in Table 14.

The values given in Table 14 are horizontal permeabilities. The permeability contrast was chosen to be 4:1 with the horizontal permeability being four times the vertical permeability. According to Brown and Parizek, 1971, core analyses of coal strata show that the horizontal to vertical permeability contrast is 4:1 for intergranular flow in all strata. Since vertical permeabilities were not determined in the field, a comparison could not be made between field and laboratory data. Brown and Parizek did, however, note several orders of magnitude difference between the core analyses and the field pumping data with the field tests giving the higher values. For the purpose of this seepage analysis, a 4:1 horizontal to vertical permeability contrast was chosen as a representative value.

TABLE 14. Range of Horizontal Permeabilities for Seepage Analysis

Material	Premining --------------> Post Mining		
Coal	1.0 ft/day[a]	3.21 ft/day[b]	4.86 ft/day[c]
Overburden	0.01 ft/day[d]	0.74 ft/day[e]	4.25 ft/day[f]
Underclay	0.0005 ft/day[g]	-------	.013 ft/day[h]

[a]1 ft/day is given as the average permeability of the Upper Freeport Coal in Miller and Thompson, 1974. This also agrees with Brown & Parizek, 1971 where 5 gpd/ft^2 or 0.67 ft/day was used as an average coal permeability. This value was used to describe relatively undisturbed coal and was derived from field pressure injection tests.

[b]3.21 ft/day is given as the average permeability for the base of the Upper Freeport Coal in Miller and Thompson, 1974. This value was used to represent a more realistic value that took into account some fracturing within the coal and was derived from field pressure injection tests.

[c]4.86 ft/day is given as 2 darcy units in the SME Mining Engineering Handbook for dense rock with high fracture permeability. This value was used when weathering was taken into account as was the case for most outcrop barriers. It also corresponds to the limiting permeability value of 5 ft/day obtained from the field data.

[d]0.01 ft/day is given in Davis & DeWeist, 1970 as the pemeability of a dolomitic sandstone with silica and carbonate cement. This value was used to represent relatively undisturbed overburden.

[e]0.74 ft/day is given in Miller and Thompson, 1974 as the average permeability of shale with sandstone over solid coal and was derived from pressure injection tests.

[f]4.25 ft/day is given in Miller and Thompson, 1974 as the average permeability of shale with sandstone bridged through a height of 44 feet over a Lower Kittanning mine void. This value was used most commonly to represent fractured overburden above a mine void and was derived from field pressure injection tests.

[g&h]The underclay permeability values were obtained from Brown and Parizek, 1971. The premining value represents granular interstitial permeability, and the post mining represents the fractured permeability.

The porosity values for the various materials were determined by using representative moisture content and specific gravities from the Keystone Coal Industry Manual, 1980. For coal, a moisture content of 5% and a specific gravity of 1.3 were used to arrive at a porosity of 6.1%. For the overburden, a moisture content of 1.2% and a specific gravity of 2.6 were used to arrive at a porosity of 3.0%. Values for underclay were not determined because it was decided to model the floor material as an impermeable boundary. This again, is a conservative estimate and will increase the amount of flow through the coal and overburden. The storage coefficients were given the same value as the porosity. Since the model was an unconfined aquifer the porosity represents an upper limit for the storage coefficient.

Seepage Analysis Models

Fourteen cases were studied using the TARGET computer program. The various permeabilities, porosities, hydraulic heads, and widths of outcrop barriers are presented in Table 15. Note that four different outcrop barrier widths and two levels of hydrostatic head were analysed. These values were chosen as a conservative estimate of what might be expected in the Eastern Bituminous Region.

TABLE 15. - Conditions Analyzed with Target

CASE	OVERBURDEN PERMEABILITY*	COAL PERMEABILITY*	WIDTH OF OUTCROP BARRIER	HEAD IN MINE & ALONG LEFT-HAND BOUNDARY	OVERBURDEN POROSITY	COAL POROSITY
1	0.01 ft/day	1.0 ft/day	100 ft.	205 ft.	0.03	0.06
2	0.74	4.86	100	205	0.03	0.06
3	0.74	4.86	200	205	0.03	0.06
4	4.25	4.86	100	205	0.03	0.06
5	4.25	4.86	200	205	0.03	0.06
6	0.74	4.86	100	105	0.03	0.06
7	0.74	4.86	150	105	0.03	0.06
8	4.25	4.86	100	105	0.03	0.06
9	4.25	4.86	150	105	0.03	0.06
10	4.25	3.21	50	205	0.03	0.06
11	0.01	1.0	150	105	0.03	0.06
12	4.25	3.21	100	205	0.03	0.06
13	4.25	3.21	200	205	0.03	0.06
14	4.25	3.21	150	105	0.03	0.06

*Values shown are horizontal permeabilities. For all cases, the vertical permeability equals 0.25 times the horizontal permeability (Brown & Parizek, 1971).

Program Output

For the first ten cases analyzed, three separate outputs were produced consisting of one printout and two plots. The printout consists of: 1) a listing of the total hydrostatic head for each cell, 2) a listing of the total horizontal discharge from each cell's left face and 3) a listing of the total vertical discharge from each cell's bottom face. Table 16 summarizes the resulting discharges for each case. The plots consist of one showing the equipotential lines and the other showing the Darcy velocities for each cell. Figures 31 through 40 illustrate the results of the two plots for analyses 1 through 10. Cases 11, 12, 13, and 14 were performed to obtain additional data points for correcting the barrier widths and have only printouts as output.

TABLE 16. - Seepage Analysis Discharge Results

Case Number		1	2	3	4	5	6	7	8	9	10	11	12	13	14
Given Conditions:															
Overburden Permeability (ft/day)		0.01	0.74	0.74	4.25	4.25	0.74	0.74	4.25	4.25	4.25	0.01	4.25	4.25	4.25
Coal Permeability (ft/day)		1.0	4.86	4.86	4.86	4.86	4.86	4.86	4.86	4.86	3.21	1.0	3.21	3.21	3.21
Width of Outcrop Barrier (ft)		100	100	200	100	200	100	150	100	150	50	150	100	200	150
Hydrostatic Head (ft)		205	205	205	205	205	105	105	105	105	205	105	205	205	105
Discharge Results (per foot of outcrop): Through Coal Barrier	ft^3/day	0.05	0.23	0.13	0.22	0.16	0.14	0.11	0.16	0.14	0.30	.018	.146	.13	.10
	gpm	10.03	44.68	25.73	42.30	31.25	26.23	20.23	30.18	26.90	58.00	3.48	28.20	25.30	18.82
Through Overburden	ft^3/day	0.01	0.48	0.27	2.69	1.49	0.13	0.07	0.72	0.47	2.75	.001	2.53	1.45	.46
	gpm	1.29	92.14	51.20	517.21	287.36	24.50	13.68	139.12	90.04	529.00	.144	487.0	278.5	88.31
Total Flow from Seepage Face	ft^3/day	0.06	0.71	0.40	2.91	1.65	0.27	0.18	0.88	0.61	3.05	0.19	2.676	1.58	.56
	gpm	11.32	136.82	76.93	559.51	318.61	50.73	33.91	169.30	116.94	587.00	3.624	515.2	303.8	107.13

The location of the phreatic surface shown on Figures 31 through 40 was determined by an iteration process. An initial estimate was made and the input data was structured so this boundary was treated as a zero-flux boundary, that is, as a streamline. Then, when the steady state condition was calculated, a check was made on the values of head along the phreatic surface. If the total head was equal to the elevation head along the boundary, then the phreatic surface was located correctly. The finest overburden finite difference cells used were 5 feet high, hence the greatest accuracy along the phreatic surface that could be achieved was within ± 2.5 feet of the correct value. If the calculated heads were within 1 cell height, the phreatic surface location was considered to be acceptable.

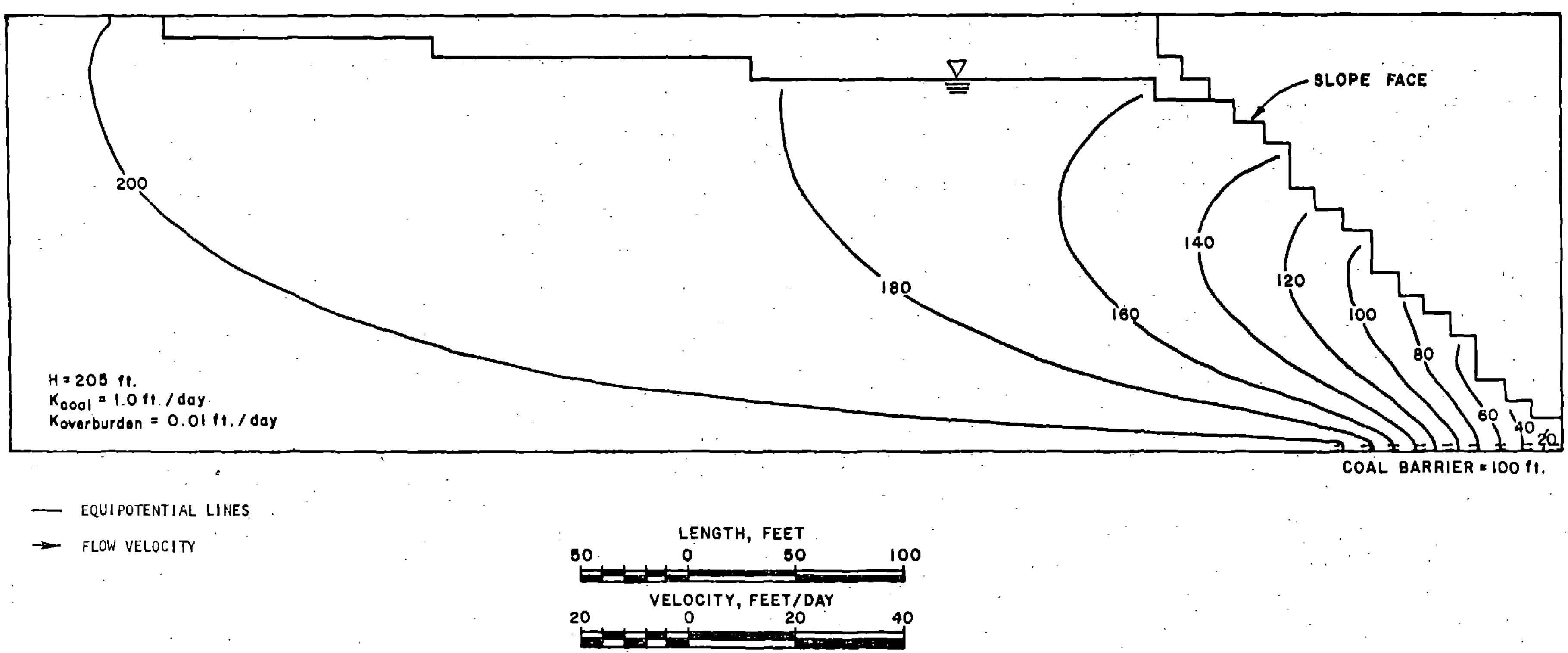


FIGURE 31 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 1

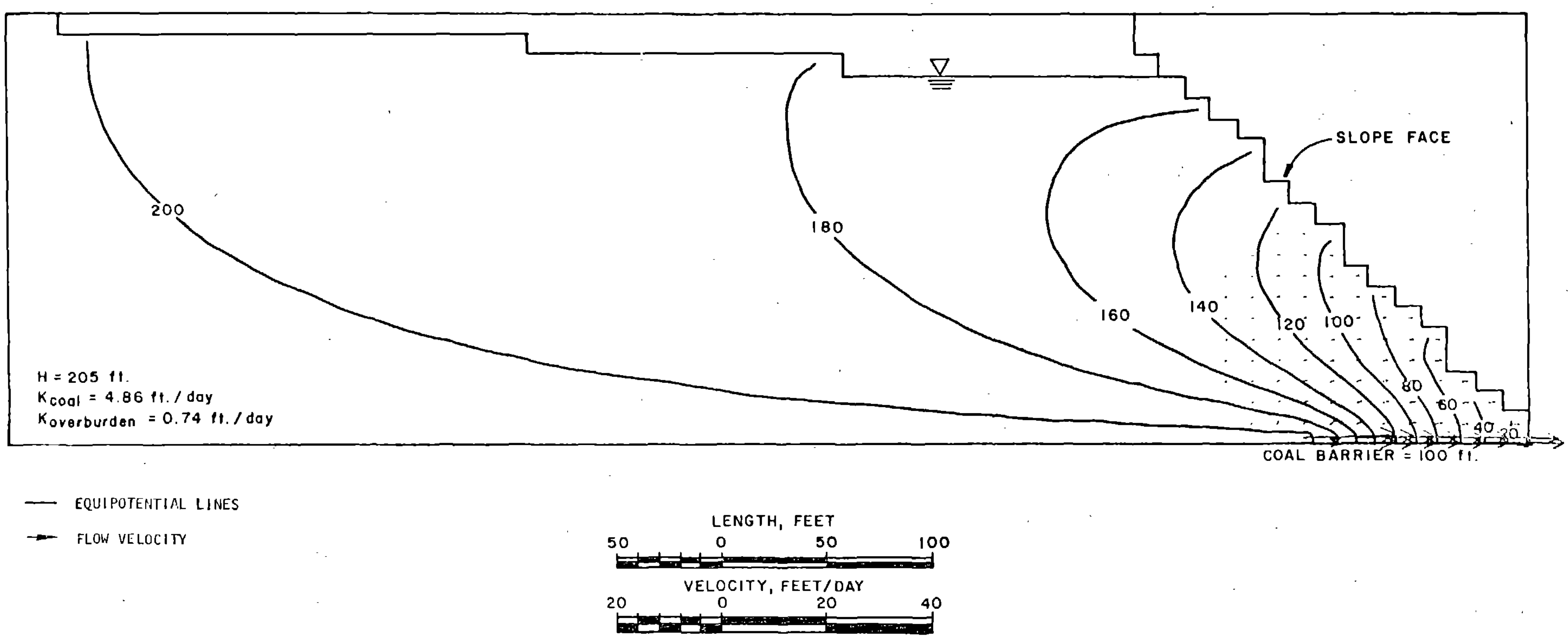


FIGURE 32 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 2

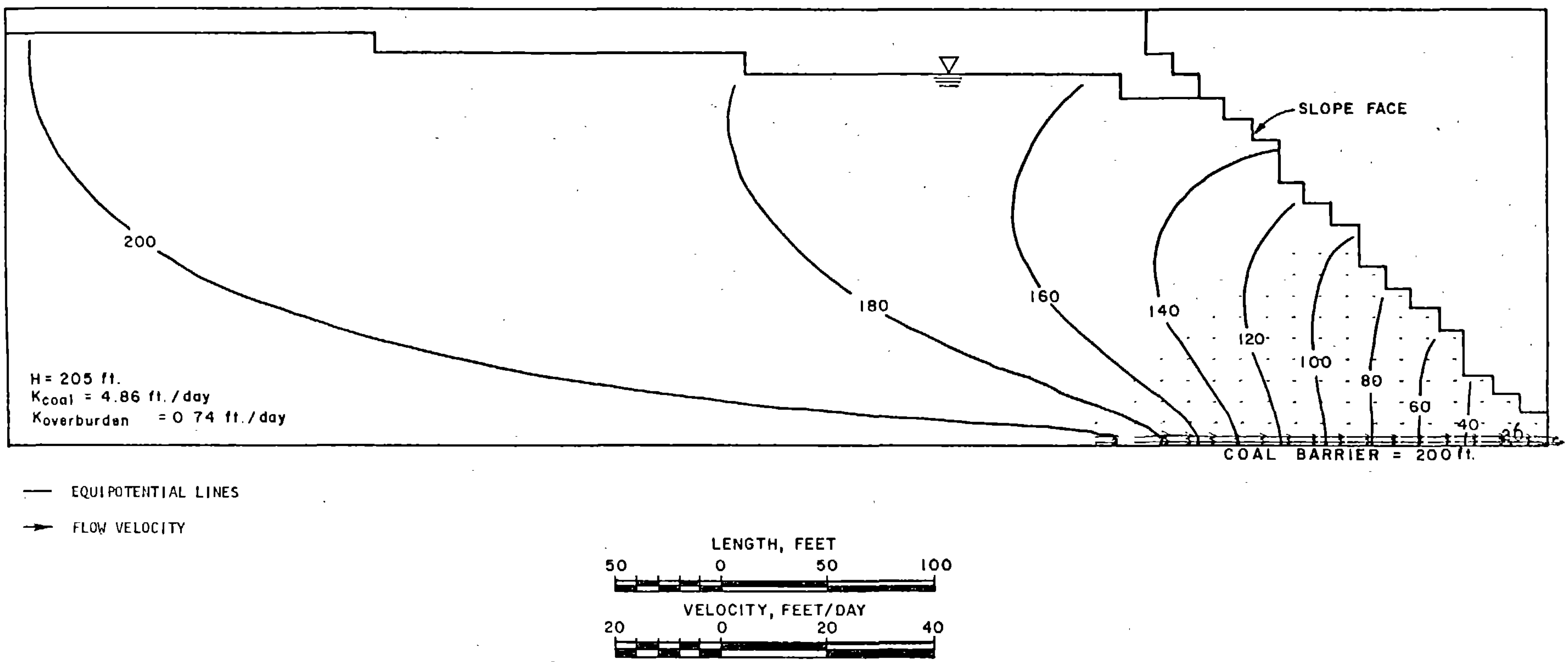


FIGURE 33 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 3

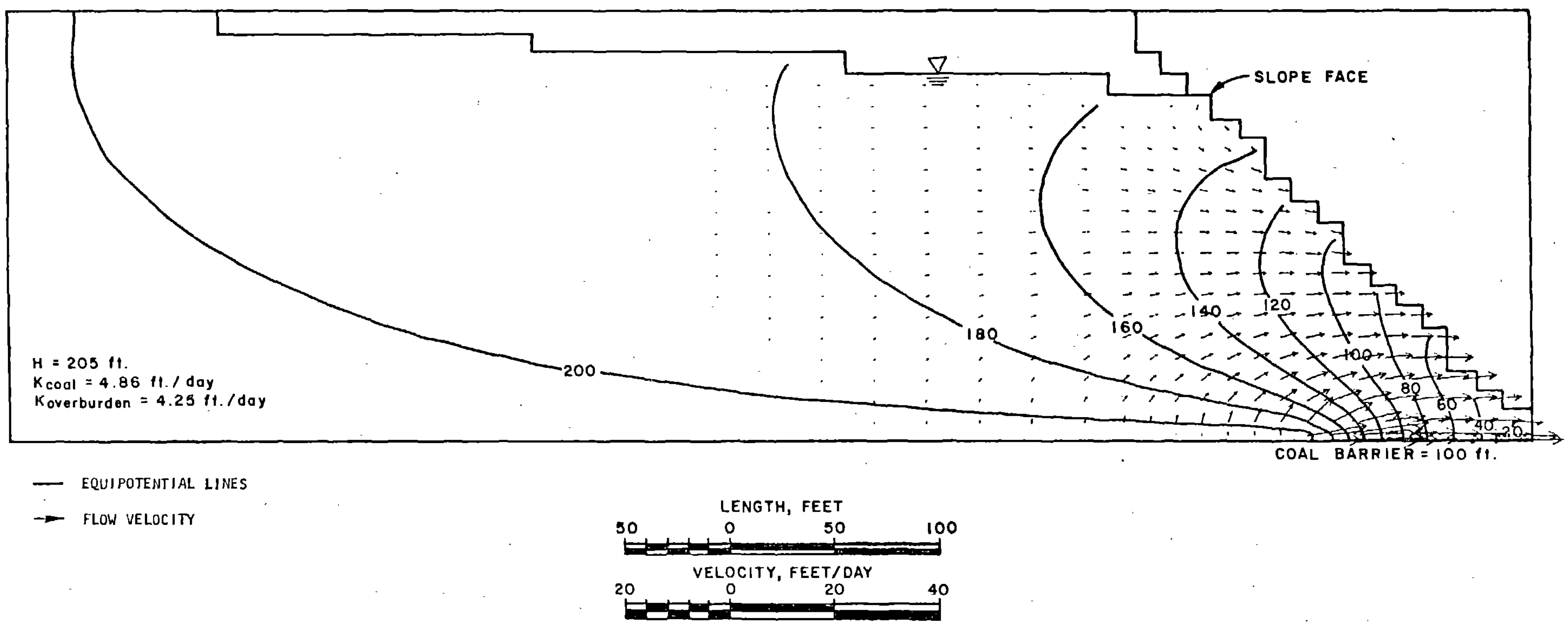


FIGURE 34 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 4

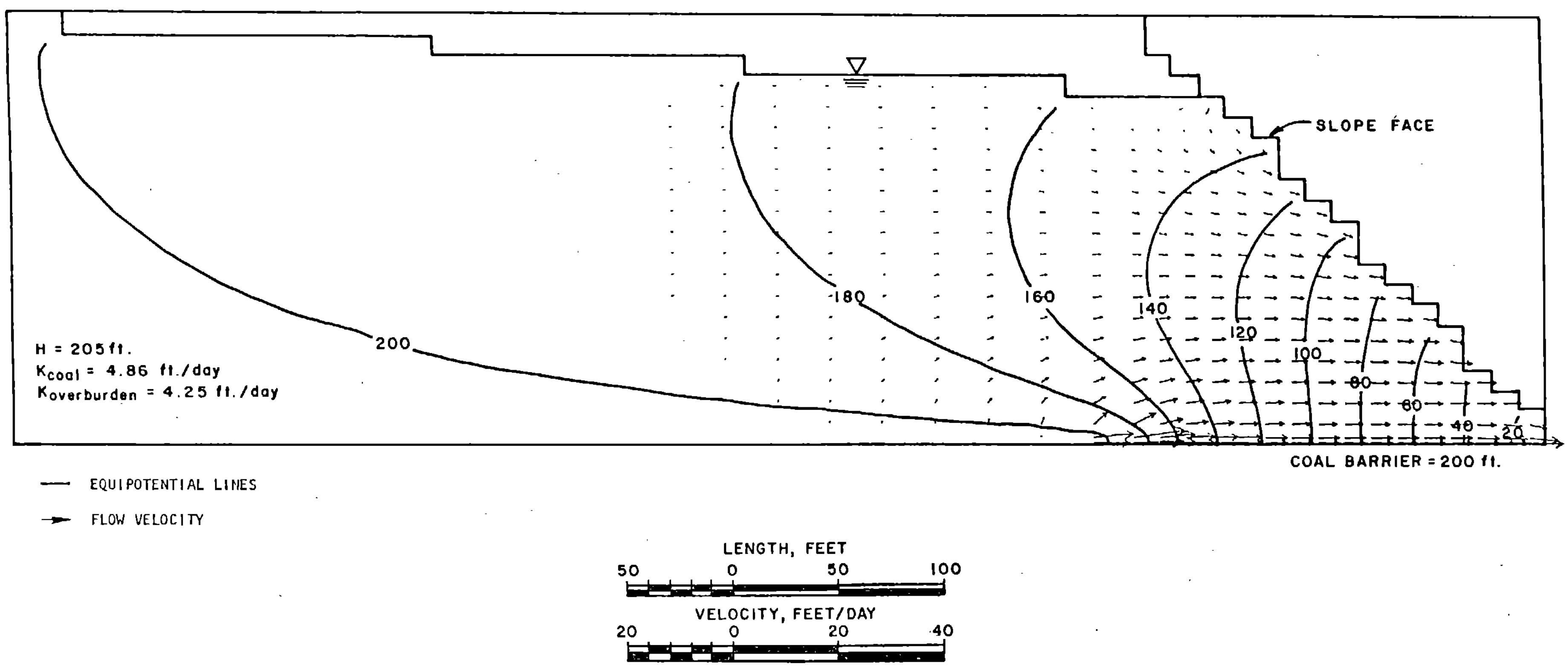


FIGURE 35 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 5

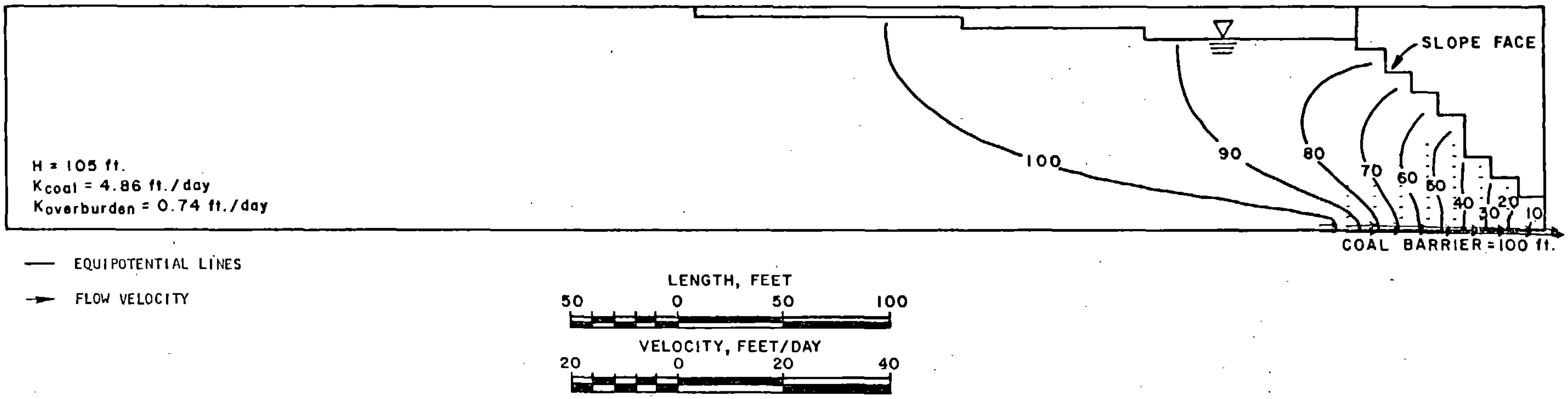


FIGURE 36 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 6

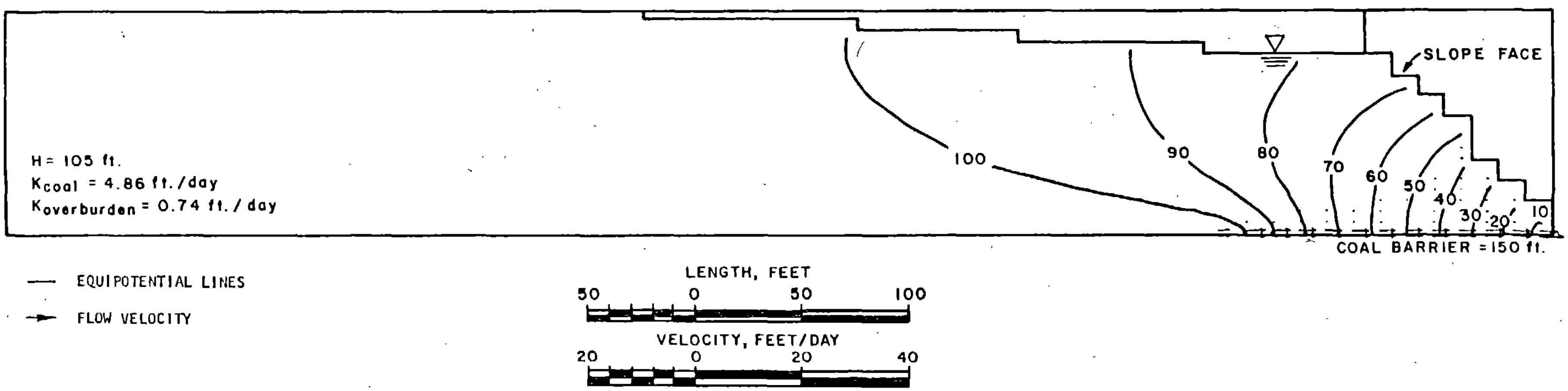


FIGURE 37 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 7

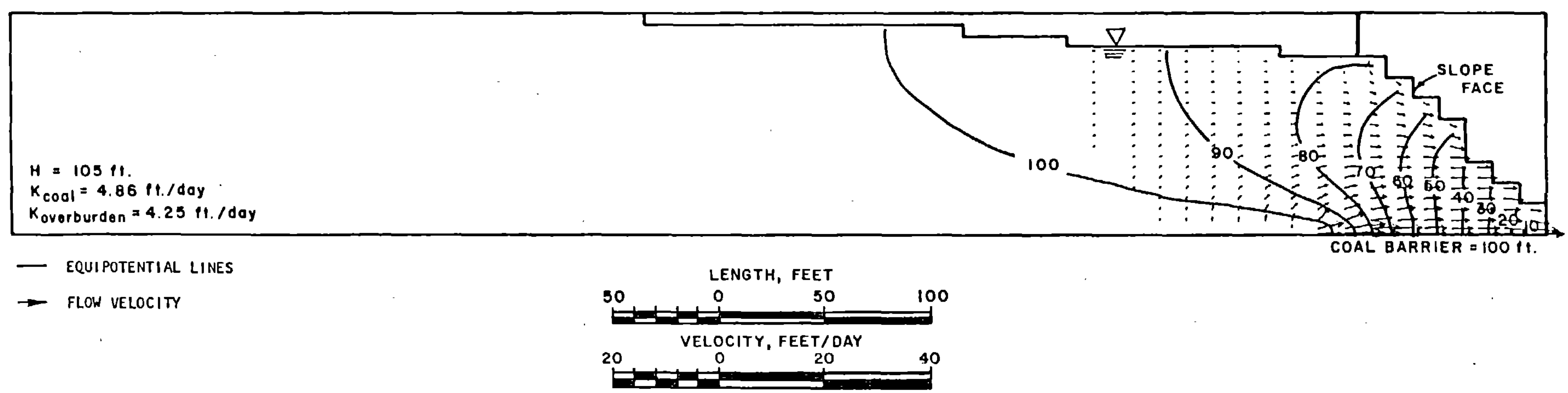


FIGURE 38 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 8

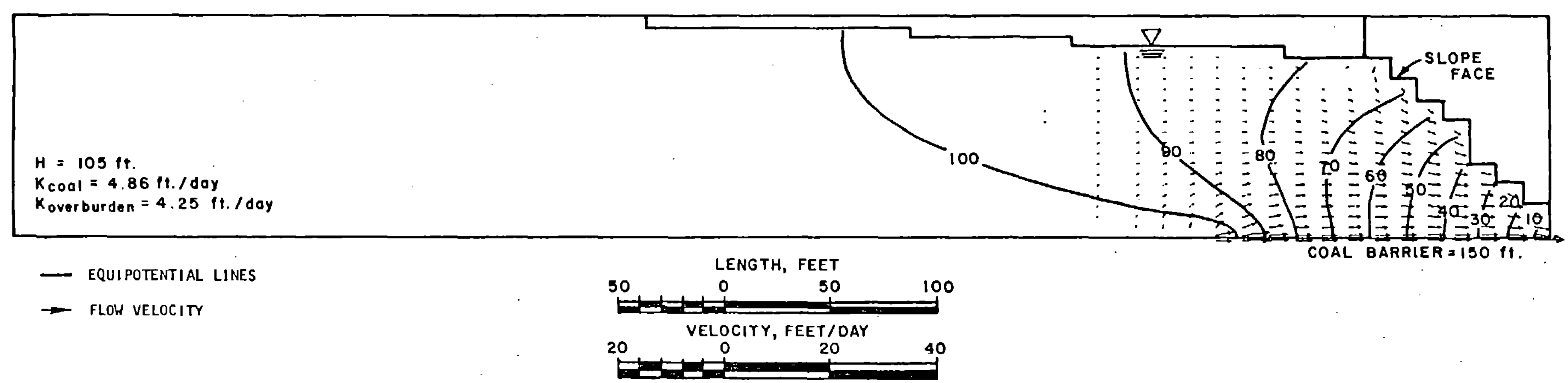


FIGURE 39 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 9

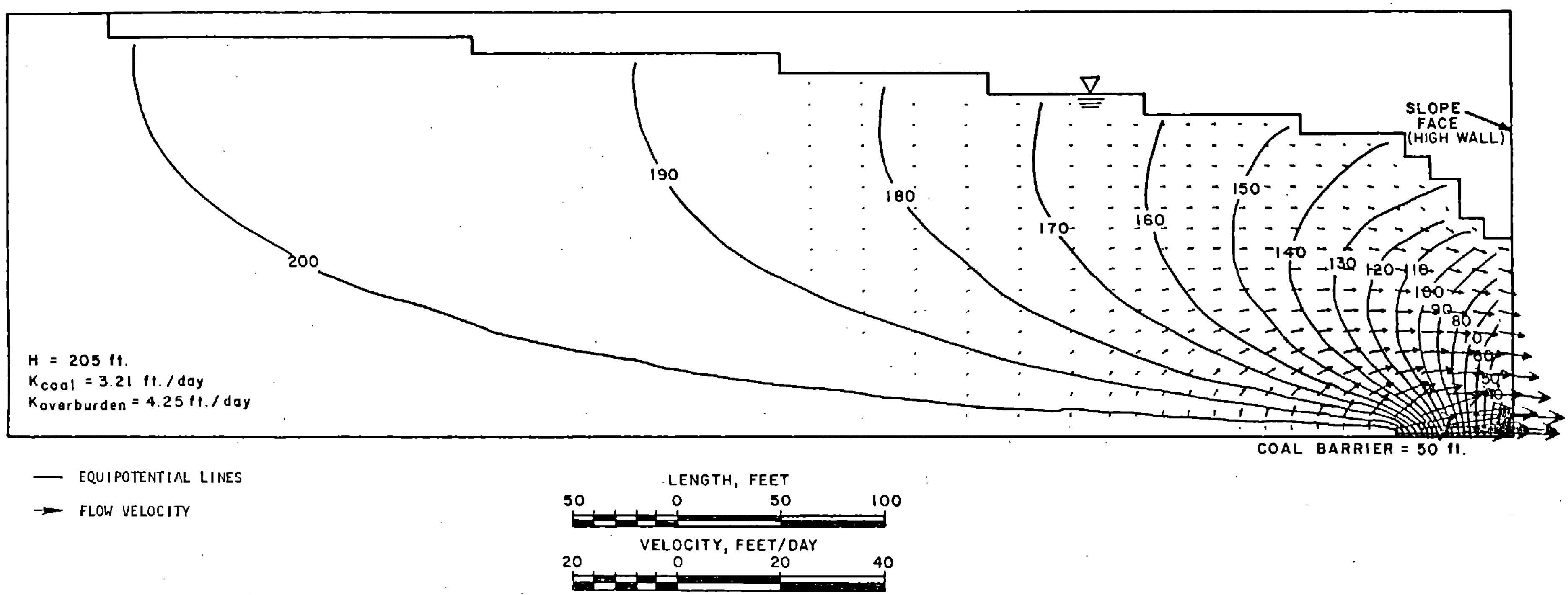


FIGURE 40 EQUIPOTENTIAL AND VELOCITY PLOTS FOR CASE NO. 10

Seepage Interpretation

A simplified two-dimensional analysis was performed to determine if it could approximate the results of the computer seepage analysis. The simplified analysis considered only the flow of impounded water through a coal outcrop barrier. The formula,

$$Q = K \times P/W \times t$$

where:

Q = flow of water through the coal barrier per foot of outcrop;

K = permeability of coal;

P = hydrostatic head existing above the coal seam in feet;

W = width of coal barrier; and

t = thickness of coal seam

was used to compute flows. The assumptions for this analysis are that both the overburden and underclay are impermeable and that all flow passes through the outcrop barrier pillar. The results of this simplified analysis are presented graphically in Figure 41. Also shown on these graphs are the data points which represent the results of the computer seepage analysis. The data points are indicated according to the ratio between the coal and overburden permeability and are labeled by their respective case numbers. It is evident that there is some deviation between the two methods.

From Figure 41 it is evident that the ratio of permeability between coal and the overburden is a factor causing a deviation between the computer analyses and the simplified model which has a permeability ratio approaching infinity since the overburden is assumed impermeable. In cases 2, 3, 6, and 7 where the permeability ratio is 6.5:1, a slight deviation from the computer analysis is seen. In cases 4, 5, 8, and 9 where the permeability ratio is 1:1, a larger deviation from the computer analysis is noted. A correction factor has been derived from the difference so that the simplified analysis can be used in the design of outcrop barriers.

The nine curves presented in Figure 41 have been summarized in Figure 42 as one curve representing a unit flow rate through

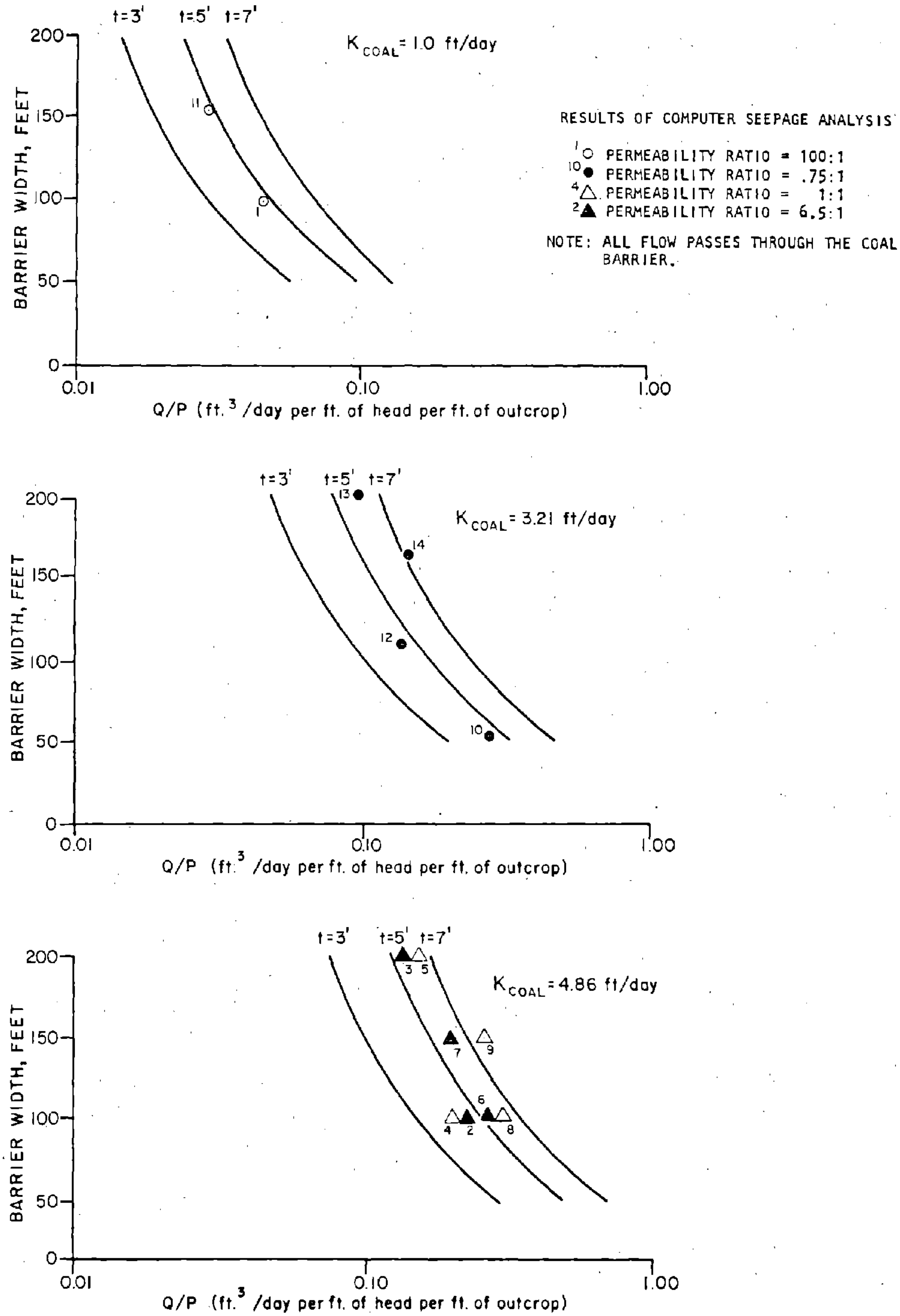


FIGURE 41

RESULTS OF SIMPLIFIED SEEPAGE ANALYSIS FOR THREE DIFFERENT COAL PERMEABILITIES (WITH COMPARISON TO COMPUTER ANALYSIS RESULTS)

the coal outcrop. The relationship of the unit flow rate to the total flow rate through the barrier is indicated by the equation,

$$Q_U = Q_T \div (PKt)$$

where:

Q_U = the unit flow rate through the outcrop; and

Q_T = the total flow rate through the outcrop.

The designer can compare the value of Q_T derived from Figure 42 with the maximum flow rate that will be tolerable for the local conditions. Q_U is a function of the barrier width, W_S, obtained from the stability anslyses presented next. W_S corresponds to a unit flow rate and was derived under the assumption that all flow passes through the coal barrier. A correction factor has been derived to account for flow through the overburden and is presented graphically as Figure 43. Notice that several coal to overburden permeability ratios are presented ranging from 100:1 to 0.75:1. Permeability ratios between these values must be interpolated. Also notice that the axes of the graph in Figure 43 are $(Q_U \times \text{head})^{-1}$. It is interesting that the coal to overburden permeability ratio of 100:1 is a line oriented 45° from the origin and indicates that beyond this ratio there is no longer any correlation for flow through the overburden. The corrected unit flow rate Q_C, obtained from Figure 43 may be used to calculate the discharge rate, Q_T as defined on Figure 42.

To summarize, the interpretation of the computer seepage analysis and the simplified analysis resulted in the curves presented in Figures 42 and 43, respectively, which make it possible to estimate the total seepage rate corresponding to the given conditions of hydrostatic head, H, and a stable outcrop barrier width, W_S. It is generally observed from the computer plots in Figures 31 through 40 that the flow path through the outcrop barrier is essentially horizontal and only small amounts of flow rise into the overburden. This is reassuring from a water quality standpoint since water polluted by the coal will not interfere with ground water quality in the overburden. Another general observation that will be utilized in stability analyses is that the point where the barrier begins in the mine is the point beyond which the slope is completely saturated. That is, if a vertical line is drawn from the origin of the outcrop barrier to the surface, the slope face and the phreatic surface coincide for the remainder of the slope.

A comparison with actual field data has been attempted and is presented in Figure 42 as data points representing field

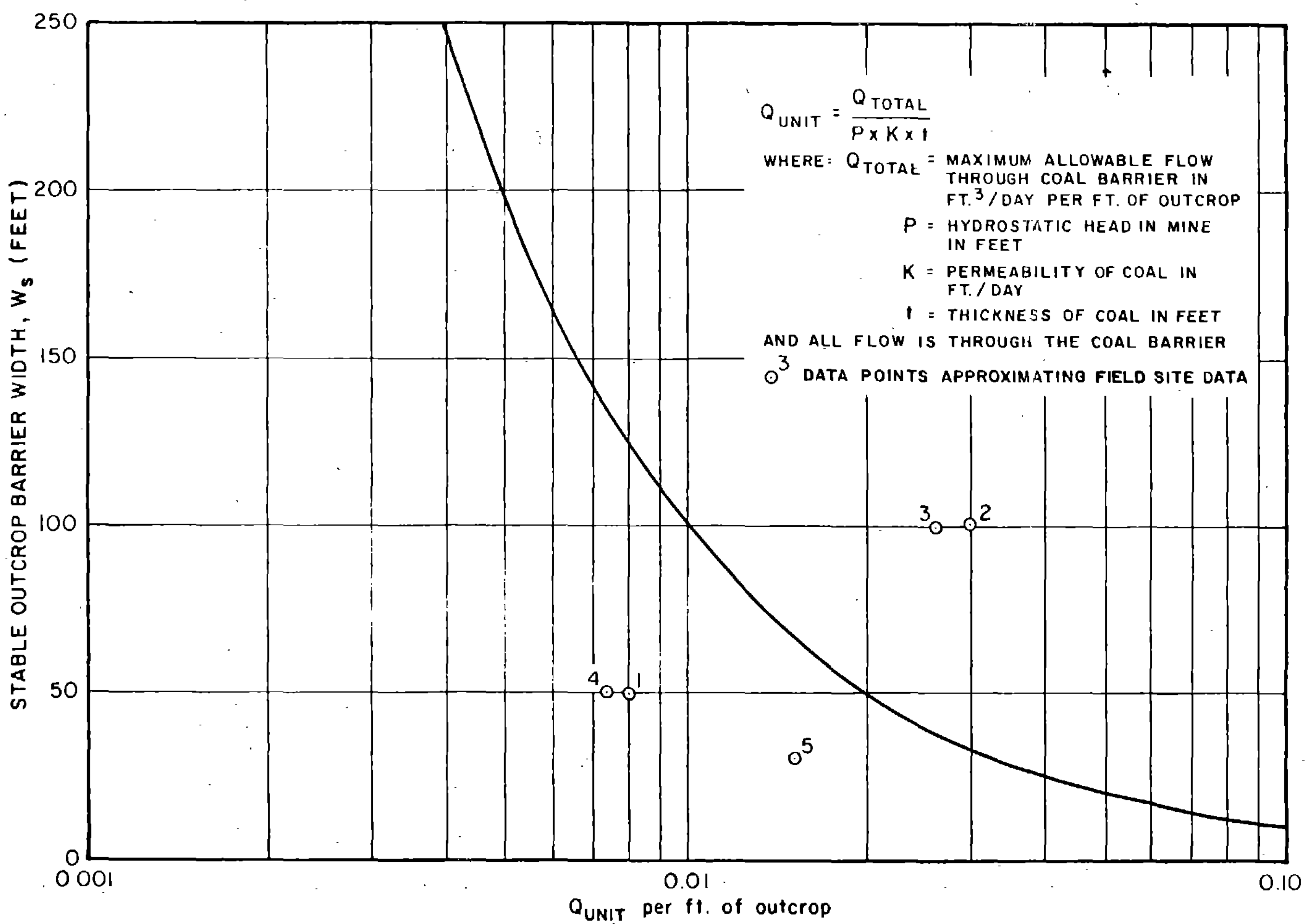


FIGURE 42

INITIAL ESTIMATE OF UNIT FLOW RATE, Qu BASED ON SIMPLIFIED ANALYSIS

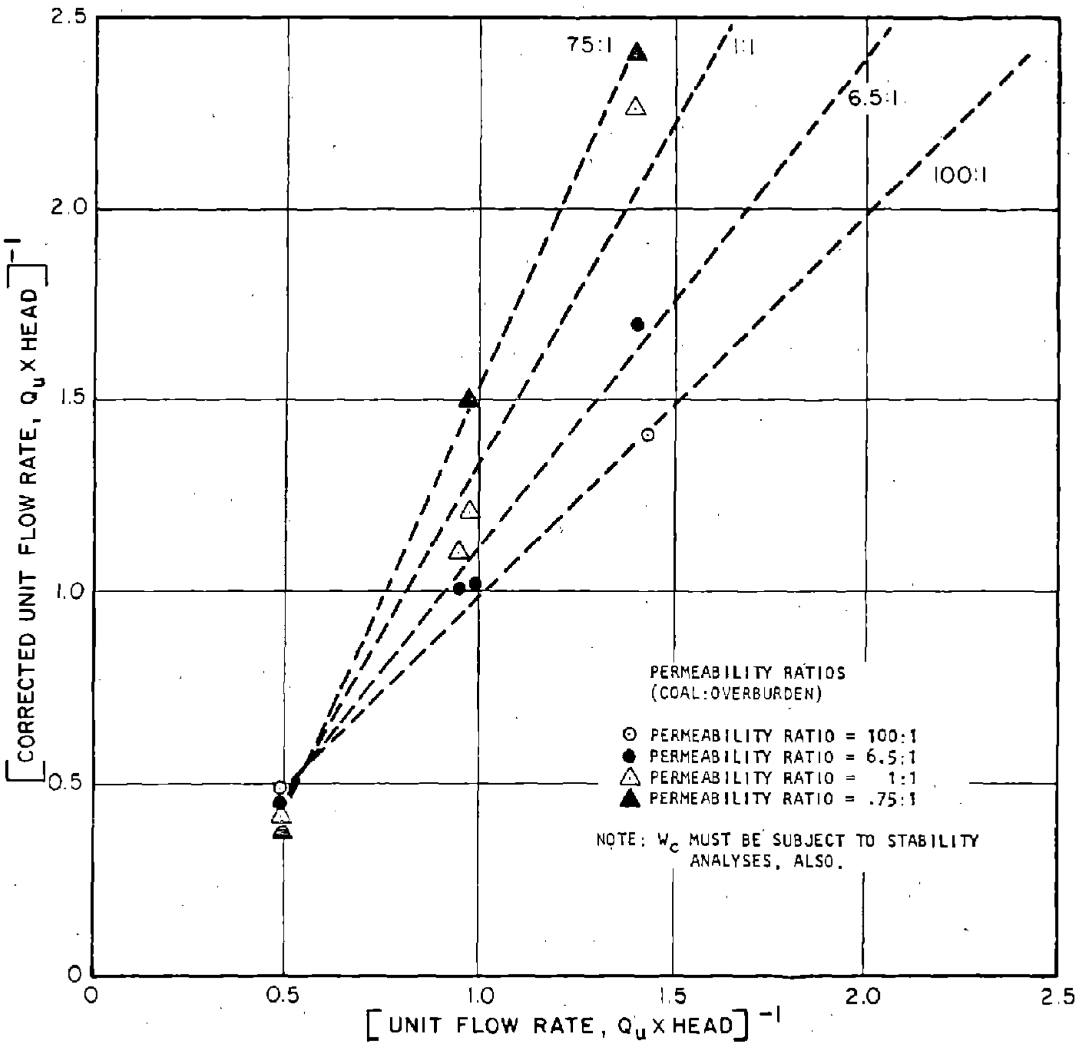


FIGURE 43

CORRECTION OF UNIT FLOW RATE TO ACCOUNT FOR OVERBURDEN PERMEABILITY

sites 1 through 5. In order to plot data points, the value of Q_U had to be calculated for each site using the formula;

$$Q_U = \frac{Q_T}{P \times K \times t}$$

Table 3 provides all of the necessary information except for K, the permeability of coal. The measured discharge flows are known to be originating at ineffective mine portal seals and can provide a limiting permeability value that the coal barrier can not exceed. Based on this assumption and the documented site conditions from Table 3, the permeability of coal is limited to less than 5 ft/day per foot of outcrop. Since this agrees with the values shown in Table 14, the value used to compute Q_U was 4.86 ft/day per foot of outcrop which is the highest effective permeability value given for rock in the SME Mining Engineering Handbook (see Table 14). The locations of these data points indicate that sites 2 and 3 have higher flows than would be anticipated while sites 1, 4, and 5 have lower flows. However, since the coal permeability was not actually measured, these discrepancies could easily be accounted to a slight difference in the permeability.

The flow rates determined as a result of the seepage analysis should be used as a quide in estimating the flow that will be expected from a stable barrier. Barriers may easily be lengthened if estimated flow rates are excessive, however, they must not be decreased in size due to the stability requirement. At present there are no established relationships between water quality and barrier width due to the numerous contributing factors. The following section describes the stability analyses and presents the resulting stable barrier widths.

Stability Analysis

The data obtained in the field investigation indicated that two possible modes of failure should be considered. Therefore, an overburden blow-out analysis was performed, followed by a wedge stability analysis. The overburden blow-outs that occurred at Sites 5 and 6 provided input into the relatively simple analysis. The design of the wedge stability analysis followed the assumptions made in the seepage analysis and used the computer output to define the hydrologic conditions. The methods and results of these two analyses are presented in the following sections.

Overburden Blow-Out Analysis

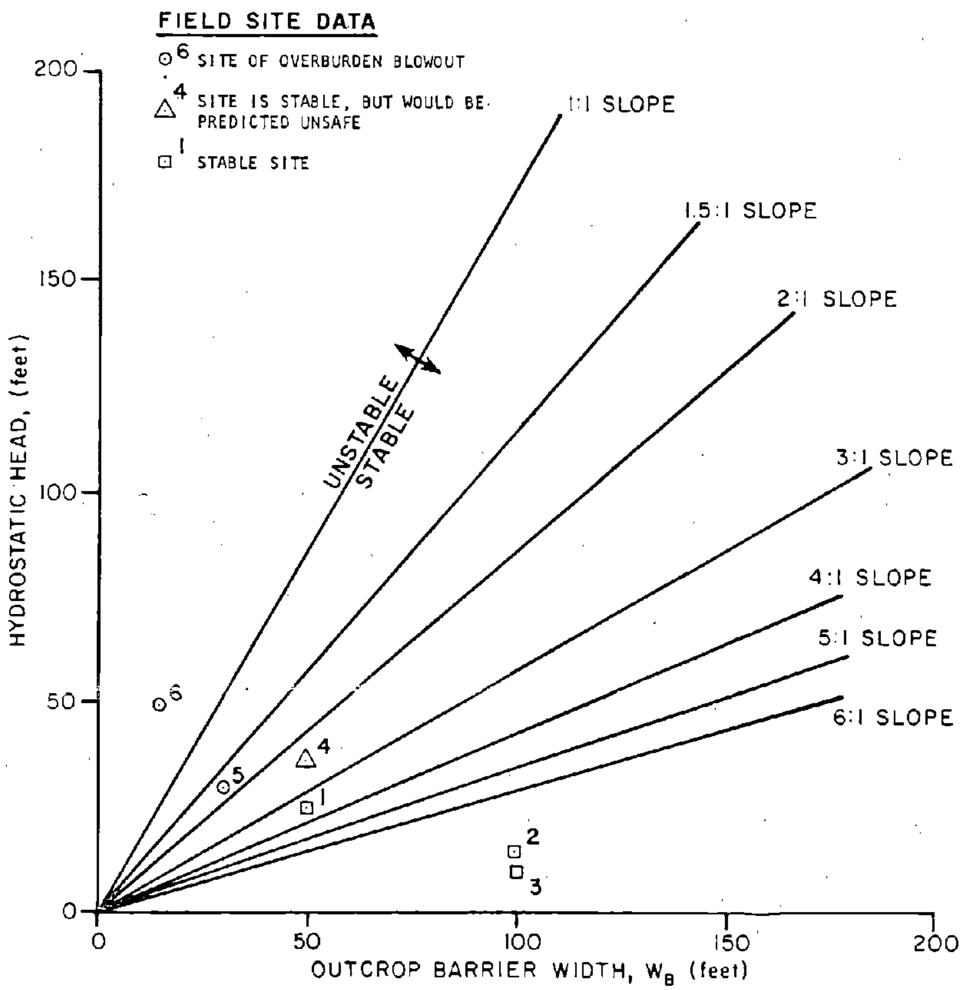
At field sites 5 and 6, blow-outs occurred through the overburden which focused attention on the limiting effect the overburden height will have on the sustainable hydrostatic head. The following analysis was performed to define the limits more clearly.

The investigation of Site No. 6 helped provide input into this analysis. By referring to Figures 26 and 28 in the case history of Site No. 6, the size of particles that blew out can be inferred. Since the particles are not massive, but rather on the order of inches, a unit weight of 130 pounds per cubic foot was used to account for joint filling and fractures that are likely to be the material that would be displaced in an overburden blow-out.

The overburden blowout analysis is simply to determine the point beyond which the bouyant forces exceed the weight of the overburden. Using a factor of safety of 1.5, a value of head which is 72% in excess of the height of overburden is calculated to be the maximum allowable. The results of this analysis are presented graphically as Figure 44. The minimum barrier requirement, W_B, varies depending upon the slope of the overburden. Figure 44 also shows six data points representing the conditions at the six field investigation sites. The two sites where blow-outs did occur would have been predicted as unstable according to this analysis. There is one case where the site has a 5:1 slope and is stable in the field, but would be predicted as unstable according to the analysis. At this site, however, an accurate mine map was not available and may actually have wider barrier pillars than assumed, which would move the data point toward a stable location. According to this comparison, the field data supports the overburden blow-out analysis. The final analysis, described next, will account for the possibility of a wedge-type failure.

Wedge Stability Analysis

A wedge stability analysis was performed to determine whether the slopes and barriers modeled could sustain the hydrostatic forces as determined by the seepage analysis. The potential failure surface was approximated by a vertical line through the overburden and a horizontal line at the coal-underclay interface. A description of the procedures and results follows. The parameters that were varied included the coal seam thickness, the hydrostatic head, the topography, and the angle of internal friction of the coal.



NOTE:

FOR EACH OVERBURDEN SLOPE, THE UNSTABLE BARRIER WIDTH IS ABOVE THE LINE AND THE STABLE BARRIER WIDTH IS BELOW THE LINE.

FIGURE 44

OUTCROP BARRIER REQUIREMENT PRESCRIBED BY OVERBURDEN BLOWOUT ANALYSIS (FOR VARIOUS OVERBURDEN SLOPES)

Method of Analysis

The hydraulic conditions for the stability analysis model were based largely on the results of the seepage analysis where it was generally found that the point where the outcrop barrier originates in the mine is the point beyond which the slope is completely saturated. This observation led to the derivation of the free body diagram shown in Figure 45. The wedge consists of a thin coal seam of thickness, T, overlain by a sloping overburden of thickness, H. The analysis assumed that the driving force was U_1, and that the resisting force was $(W-U_2)\tan \Phi$. Initially, twenty cases were analyzed for a variety of different conditions. Table 17 presents the cases and the given conditions for each.

Assumptions

In order to keep the model basically the same as the seepage analysis, only two material types were assumed, that is, coal and overburden. A wide range of physical and engineering properties exist in the Eastern Bituminous Region. The values chosen are representative of what may actually be found in some locations, and adhere to the assumptions made in the seepage analysis. Specific gravities of 1.3 and 2.6 were assumed in the seepage analysis for coal and overburden respectively which provide values of 80 and 162.5 pounds per cubic foot, for their densities.

The slope of the face input in the seepage analysis was 45°; however, three slopes were analyzed for the wedge stability analysis. The slope (S) was defined as the horizontal change for one unit of vertical change (see Figure 45). Three values of S were analyzed representing a slope from 45° to 26°.

The failure surface was assumed to be a vertical line through the overburden and coal, and a horizontal line at the coal-underclay interface. Based on the case history of Site No. 6 where a vertical failure did occur, the assumption seems reasonable, however, the major fracture orientation will ultimately determine the failure surface and has only been assumed as vertical for the purpose of this analysis. Several analyses were performed to determine the influence of the failure plane's orientation and it was decided that a sloping failure surface was too penalizing. Overall, the stability analysis is sufficiently conservative especially with assumptions on cohesion and pore pressure distribution taken into account. The steady state pore pressure distribution along the failure surface was assumed to be linear which is conservative when compared to the distributions resulting from the seepage analysis.

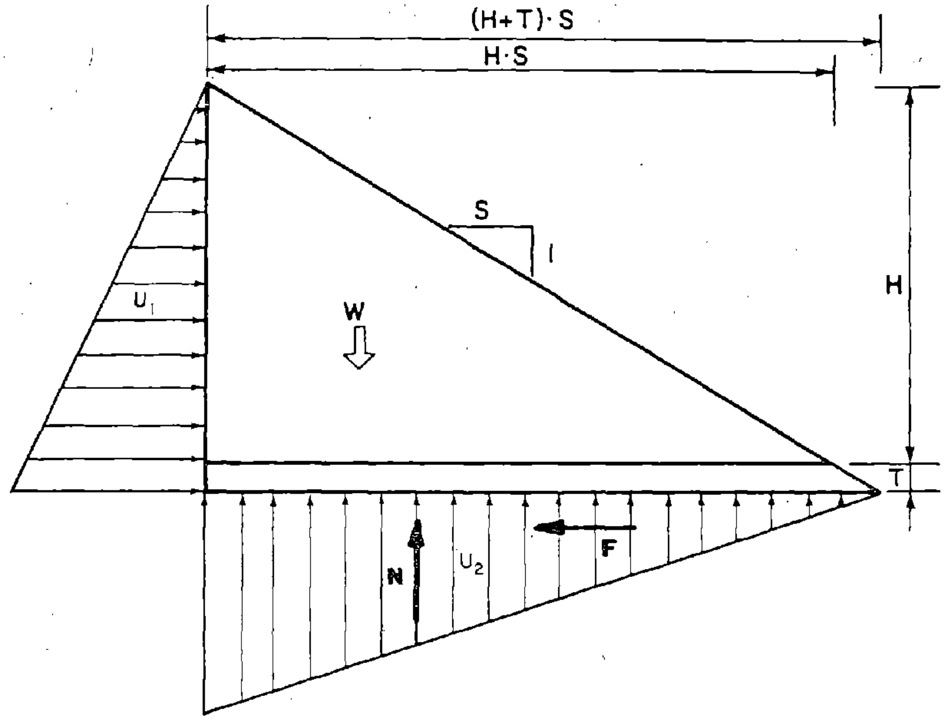


FIGURE 45 FREE BODY DIAGRAM
FOR WEDGE STABILITY ANALYSIS (CASES 1 THROUGH 20)

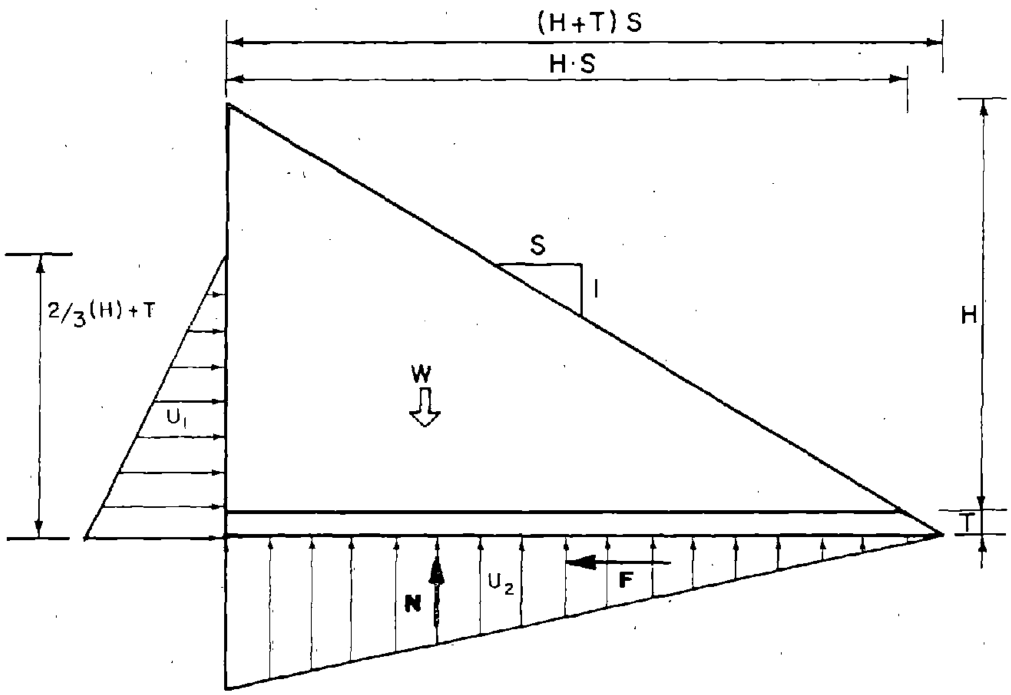


FIGURE 46 FREE BODY DIAGRAM FOR
WEDGE STABILITY ANALYSIS (CASES 21 THROUGH 40)

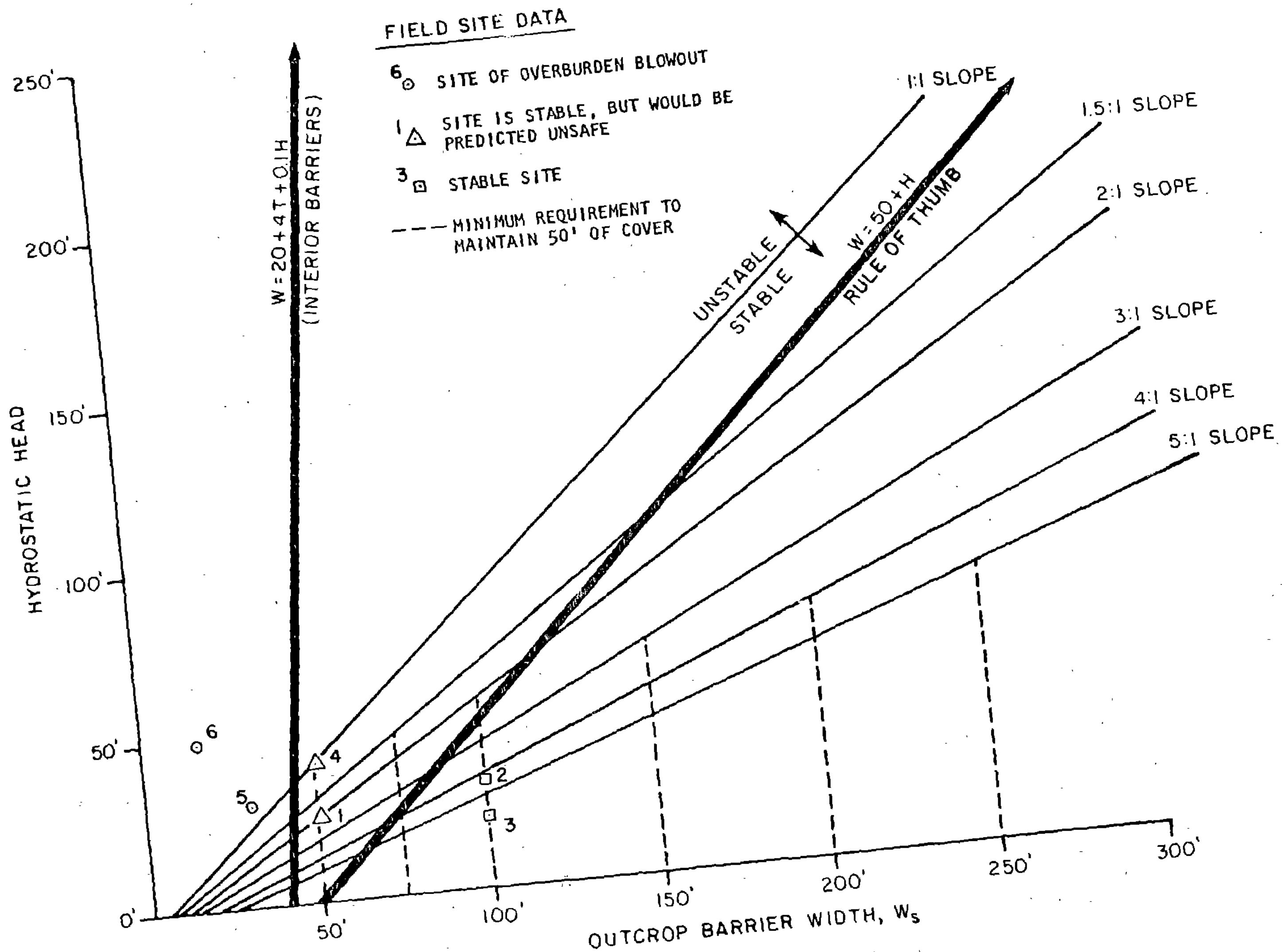
Values for cohesion and the angle of internal friction, Φ, were chosen based on documented cases. The cohesion of the coal, underclay, and overburden was set equal to zero which assumes the existance of open fractures. The values for the angle of internal friction chosen for coal, underclay, and overburden are peak values rather than residual values. This assumes that the rock mass is still intact and has not experienced any major deformation. Since a failure of this type did not occur at any of the field investigation sites, a comparison with actual data cannot be made. However, for a coal outcrop not disturbed by mine portals, it was assumed that weathering would not cause enough degradation to prescribe the use of residual strength values. The Φ value for coal was assumed to be 35^o, which is slightly less than values cited in the literature.(15,28) Values of 30^o and 40^o for Φ_{coal} were used in a sensitivity analysis in cases 15 through 20; however, $\Phi_{overburden}$ was chosen as 50^o based on sample analyses of roof rock.(9) $\Phi_{overburden}$ was only required when non-vertical failure surfaces were analyzed. The value of $\Phi_{underclay}$ was assigned based on a documented residual strength value of 29^o.(8) It was assumed that the peak value would closely approximate the 35^o value that was used for Φ_{coal}.

Results of Analysis

For cases 1 through 20, the factors of safety were computed and are presented in Table 17. Upon completion of the first twenty cases, it was decided to alter the height of the phreatic surface to a level of 2/3(H)+T to determine what effect the restoration of the ground water table would have on stability. Figure 46 illustrates the free body diagram for cases 21 through 40. This analysis may be more realistic in terms of actual post-mining restoration of the ground water level. The results of the stability analysis for cases 21 through 40 are also presented in Table 17. The obvious result is that stability is greatly enhanced.

Stability Interpretation

The results of cases 1 through 14, shown in Table 17, have safety factors varying from 0.84 to 2.15. In order to derive barrier widths that consistently represent an assigned safety factor of 1.5, the cases were reanalyzed under a constant hydrostatic head. Under this condition, the safety factor varies directly with the outcrop barrier width. Rather than solving for the factor of safety as in the first analysis, a value of 1.5 was assigned as the factor of safety, enabling the outcrop barrier width to be established. Figure 47 summarizes the results of the second analysis and presents the recommended outcrop barrier widths for various overburden



NOTE:

FOR EACH OVERBURDEN SLOPE, THE UNSTABLE BARRIER WIDTH IS ABOVE THE LINE AND THE STABLE BARRIER WIDTH IS BELOW THE LINE.

FIGURE 47

BARRIER WIDTH REQUIRED TO MAINTAIN A SAFETY FACTOR OF 1.5

slopes. Two additional lines are also presented on Figure 47 and represent the formulas presently used to some extent by regulatory agencies in the Appalachian Region. The rule of thumb, $W = 50 + H$, is in general agreement with the results of this analysis for steeper slopes. The other formula, $W = 20 + 4t + 0.1H$, was derived as an interior barrier guideline and indicates there is no application to outcrop barrier design.

The data points representing the six field investigation sites are also shown on Figure 47. In this analysis, there are two sites that would be predicted as unstable; yet in the field, they have shown no signs of failure. The two failures indicated are overburden blow-outs and not wedge failures. A wedge failure was not observed at any of the field sites.

TABLE 17. - Given Conditions and Results of Stability Analyses for Cases 1 through 40

CASE NO.	GIVEN CONDITIONS				RESULTING FACTOR OF SAFETY	CASE NO.	GIVEN CONDITIONS				RESULTING FACTOR OF SAFETY
	T[a]	H[b]	S[c]	Φ[d]			T[a]	H[b]	S[c]	Φ[d]	
1	2.5 ft	100 ft	2.0	35°	2.15	21	2.5 ft	67 ft	2.0	35°	5.89
2	5.0	100	2.0	35	2.07	22	5.0	67	2.0	35	5.70
3	10.0	100	2.0	35	1.92	23	10.0	67	2.0	35	5.37
4	5.0	25	2.0	35	1.68	24	5.0	17	2.0	35	4.82
5	5.0	50	2.0	35	1.92	25	5.0	35	2.0	35	5.37
6	5.0	200	2.0	35	2.15	26	5.0	134	2.0	35	5.89
7	5.0	25	1.5	35	1.26	27	5.0	17	1.5	35	3.61
8	5.0	50	1.5	35	1.44	28	5.0	35	1.5	35	4.03
9	5.0	100	1.5	35	1.55	29	5.0	67	1.5	35	4.28
10	5.0	200	1.5	35	1.61	30	5.0	134	1.5	35	4.42
11	5.0	25	1.0	35	0.84	31	5.0	17	1.0	35	2.41
12	5.0	50	1.0	35	0.96	32	5.0	35	1.0	35	2.68
13	5.0	100	1.0	35	1.03	33	5.0	67	1.0	35	2.85
14	5.0	200	1.0	35	1.08	34	5.0	134	1.0	35	2.94
15	2.5	100	2.0	30	1.77	35	2.5	67	2.0	30	4.86
16	2.5	100	2.0	40	2.58	36	2.5	67	2.0	40	7.06
17	5.0	100	2.0	30	1.70	37	5.0	67	2.0	30	4.70
18	5.0	100	2.0	40	2.48	38	5.0	67	2.0	40	6.83
19	10.0	100	2.0	30	1.58	39	10.0	67	2.0	30	4.43
20	10.0	100	2.0	40	2.30	40	10.0	67	2.0	40	6.44

[a]Thickness of coal seam.

[b]Maximum thickness of overburden.

[c]Horizontal change for 1 unit of vertical change along seepage face.

[d]Peak value for internal angle of friction for coal.

RECOMMENDED DESIGN PROCEDURES

Based upon the background information and data analysis presented, recommendations intended to guide mine planners and designers in the initial development of mine barriers are presented. The recommendations begin with a pre-mining investigation program to determine a few basic site parameters, followed by a method for determining the recommended barrier width and, finally, some supplementary alternatives for increased strength and decreased seepage. The recommendations that follow are based on the assumption that post-mining inundation will occur and that the coal outcrop is not disturbed by a mine entry or portal. Cases involving disturbed outcrops will be discussed separately.

Determination of Site Conditions

In order to design an outcrop barrier, the site conditions will have to be evaluated. The determination of the following site parameters is recommended in order to utilize the results of the seepage, overburden blow-out, and wedge stability analyses presented herein.

In order to utilize the seepage analysis, five parameters must be established:

1. The level of hydrostatic head that will become established after closure of the mine;
2. The permeability of the coal barrier;
3. The permeability of the overburden;
4. The thickness of the coal seam; and
5. The stable outcrop barrier width.

Knowing these five parameters will provide the input to estimate the discharge from a stable outcrop barrier. A conservative estimate of the level of head that will become established after mine closure will be provided by the hydrologic network that exists prior to mining. A potentiometric surface map should be developed by mapping the locations and elevations of all springs and seeps, water levels in test borings and wells above the area to be mined, and any oil and gas wells that may be on the property. If boreholes are available for pumping tests, in-situ measurements for permeability can be performed. The coal barrier's permeability will depend heavily on the cleat orientation. The overburden permeability is important for determining flow rates that will induce inundation.

For the overburden blow-out analysis, particular attention should be devoted to establishing the properties of fractures and joints. In the analysis presented earlier, a density of 130 pounds per cubic foot was assumed for the fracture filling. Any large discrepancy from this assumption will require a closer examination of the overburden blow-out analysis to modify the assumption. The local jointing patterns should be measured and plotted to make their location and direction clear.

For the wedge stability analysis, three additional parameters should be established: the location of geologic discontinuities in the overburden and coal; the slope of the overburden at the surface; and the Φ value at the coal-underclay interface. Geologic discontinuities, along with fractures and joints, will determine the likely failure surface for a wedge failure. Any appreciable deviation from the vertical plane, which was assumed in the analysis presented earlier, should be considered more closely. The overburden slope can easily be determined from topography maps, and as shown on Figures 44 and 47, barrier widths are prescribed for a variety of slopes ranging from 1:1 to 5:1. The Φ value at the coal-underclay interface is a very important aspect of the wedge stability analysis. A value of 35° was used to recommend barrier widths; however, any appreciable deviation from 35° may warrant a modification to the recommendations.

In addition to the parameters required for specific analyses, there are other basic properties that will provide useful information for the overall design. It is recommended that a basic knowledge of the coal and overburden mineralogy be obtained to provide a reasonable indication of effluent quality. Theoretically, if the mine pool becomes part of the post-mining ground water flow system, it would be discharged at a rate the surface water could assimilate. If this is the case, mine water will seep out through normal ground water discharge areas. Once the mine water becomes part of the surface water flow system, the effluent becomes diluted and also has less chance for chemical reactions. According to a recent study,(17) the travel time for water in vertical ground water flow in the fractured strata over an abandoned mine is five to six orders of magnitude longer than travel time in a surface water stream over a comparable distance. Consequently, the time available for chemical reactions between mine drainage as ground water and minerals in the fractured strata compared to time for reaction between mine drainage as stream flow and minerals in the streambed is correspondingly five to six orders of magnitude greater.

Estimating Outcrop Barrier Widths

The seepage, overburden blow-out, and wedge stability analyses resulted in the derivation of curves to predict seepage rates and stable slopes for a given hydrostatic head (refer to Figures 43, 44, and 47, respectively). If the assumptions are in general agreement with the site conditions, the curves presented on Figures 44 and 47 are straightforward and require only hydrostatic head and overburden slope as known variables. The recommended barrier widths, W_B and W_S, can easily be obtained from these figures. The seepage analysis, however, requires more judgment by the designer. As shown by the graph on Figure 42, the quantity Q_u never reaches zero; hence, seepage can be minimized but is not likely to be eliminated. It is up to the designer to determine a satisfactory amount of discharge, Q_T, depending upon the flow rate required to induce inundation and upon consideration of the probable effluent quality. With only six field investigation sites covered in this study, it is impossible to draw conclusions regarding effluent quality and its relationship to site conditions. However, a study discussed earlier in this report did such an analysis with 86 sites in the Appalachian Region.[5] It was shown that there is a direct correlation between mine effluent quality and the composition of the coal and overburden.[5] The obvious relationships are that high sulfur contents lead to increased acidity and that calcareous material acts as a neutralizing agent.[5] Having a basic knowledge of the coal and overburden mineralogy allows a reasonable estimate of the effluent quality to be deduced. There have not been any studies relating water quality to barrier widths. The recommended procedure, therefore, is to provide a post-mining closure plan that will induce the restoration of ground water to its normal configuration. In order to induce ground water restoration, the seepage rate must be less than the recharge rate, which is likely to be very small due to the presence of aquitards in the overburden of deep mines in the Eastern Bituminous Region. Once the ground water level is restored, seepage through normal discharge areas should be allowed if it is at a rate that the surface waters can assimilate.

Supplementary Support Alternatives

Several alternatives are available for supplementary support of the outcrop barrier. The alternatives discussed will either decrease seepage or increase the stability of the slope. It is not within the scope of this project to make economic evaluations; therefore, alternatives will be discussed in terms of advantages and disadvantages to post-mining inundation attempts.

Curtain Grouting

Curtain grouting has been used as a remedial measure for abandoned mine reclamation in Pennsylvania and other Eastern Coal Region states. Based upon these installations, some conclusions can be drawn. When a single line of grout is installed, it has been ineffective in preventing the passage of water. A double line of grout curtain should be installed to reduce seepage. The proper installation of grout curtains is very time consuming but essential to their effectiveness. At present there are only a few trained personnel who are experienced in the installation of grout curtains. If a grout curtain is installed and is effective in preventing seepage, an increased overburden thickness will be required to safely maintain the additional head.

All of the disadvantages listed above must be weighed against the advantage of extracting more coal by leaving a smaller outcrop barrier.

Compartmentalized Barrier Systems

A new concept in reducing the hydrostatic pressure on the outcrop barrier is to design an interior barrier that can be sealed after that portion of the mine is completed. By leaving a barrier in the coal mine, the mine is basically divided into two compartments, each of which should be sealed individually and would be exposed to a reduced head.

Since this concept is new, it is difficult to draw conclusions; however, some general comments can be made. The degree to which heads are reduced across internal mine barriers will depend on the amount of fracturing that develops in the overburden as a result of mining. If vertical ground water flow is enhanced by fracturing, there may be no noticeable reduction in head across the barrier; however, there may be a difference in the circulation pattern of the ground water once equilibrium is established.

Relief Wells

A procedure that is highly recommended to prevent an excess accumulation of mine water or a build-up of hydrostatic head beyond the designed level is to drill relief wells from the surface into the mine void so that there is a direct passage for mine water to the surface. The surface elevation of the relief wells should be at or lower than the level of hydrostatic head for which the barrier was designed. According to the mine pool water quality obtained from the six field sites, there is less reason to be concerned about mine pool water being discharged at the surface than there is for mine water seeping through the outcrop. Wells such as this will also allow for periodic sampling of the mine pool and observation of the level of hydrostatic head on the outcrop barrier. In mines sealed by the Pennsylvania DER, these wells have been utilized successfully and provide information in addition to relief from excessive head. The DER has not had a problem with maintenance of the wells; however, most holes are not in excess of 50 feet in depth.

Dewatering Wells in Overburden

Another concept to relieve some of the hydrostatic head on the outcrop barrier is to drill angled boreholes into the hillside at a point in the overburden that would allow the gravity drainage of ground water to the surface. Depending upon the elevation at which the boreholes were drilled, any amount of head could be drained off the slope and thus relieved from the outcrop barrier. The disdavantages of this concept are that the drain holes would require perpetual maintenance to prevent clogging, and that the effect on water quality using this procedure is not known. This technique may also be impractical if it disrupts the water supply to local communities and residents.

Backfilling Over Outcrop

An option available to increase the stability of the slope is to backfill over the outcrop and a portion of the overburden. If material is available for deposition such as coal refuse or mine waste, it may be a viable alternative. One area of concern will be to assure that the material is deposited properly to avoid additional slope stability problems. Due to the presence of a seepage face along the original slope, it may be difficult to maintain the stability of the backfill material. In addition to slope stability, changes in the configuration of the ground water flow will also occur depending upon the nature of the backfill material.

Consideration of Special Cases

Several cases have been mentioned throughout this report that do not agree with the basic assumption that the outcrop is undisturbed. Therefore, cases when the outcrop has been removed by strip mining and when portals have been driven through the outcrop will be discussed with regard to mine closure.

Outcrop Is Removed by Strip Mining

When strip mining has removed the outcrop coal, the design of a coal barrier becomes somewhat of a cross between an interior barrier and an outcrop barrier. The legal requirements in this case are not clear. Since the barrier does not actually contain the coal outcrop, an internal barrier would be prescribed. However, due to the effects of blasting from the strip mine, regulators prefer to prescribe outcrop barrier requirements.

In order to perform the analysis, the site conditions must be determined. The permeability of the coal will be less influenced by weathering, yet more fractured due to blasting associated with the strip mine. It is very difficult to assign a permeability value to such a variable quantity. Nevertheless, a case was examined for seepage and stability using a coal permeability value of 3.21 feet per day. The computer plot resulting from the seepage analysis is shown on Figure 40. The model has a 200-foot highwall on the right. As shown on the figure, a 50-foot barrier of coal results in a seepage face that intersects the highwall at a height of approximately 100 feet. According to the data presented in Table 16, the total outflow from the seepage face is 3.05 gallons per minute per foot of outcrop.

A stability analysis was performed using material with a density of 125.0 pounds per cubic feet as the backfill for the strip mined area. It was assumed that the backfill material was lying on underclay with an internal angle of friction of 15^{o}.[8] This is a residual value for underclay, since there has been severe disturbance due to stripping operations. Figure 48 illustrates the free body diagram for the stability analysis. The factor of safety for this case is 2.74.

This brief analysis indicates that with a 50-foot barrier the safety factor is satisfactory; however, the seepage rate is higher than any of the other cases studied so far. This is largely due to the narrow width of the barrier. Again, the effluent quality and the local ground water recharge rates will determine how important it is to restrict the flow. In this situation, it may be possible to induce inundation by restricting flow in the backfill material rather than to leave a larger coal barrier.

Outcrop Is Disturbed by Portals

When an outcrop is disturbed by mine portals, the success of the mine closure will depend largely on the ability of the portals to be effectively sealed. To perform an analysis, it

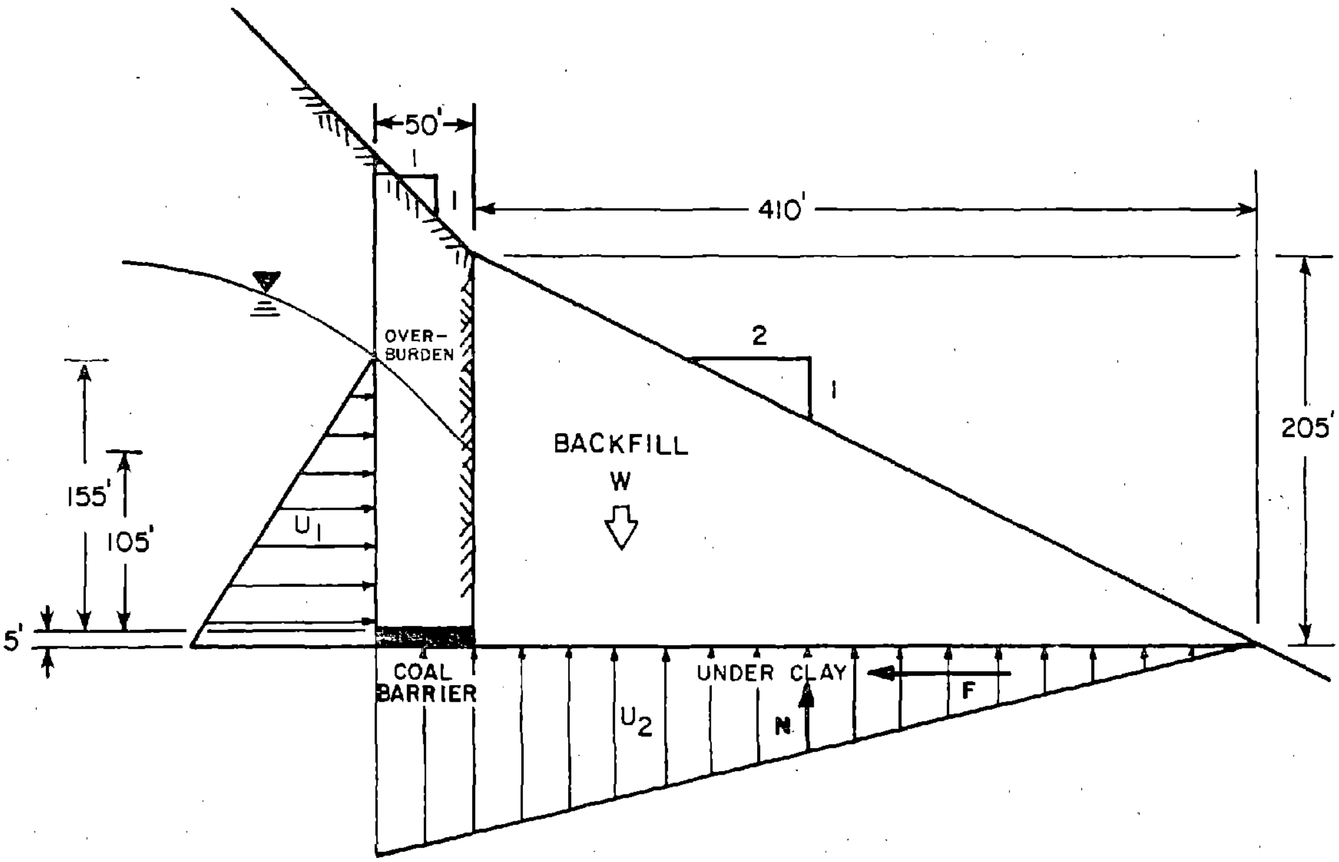


FIGURE 48 FREE BODY DIAGRAM FOR CASE WHEN OUTCROP IS REMOVED BY STRIP MINING

must be assumed that portal seals would be installed properly and that supplemental curtain grouting would be effective in reinforcing the weakened rock along the portal. Under these assumptions, the outcrop barrier should be designed following the guidelines established for an undisturbed barrier.

CONCLUSIONS

It has been demonstrated that with a minimal amount of pre-mine planning and investigation, an outcrop barrier may be designed that will provide stability of the slope and minimize seepage. In addition, several alternatives have been presented that provide supplemental support to the outcrop barrier by either reducing the seepage rate or increasing the slope's stability. In contrast to the "rule of thumb," which requires a minimum of 50 feet of outcrop barrier plus an additional foot for every foot of hydrostatic head, the recommended barrier widths must satisfy three separate criteria:

1. That the slope is stable;
2. That a sufficient amount of overburden is present to prevent a blow-out; and
3. That seepage flow is acceptable.

In general, the existing outcrop barrier regulations do not adequately address post-mining inundation or methods to assure its stability. The "rule of thumb" is too conservative in some cases, yet becomes inadequate as the overburden slope decreases (see Figure 47). Throughout the Appalachian Region, outcrop barrier guidelines are very inconsistent, as is the philosophy of regulatory agencies.

Most of the applicable technology that has been utilized in the design of outcrop barriers was derived from abandoned mine reclamation programs. Mine closure techniques have been demonstrated at many sites throughout the Appalachian Region, and water-tight seals have been an integral part of the technology development. Unfortunately, most cases where documentation is possible have outcrops that are disturbed by entries or portals, and the greatest problem has been to seal the portals. Existing sources provided limited input to the physical properties of coal measure strata, especially permeability values. This is one area where further work will be required in order to establish a seepage rate that will induce inundation. The permeability values for overburden are likely to be very small due to the presence of aquitards and, hence, the steady-state conditions modeled in the seepage analysis may take an extended period of time to achieve.

The stability analyses results are generally supported by the field investigation data. The sites where overburden blow-outs occurred are clearly indicated as being below the minimum barrier requirement (see Figure 44). There are two cases that are stable in the field, yet do not meet the minimum barrier requirements according to the wedge stability analysis (see Figure 48). Hence, there is a certain degree of conservatism in the model. This conservatism is justified, since the outcrop barrier must remain stable for as long as water is impounded. In addition to the design recommendations presented on Figures 43, 44, and 47, it is strongly recommended that a minimum of 50 feet of outcrop barrier be maintained at all times and that a relief system be constructed to prevent the development of hydrostatic head in excess of the design assumption.

From a water quality standpoint, successful inundation attempts have shown generally good water quality in the mine pool but degraded water quality as it seeps out of the mine. At present, there has been no explanation of this phenomenon; however, in the recommended relief well system there is a direct passage for mine pool water to the surface without passing through any outcrop coal, with the expectation that this will minimize degradation of water quality.

The guidelines that have been set forth are intended to provide a safe outcrop barrier pillar based on assumptions regarding material properties that are typical of the Appalachian Region. Differences from the assumed properties should be considered with respect to their effect on seepage or strength. Through the use of supplementary support alternatives, mine operators have the option to control the level of hydrostatic head, which thereby allows leaving the minimum amount of coal in the ground.

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APPENDIX A

PENNSYLVANIA BITUMINOUS COAL MINING LAWS
ACT NO. 729

MINING IN SAFETY ZONES

Act No. 729, approved December 22, 1959 (P.L. 1994)

AN ACT

Prohiting mining in certain areas without prior approval by the Department of Mines and Mineral Industires; establishing standards for the approval of plans for mining in such areas; imposing powers and duties on the mine foremen and the Department of Mines and Mineral Industries; and providing penalties.

The General Assembly of the Commonwealth of Pennsylvania hereby enacts as follows:

Section 1. Establishment of Safety Zones. -- A safety zone is hereby established beneath and adjacent to every stream, river and natural or artificial body of water in the Commonwealth that is sufficiently large to constitute a hazard to mining in the opinion and in the discretion of the department. Such safety zone shall, in the case of such streams and rivers, extend horizontally two hundred feet from the high water mark of each bank. In the case of any other body of water sufficiently large to constitute a hazard to mining in the opinion and in the discretion of the department, it shall extend horizontally two hundred feet from the known perimeter. In any case, the zone shall extend downward to the limit of the workable beds.

Section 2. Written Authorization Needed to Mine Within Safety Zone. -- (a) No mining or removal of minerals whatsoever shall be permitted within the safety zone unless authorization is specifically granted in advance and in writing by the Department of Mines and Mineral Industries.

(b) Such authorization shall only be granted upon application of the operator and/or the lessor. Such application shall be accompanied by four copies of a plan of the proposed mining operation. The plan shall indicate the thickness of the unconsolidated strata, the thickness of the rock strata and coal beds overlying the bed to be mined, the thickness of the bed, the widths of the openings to be made and the width of the pillars to be left, and any other special features that may be deemed necessary as affecting the contemplated first mining.

(c) The Department of Mines and Mineral Industries shall make periodic examinations to determine the accuracy of all plans, maps and drawings submitted to them under the provisions of this act.

Section 3. Requirements for Plan Approval. -- In no instance will any plan be approved if there is less than thirty-five feet of rock cover. Factors considered in plan approval shall include thickness of bed, width of mine openings, width of pillars and such other facts as are deemed applicable by the Department of Mines and Mineral Industries.

Section 4. Pillar Recovery. -- No pillar recovery shall be undertaken until such time as the plans are approved by the Department of Mines and Mineral Industries. Application for pillar recovery shall be accompanied by four copies of the plan which shall include such information as shall be determined by the department. The approval or disapproval shall be based on the factors of depth, the thickness of the bed, the percentage of pillars proposed to be extracted and to be left, the effect on pillars remaining in overlying beds, and any other special features that may be deemed necessary by the department.

Section 5. Proof of Rock Cover. -- (a) Proof of the existence of thirty-five feet of rock cover shall accompany any plans submitted.

(b) Said proof of rock cover is to be ascertained by test holes drilled on intersecting lines forming rectangles or squares where the cover thickness is less than fifty feet. These holes shall be drilled on spacing of not more than thirty-five feet centers.

Section 6. Copies of Plans and Proof of Rock Cover to be Signed. -- All copies of the aforementioned plans and proof of rock cover must indicate the location of the test holes and the depth of the rock cover, and they must be signed before submission to the Department of Mines and Mineral Industries by a Registered Mining Engineer representing the operator and a Registered Mining Engineer of the lessor and/or owner.

Section 7. Approval of Disapproval of Plans. -- (a) After examination and approval of the plans by a Registered Mining Engineer for the Department of Mines and Mineral Industries and the secretary of the department, they shall sign all copies. The original shall be retained in the department, one copy shall be forwarded to the State mine inspector for the area in which the mining is to be carried on, one copy is to be forwarded to the Registered Mining Engineer representing the operator and/or the lessor or owner, and one copy is to forwarded to the operator.

(b) If the plan is disapproved, the Registered Mining Engineer for the Department of Mines and Mineral Industries and the secretary of the department shall note their reasons and attach a copy of each set of plans. One copy of the plan shall then be returned to the operator, one to the State mine inspector for the area, one to the Registered Mining Engineer for the operator and/or owner or lessor and one shall be retained by the department.

Section 8. Notice to Miners Working Within the Safety Zone. -- After approval of the plan by the Department of Mines and Mineral Industires, no mining or removal of minerals may begin within the safety zone until the mine foreman has conspicuously posted a notice on the outside of the mine and has orally notified each miner affected that he is working in the safety zone.

Section 9. Penalties. -- Any agent of the mine operator or any of its officiers or supervisory employes, or any agent of the owner or any of the owner's officers or supervisory employes, if said owner engages in active supervision and control over the operator, or any mine inspector who by acts of commission or omission, wilfully and knowingly violates any provisions of this act, and the act of commission or omission is the contributory cause of an incident which results in death or serious bodily harm or anyone lawfully in the mine, shall be guilty of a felony, and, upon conviction, be sentenced to pay a fine of not more than five thousand dollars ($5,000) and undergo imprisonment for a period not exceeding three years, or both.

APPENDIX B

THE CLEAN STREAMS LAW OF PENNSYLVANIA
SECTION 315

SECTION 315. OPERATION OF MINES

(a) No person or municipality shall operate a mine or allow a discharge from a mine into the waters of the Commonwealth unless such operation or discharge is authorized by the rules and regulations of the board or such person or municipality has first obtained a permit from the department. Operation of the mine shall include preparatory work in connection with the opening or reopening of a mine, backfilling, sealing, and other closing procedures, and any other work done on land or water in connection with the mine. A discharge from a mine shall include a discharge which occurs after mining operations have ceased, provided that the mining operations were conducted subsequent to January 1, 1966, under circumstances requiring a permit from the Sanitary Water Board under the provisions of section 315(b) of this act as it exited under the amendatory act of August 23, 1965 (P.L. 372). The operation of any mine or the allowing of any discharge without a permit or contrary to the terms or conditions of a permit or contrary to the rules and regulations of the board, is hereby declared to be a nuisance. Whenever a permit is requested to be issued pursuant to this subsection, and such permit is requested for permission to operate any mining operations, the city, borough, incorporated town or township in which the operation is to be conducted shall be notified by registered mail of the request, at least ten days before the issuance of the permit or before a hearing on the issuance, whichever is first.

(b) The department may require an applicant for a permit to operate a mine, or a permittee holding a permit to operate a mine under the provisions of this section, to post a bond or bonds in favor of the Commonwealth of Pennsylvania and with good and sufficient sureties acceptable to the department to insure that there will be compliance with the law, the rules and regulations of the board, and the provisions and conditions of such permit including conditions pertaining to restoration measures or other provisions insuring that there will be no polluting discharge after mining operations have ceased. The department shall establish the amount of the bond required for each operation and may, from time to time, increase or decrease such amount. Liability under each bond shall continue until such time as the department determines that there is no further significant risk of a pollutional discharge. The failure to post a bond required by the department shall be sufficient cause for withholding the issurance of a permit or for the revocation of an existing permit.

APPENDIX C

PENNSYLVANIA BUREAU OF WATER QUALITY MANAGEMENT
POLICY AND PROCEDURE

PENNSYLVANIA BUREAU OF WATER QUALITY MANAGEMENT

POLICY AND PROCEDURE

January 24, 1977

SEALING OF UNDERGROUND MINES

POLICY

A. Sealing of underground mine openings is an essential element of mining that must be achieved to prevent pollution from mining upon the cessation of an operation. To insure that the mine seals are effective, the design, location and installation must be reviewed and monitored.

PROCEDURE

B. Upon learning that a mine is inactive or that mining has been completed, either the BWQM field inspector or the Deep Mine Safety inspector shall arrange a joint, on-site inspection. The purpose shall be to determine the conditions of the openings, the type of seal the operator proposes to install, the specific details of the seal design and the location of the proposed seals.

C. Upon learning that a mine is inactive or that mining has been completed, the BWQM Regional Office will request from the mine operator two copies of the most recent mine map, one copy of which will be sent to Central Files through the Division of Industrial Wastes and Erosion Regulation. Nothing here should be construed as relieving the operator of the obligation to submit a final mine map as required by the Pennsylvania Bituminous Coal Mine Act.

D. The joint, on-site inspection should normally follow receipt and review of the mine map by the Office of Deep Mine Safety and the Division of Industrial Wastes and Erosion Regulation.

E. The on-site inspection for the BWQM should be made by the Division geologist and the Regional geologist together. However, if one geologist is unable to participate, the other geologist will represent the BWQM.

F. At the time of the joint, on-site inspection the Deep Mine Safety inspector and the BWQM geologist(s) will complete and sign the Mine Closure Report form recommending approval of the proposed sealing plans or refusal because natural or operating conditions have changed since the permit was issued. The deficiencies of the proposed sealing plan should be noted on the Mine Closure Report form. The joint inspection may result in recommendations for a further evaluation of the conditions or for additional inspections. The Deep Mine Safety inspector must accompany all subsurface inspections.

G. The following will receive one copy of the signed Mine Closure Report form and the letter of Verification of Sealing: 1) the mine operator, 2) the Deep Mine Safety inspector, 3) the Commissioner of the Office of Deep Mine Safety, 4) the BWQM Regional Office, and 5) the Division of Industrial Wastes and Erosion Regulation.

H. If the mine closure plan is approved, the operator may proceed to seal the openings in accordance with the permit. If the proposed seal design is refused, the operator must submit revised plans based on the findings of the joint, on-site inspection and formally apply for a permit amendment through the appropriate BWQM Regional Office. Copies will be distributed by the BWQM Regional Office to the Division of Industrial Wastes and Erosion Regulation and the Office of Deep Mine Safety for review. If the revised plans are approved, the operator may proceed to seal the openings in accordance with the amended permit.

I. If the field inspectors disagree on the plan of closure, they will note their differences on the Mine Closure Report form. Copies of the form will be forwarded to the Commissioner, Office of Deep Mine Safety, and the Chief, Division of Industrial Wastes and Erosion Regulation, for resolution. If the differences cannot be resolved by the Commissioner, Office of Deep Mine Safety, and the Chief, Division of Industrial Wastes and Erosion Regulation, the Secretary of the Department of Environmental Resources will resolve the difference. The Chief, Division of Industrial Waste and Erosion Regulation, shall notify the BWQM Regional Office of the results of the resolution. The procedure shall follow as outlined in paragraph H.

J. The Deep Mine Safety inspector will notify the BWQM Regional Office of the time for completion of the sealing operation and will arrange with the BWQM geologist for a final joint inspection of the completed seal. A letter of Verificaiton of Sealing will be signed by the Deep Mine Safety inspector and the BWQM geologist at or after the final inspection. Copies of the signed letter will be distributed as outlined in paragraph G.

K. Inspections of sealed mines should be made every six months for a period of three years and biannually for a period of four succeeding years. Inspections will be the responsibility of the BWQM Regional Office and will be performed by the Regional Geologist or an Environmental Protection Specialist experienced in mine drainage. In case of leakage, rupture, fracture or any other sign indicating a loss of soundness of the seals or surrounding bedrock during the above period of time, the company should be notified that corrective work is necessary. Failure of the company to respond to the notification should result in appropriate enforcement action.

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APPENDIX D

WEST VIRGINIA ADMINISTRATIVE REGULATIONS
CHAPTER 20-5 AND 20-5A

WEST VIRGINIA ADMINISTRATIVE REGULATIONS

STATE WATER RESOURCES BOARD

Chapter 20-5 and 20-5A
Series 1
(1965)

Subject: REQUIREMENTS GOVERNING THE DISCHARGE OR DEPOSIT OF SEWAGE, INDUSTRIAL WASTES AND OTHER WASTES INTO THE WATERS OF THE STATE.

The State Water Resources Board and the Chief of the Division of Water Resources in the State Department of Natural Resources, under Chapter 20, Article 5A, Code of West Virginia, have the power and authority to determine whether any person, firm, municipality or corporation is polluting any of the waters of the State and to prevent, control, eliminate or reduce such pollution. In making such determination, due consideration shall be given in accordance with the public policy of the State of West Virginia, to the use of available and reasonably practicable methods to control and reduce pollution. In so doing, recognition shall be given to the fact that each stream in the State may represent a separate problem and further, that the use of a watercourse for assimilation of wastes is proper so long as the net results do not cause or contribute to conditions hereinafter not allowed.

* * * * *

Section 5. ACID MINE DRAINAGE CONTROL MEASURES

5.01 Certain acid mine drainage control measures were adopted by the Ohio River Valley Water Sanitation Commission and promulgated as Resolution No. 5-60, as amended January 10, 1963. The State of West Virginia is a member of the Ohio River Valley Water Sanitation Compact and as such has agreed to carry out the control measures so established. Waters of the State of West Virginia are being polluted by acid discharges from coal mining and related operations, hereinafter referred to as "acid mine drainage," contrary to the language and intent of the State Water Pollution Control Law.

5.02 It has been demonstrated that the conscientious application of certain principles and practices will, under certain conditions, alleviate the pollution from acid mine drainage. Therefore, in furtherance of the policy and procedures of the State Water Resources Board, the following measures are hereby adopted by the Water Resources Board for the control of acid mine drainage pollution in the State of West Virginia:

(a) 1. Surface waters and ground waters shall be diverted where practicable to prevent the entry or reduce the flow of waters into and through workings.

2. Water that does gain entry to the workings shall be handled in a manner which will minimize the formation and discharge of acid mine drainage to streams.

(b) Refuse from the mining and processing of coal shall be handled and disposed of in a manner which will minimize discharge of acid mine drainage therefrom to streams. Where acid-producing materials are encountered in the overburden in stripping operations, these materials shall be handled so as to prevent or minimize the production of acid mine drainage, taking into consideration the need for stream pollution prevention and all economic factors involved.

(c) Discharge of acid mine drainage to streams shall be regulated insofar as practicable to equalize the flow of daily accumulations throughout a 24-hour period.

(d) Upon discontinuance of operations of any mine, all practicable mine-closing measures, consistent with safety requirements, shall be employed to minimize the formation and discharge of acid mine drainage.

(e) Under appropriate circumstances, consideration shall be given to the treatment of acid mine drainage by chemical or other means in order to mitigate its pollution properties.

APPENDIX E

THE MARYLAND REGISTER
TITLE 08, SUBTITLE 13, SECTION 02

08. 13.02. 11 Mine Opening Sealing.

A. Mine opening seals shall be designed and constructed, using proven techniques and materials, in such manner as to prevent seepage or transfer of surface and ground water through the opening, to eliminate health and safety hazards, and to protect the integrity of the seal against roof falls, subsidence, hydrostatic pressure,and other destructive forces.

B. All mine opening seals shall be certified by a registered professional engineer as having been constructed according to the approved design.

08. 13.02. 12 Barriers.

A. Barriers shall be of sufficient width to withstand the weight of supported overburden without squeezing, crushing, or punching into the floor or roof of the mine, and shall be capable of withstanding anticipated hydrostatic pressure and deterioration over time without seepage or failure.

B. Barriers shall be a minimum of 50 feet in width or greater as determined by the Formula

$W = 20 + 4T + 0.1D$,

where:

W = width of the barrier,
T = thickness of the coal seam, and
D = depth of overburden.

Perimeter barriers shall have a minimum of 50 feet of overburden.

C. Barriers surrounding gas, oil, or water wells shall be a minimum of 300 feet in diameter unless written consent of the owner and prior approval of the Bureau is obtained.

D. Where mulitple seam mining occurs, barriers in upper seams shall be supported by solid barriers in lower seams, the width of lower barriers shall be determined by the Formula

$$Wl = Wu + 0.6T,$$

where

Wl	= width of lower barrier,
Wu	= width of upper barrier, and
T	= the thickness of the strata between the two coal seams.

E. Barriers with hydrostatic head shall be at least 1 foot wide for each foot of hydrostatic head.

APPENDIX F

REGULATIONS OF THE STATE OF TENNESSEE DEPARTMENT OF LABOR TITLE 58-1012

58-1012. ABANDONED WORKINGS. -- (a) The entrances to abandoned workings shall be posted to warn unauthorized persons against entering the territory.

(c) Where abandoned workings are sealed, the sealing shall be done in a substantial manner with incombustible material. In every sealed area, one or more of the seals shall be fitted with a pipe and cap or valve to permit the gases behind the seals to be sampled and also to provide a means of determing any existing hydrostatic pressure.

APPENDIX G

CODE OF FEDERAL REGULATIONS TITLE 30
SECTION 75:1711

CODE OF FEDERAL REGULATIONS

TITLE 30

Sec. 75.1711. SEALING OF MINES.

(Statutory Provisions)

On or after March 30, 1970, the opening of any coal mine that is declared inactive by the Operator, or is permanently closed, or abandoned for more than 90 days, shall be sealed by the operator in a manner prescribed by the Secretary. Openings of all other mines shall be adequately protected in a manner prescribed by the Secretary to prevent entrance by unauthorized persons.

Sec. 75.1711-2. SEALING OF SLOPE OR DRIFT OPENINGS

Slope or drift openings required to be sealed under Sec. 75.1711 shall be sealed with solid, substantial, incombustible material, such as concrete blocks, bricks or tile, or shall be completely filled with incombustible material for a distance of at least 25 feet into such openings.

APPENDIX H

CODE OF FEDERAL REGULATIONS TITLE 30
SECTIONS 817.15 and 817.50

Sec. 817.15.

CASING AND SEALING OF UNDERGROUND OPENINGS: PERMANENT

When no longer needed for monitoring or other use approved by the regulatory authority upon a finding of no adverse environmental or health and safety effects, or unless approved for transfer as a water well under Section 817.53, each shaft, drift, adit, tunnel, exploratory hole, entryway or other opening to the surface from underground shall be capped, sealed, backfilled, or otherwise properly managed, as required by the regulatory authority in accordance with Sections 817.13 and 817.50 and consistent with 30 CFR 75.1711. Permanent closure measures shall be designed to prevent access to the mine workings by people, livestock, fish and wildlife, machinery and to keep acid or other toxic drainage from entering ground or surface waters.

Sec. 817.50.

HYDROLOGIC BALANCE: UNDERGROUND MINE ENTRY AND ACCESS DISCHARGES

(a) Surface entries and accesses to underground workings, including adits and slopes, shall be located, designed, constructed, and utilized to prevent or control gravity discharge of water from the mine.

(b) Gravity discharge of water from an underground mine, other than a drift mine subject to Paragraph (c) of this Section, may be allowed by the regulatory authority, if it is demonstrated that --

(1)(i) The discharge, without treatment, satisfies the water effluent limitations of 30 CFR 817.42 and all applicable State and Federal water quality standards; and

(ii) That discharge will result in changes in the prevailing hydrologic balance that are minimal and approved postmining land uses will not be adversely affected; or,

(2)(i) The discharge is conveyed to a treatment facility in the permit area in accordance with Section 817.42(a);

(ii) All water from the underground mine discharged from the treatment facility meets the effluent limitations of Section 817.42 and all other applicable State and Federal statutes and regulations; and

(iii) Consistent maintenance of the treatment facility will occur throughout the anticipated period of gravity discharge.

(c) Notwithstanding anything to the contrary in Paragraphs (a) and (b) of this Section, for a drift mine first used after the implementation of a State, Federal, or Federal lands program and located in acid-producing or iron-producing coal seams, surface entries and accesses shall be located in such a manner as to prevent any gravity discharge from the mine.

APPENDIX J

A COMPREHENSIVE MATHEMATICAL MODEL SUITABLE FOR THE PREDICTION OF COUPLED HYDRODYNAMICS, HEAT TRANSFER AND CHEMICAL-SPECIES TRANSPORT IN SATURATED POROUS MEDIA.

INTRODUCTION

In this appendix, prior to the description of the present mathematical model recent published literature concerning prediction procedures for flows in porous media coupled with heat and/or mass transfers, is first reviewed. Underground porous media are the primary subjects of concern, and the review is intended to provide an overview of the state-of-the-art. For this reason, it is necessarily brief.

The prediction procedures considered here include both analytical solution techniques for simplified governing equations and mathematical models based upon numerical solution techniques for sets of coupled and uncoupled governing differential equations. Attempts have not been made to prepare here a detailed classification of these procedures based upon vigorous mathematical criteria. The intention rather is to present a coherent summary which highlights the salient features and recent advances.

It is recognized that in preparing this review complete attention may not have been paid to the degree of validation, in respect of reliable field and laboratory data, that each available procedure may have been subjected to. The degree of sophistication and flexibility built into the procedures which permit them to accept such data in some convenient form, will be considered sufficient for purposes of review. This is largely due to the paucity of data, in sufficient quantity and of suitable quality, available for purposes of validation.

In the following sub-sections, reviews are separately presented for hydrodynamic aspects, aspects of chemical-species transport, and heat-transfer aspects respectively. This loose sub-division is maintained purely for reasons of convenience in presentation.

HYDRODYNAMICS

Analytical techniques for solving simplified equations of groundwater mass balance have been employed now for a number of years. The techniques involve basic assumptions about the geometric configuration of the flow domain and the uniformity of material properties. The employment of these techniques usually results in closed-form expressions for hydraulic head as a function of space and time. The velocity fields are then extracted by the approximate use of Darcy's law.

The deployment of such techniques for the flow distribution in multiple inter-connected aquifers has been reported by Bredehoeft and Pinder (1970). A recent and elegant treatment of leakage flow between aquifers is presented by Dever and Cleary (1979). A principal assumption involved in the above procedures is that the flow field is entirely saturated. The analysis of the more difficult problem of unsaturated flows has received relatively little quantitative attention. Braester et al. (1971) have prepared a comprehensive survey of governing equations for unsaturated flows. Gambolati (1973) has presented a discussion of vertical unsaturated flow analysis. It may be concluded, however, that versatile analytical procedures for saturated-unsaturated flow predictions do not, in general, exist. A simple one-dimensional procedure for predicting purely unsaturated flows has, however, recently been reported by McWhorter and Nelson (1979) who applied it to the prediction of seepage beneath tailings ponds.

Recent years have seen the proliferation of mathematical models based upon numerical schemes for solving the non-linear form of the mass-conservation equation. Narasimhan and Witherspoon (1977) review much of the current literature on the subject and indicate that both finite-difference and finite-element techniques have been employed with varying degrees of success. The premier ones of the former variety are those developed by Bredehoeft and Pinder (1970), Prickett and Lounquist (1971), Cooley (1974), Trescott et al. (1976), and Sharma (1979). Of the latter variety, the works by Narasimhan et al. (1976), Neuman (1973) and Pinder (1973) represent the principal ones. Trescott and Larson (1977) compare the efficacy of iterative methods used to solve sets of algebraic equations resulting from any form of numerical discretization.

Numerical procedures particularly suited to the prediction of saturated/unsaturated flows have also been developed (see for example Freeze, 1971; Narasimhan et al., 1977; Sharma and Hamilton, 1978; etc). The numerical formulation of leakage interactions between elements of a multiple aquifer system are extensively discussed by Frind (1979). The simulation of individual wells as well as the interactions amongst them have been reported by Prickett and Lounquist (1971) and Akbar et al. (1974).

An assessment of these and other similar procedures, in formulation and especially in implementation, has been prepared recently by Weston (1978). In agreement with this assessment, it is agrued here that numerical procedures, of sufficient degrees of comprehensiveness are presently available for application to the range of problems currently encountered. The major area of weakness in this is the prediction of flows in porous media with superposed fracture distributions. The state-of-the-prediction art for such flows has been thoroughly reviewed recently by Gringarten (1979).

MASS TRANSFER

The use of the term mass transfer here is intended to signify the transport of reacting chemical species within porous media by the complex interaction of several physical and chemical mechanisms. The set of such mechanisms considered here as a basis for review is:

- convection;
- diffusion and dispersion;
- buffering of pH;
- chemical precipitation by reactions with the solid matrix as well as the interstitial water;
- hydrolysis and precipitation;
- oxidation-reduction reactions;
- radioactive decay;
- volatilization;
- mechanical filtration;

- biological degradation; and,
- cation-exchange reactions.

It must be emphasized that specialized knowledge of the in-situ effects of individual mechanisms are understood only to a limited extent. The set of sophisticated measurements necessary to quantify these influences are currently being made in a variety of contexts. It is thus reasonable to suppose that soon the data obtained from these measurements will be available for purposes of refining the available mathematical models.

Analytical solutions to the convective dispersion equation have been developed by a number of authors, each of whom has been interested in specific geometric configurations and specific chemical species. The deployment of these solutions has been governed to a large extent by the requirements of the technical discipline encompassing each problem. For instance, a one-dimensional solution including adsorption effects has been developed by Gupta and Greenkorn (1973) as a tool in soil-chemistry. The work by Aikens et al. (1979) presents a variety of useful analytical solutions which take radioactive decay into account. Such solutions are indeed simple to use, and provide order-of-magnitude results in respect of concentration distributions with a modicum of effort. However, as geometries, material properties or the reactive mechanisms themselves become more complex, it is more convenient to employ mathematical models based upon numerical solution techniques.

One-dimensional models of this type abound in the literature. An interesting work by Selim et al. (1977) is concerned with finite-difference simulations of reactive solute transport through multilayered soils. Davidson et al. (1978) report the extension of this work to the finite-difference treatment of coupled adsorption, convective dispersion as well as biological degradation. This work represents an excellent study of the effects of pesticides in soils. The recent publication by Konikow and Bredehoeft (1978) describes a comprehensive finite-difference procedure for solving the coupled flow and chemical-species transport equations. A similar procedure, employing a sophisticated hybrid differencing scheme, has also been developed by Sharma (1979). These procedures are typical of economical schemes being currently reported, and to be entirely valid must be supplied with reliable physical and chemical data.

In like fashion, finite-element based numerical methods have been developed by researchers for predicting chemical-species transport in porous media. Rubin and James (1973) present one such method which uses the Galerkin approach. Gray and Pinder (1976) discuss the efficacy of this and other finite-element approaches, and in addition compare their relative accuracies. The application of one such approach by Pinder (1973) to groundwater contamination in Long Island is a meticulously-documented study augmented by field measurements. The application of finite-element methods to other types of problems involving transport of chemical species has also been achieved. One such application by Kealy et al. (1974) involves the analysis of seepage from tailings ponds. In this connection the work by Duguid and Reeves (1976) is well known. Weston (1978) presents a comprehensive review of major models of the above types and commends some for routine application. In short, a wide range of models covering a range of applicability is presently available for use in predicting the transport of reacting chemical species. The data requirements for these models are not available in the same level of quality.

HEAT TRANSFER

The analysis of heat transfer coupled with fluid flow in porous media, has also been conducted using both analytical and numerical techniques. The analytical solutions have, depending on specific boundary conditions, have much in common with those for transport of chemical species. However, the range of application of both analytical and numerical solutions for heat transfer is limited when compared with transport of chemical species.

The work by Harlan (1973) on the prediction of freezing in soils is an excellent early example of the use of a numerical procedure for the analysis of freezing fronts in porous media. Likewise, Holst and Aziz (1972) as well as Rubin and Roth (1979) examine aspects of thermally-induced convection in porous media and the stability of such flows. Special attention has been paid by Runchal et al. (1978) to the problem of heat-transfer effects, resulting from the disposal of high-level radioactive waste, upon groundwater motion. All such procedures depend, of course, on the supply of adequate field data, of sufficient quantities and of sufficient quality for purposes of input and validation. Such data, in respect of heat transfer, is extremely sparse, and hence most heat transfer models must be considered to be in a state of development. A recent example of field measurements of temperature effects in porous-media flows is that by Molz et al (1978). These measurements were specifically made in connection with thermal energy storage in aquifers. The problems involved in such storage have been discussed by Werner and Kley (1977). Theoretical studies of this problem, using both finite-difference and finite-element methods have been reported. Amongst the former is the work by INTERCOMP (1976); examples of the latter are: Mercer et al (1975); and, Papadopulos and Larson (1978).

MATHEMATICAL FOUNDATIONS

PREAMBLE

In what follows, a mathematical description is provided of a mathematically general version of the model. Two-dimensional versions of the model have been sucessfully employed in a variety of engineering applications. A simple three-dimensional version of the model has been developed, tested and applied recently by Dames & Moore (Sharma, 1979; and Hamilton and Sharma, 1979). It is economical of computational effort, whilst retaining the sophistry of physical and chemical formulations embedded other models mentioned above.

GOVERNING EQUATIONS

The symbols in the following equations are described in the Nomenclature list.

a. Piezometric head, h:

It can be shown (Narasimhan, 1975) that the partial differential equation governing the distribution of piezometric head is:

$$S \frac{\partial h}{\partial t} = \frac{\partial}{\partial x}\left\{ \Gamma_x^h \frac{\partial h}{\partial x} \right\} + \frac{\partial}{\partial y}\left\{ \Gamma_y^h \frac{\partial h}{\partial y} \right\} + \frac{\partial}{\partial z}\left\{ \Gamma_z^h \frac{\partial h}{\partial z} \right\} + s^h \quad . \tag{1}$$

b. Fluid velocity components, U, V, W:

The well known Darcy Hypothesis is used to relate the velocity components to the distribution of piezometric head thus:

$$\begin{aligned} U &= -\Gamma_x^U \frac{\partial h}{\partial x} + s^U \\ V &= -\Gamma_y^V \frac{\partial h}{\partial y} + s^V \\ W &= -\Gamma_z^W \frac{\partial h}{\partial z} + s^W \end{aligned} \tag{2}$$

c. Concentration of chemical species j, C_j:

It has been shown (Sharma, 1979) that the convective transport equation for the concentration of species is:

$$\frac{\partial}{\partial t}(dC_j) + \frac{\partial}{\partial x}(dUC_j) + \frac{\partial}{\partial y}(dVC_j) + \frac{\partial}{\partial z}(dWC_j)$$

$$= \frac{\partial}{\partial x}\left\{ \Gamma_x^{C_j} d \frac{\partial C_j}{\partial x} \right\} + \frac{\partial}{\partial y}\left\{ \Gamma_y^{C_j} d \frac{\partial C_j}{\partial y} \right\} + \frac{\partial}{\partial z}\left\{ \Gamma_z^{C_j} d \frac{\partial C_j}{\partial z} \right\}$$

$$+ s^{C_j} \tag{3}$$

d. Thermal energy, T:

The equation governing the conservation of thermal energy may be expressed as follows.

$$\frac{\partial}{\partial t}(\bar{C}_v T) + \frac{\partial}{\partial x}(U\bar{C}_p T) + \frac{\partial}{\partial y}(V\bar{C}_p T) + \frac{\partial}{\partial z}(W\bar{C}_p T)$$

$$= \frac{\partial}{\partial x}\left\{\Gamma_x^T \frac{\partial T}{\partial x}\right\} + \frac{\partial}{\partial y}\left\{\Gamma_y^T \frac{\partial T}{\partial y}\right\} + \frac{\partial}{\partial z}\left\{\Gamma_z^T \frac{\partial T}{\partial z}\right\}$$

$$+ \quad s^T \qquad (4)$$

INITIAL AND BOUNDARY CONDITIONS

Initial and boundary conditions, respectively within and on the boundary of the solution domain, for each of the dependent variables must be supplied in order to complete the mathematical specification of the problem.

Initial conditions designate the distribution of h, C_j and T, over the entire solution domain of interest, at the commencement of the solution. Such conditions may be obtained from the results of a field-measurement program , as for example would be the regional piezometric head distribution. Alternatively, they may be obtained from laboratory-scale experiments, as for example the ambient concentration of chemical species in ground water. They may also be supplied from the results of previous calculations of a similar nature.

Boundary conditions represent variations of the dependent variables, their fluxes or combinations thereof, at the boundaries of the solution domain. Such conditions may also be obtained from the results of a field-measurement program, as would be the case with recharge boundaries. It is important to note that boundary conditions may vary with time, and as a result, influence the accuracy of results obtained with computational solution procedures.

In addition to the above it must be noted that certain man-made as well as natural influences affect the distribution of h, C_j and T

within the solution domain. Such influences include discharging (and recharging) wells; artifical and natural barriers occuring locally to flow (and, heat and mass transfer) within the domain.

NUMERICAL SOLUTION PROCEDURE

GENERAL

The numerical procedure adopted in the present model is of the integrated finite-difference (IFD) variety with origins in an earlier work on computational fluid mechanics and heat transfer (Sharma, 1974). Details of the present procedure are available in Sharma (1979). A brief description is provided in this section.

NUMERICAL GRID AND VARIABLE LOCATIONS

An illustration of the numerical grid adopted in the x-y plane is illustrated in Figure J-1. In this figure the faces of control volumes, used in deriving the discretised equations, are indicated as dashed lines. The intersections of grid lines, termed grid nodes, are chosen to lie in the geometric center of the associated control volumes. An exception is made at the boundaries of the domain wehre the nodes lie on the boundaries themselves.

All problem variables, with the exception of the velocity components U, V and W, are presumed to be located at grid nodes. The x-direction velocity components U are presumed to lie on the intersections of the control-volume faces in the y-z plane with the x direction grid lines. Likewise, the y-direction velocity components V are presumed to lie on the intersections of the control-volume faces in the x-z planes with y-direction grid lines. In general, with the possibility of using variable grid spacings in any given direction, it is important to note that velocity components in any given direction do not lie exactly midway between grid nodes in that direction.

THE DISCRETISED EQUATIONS

Discretised forms of the partial-differential equations (1), (3), and (4) are obtained by integrating them over the above-mentioned control volumes. It is presumed for purposes of integration that the dependent variables vary linearly between successive grid nodes. Furthermore, one such discretised algebraic equation per dependent variable, may be derived thus for each control volume within the solution domain. Such an algebraic equation represents, in finite-difference form, the conservation of mass or of chemical species. The preservation of these conservation principles in the simultaneous solution of the algebraic equations permits an exact accounting of mass and momentum to be made. It is of great importance to note that such precise accounting of chemical species is vital in problems concerning the limited disposal of waste at a given site. Many, otherwise praiseworthy mathematical models, do not ensure that this is the case.

The discretised equations, at an arbitrary grid node , have the following forms:

a. piezometric head:

$$\left\{ \sum_{P} A^{h} - SN_{P}^{h} \right\} h_{P} = \sum_{\substack{i = \\ E,W,N,S,F,B,O}} A_{i}^{h} h_{i} + SO_{P}^{h} \qquad .(5)$$

b. Species concentration:

$$\left\{ \sum_{P} A^{C_j} - SN_{P}^{C_j} \right\} C_{j_P} = \sum_{\substack{i = \\ E,W,N,S,F,B,O}} A_{i}^{C_j} C_{j_i} + SO_{P}^{C_j} \qquad .(6)$$

c. Temperature:

$$\left\{ \sum_{P} A^{T} - SN_{P}^{T} \right\} T_{P} = \sum_{\substack{i = \\ E,W,N,S,F,B,O}} A_{i}^{T} T_{i} + SO_{P}^{T} \quad . (7)$$

In the above, A's denote coefficients computed from known (or sometimes presumed known temporarily) values of hydraulic conductivity, dispersion coefficients etc.; and SO, SN are components of a linearised source term; i denote respectively the neighbouring grid nodes in space; O denotes the coefficient associated with the previous-time value of the appropriate dependent variable; and F, B denote the forward or backward application of the block correction procedure. Various forms which the source terms may take are shown in Table J-1 presented overleaf.

THE SOLUTION ALGORITHM

The sets of simultaneous algebraic equations noted above are solved by the efficient application of an alternating-direction, heavily-implicit, line-by-line solution algorithm coupled to a plane-by-plane block correction proecedure. Details are provided by Sharma (1979). This algorithm applied iteratively leads to relatively monotonic solutions for most problems with commonly-encountered boundary conditions.

COMPUTER-PROGRAM DETAILS

The algorithm mentioned above has been incorporated into a set of computer programs written for one-, two- and three- dimensional problems. These programs, called TARGET (for Transient Analyser of Reacting Ground Water Effluent Transport), are written in standard FORTRAN-IV. They are thus capable of being run on most available computers. On a CDC-6600 machine a typical computer run for an unsteady two-dimensional problem requires approximately 60 seconds of central processor time.

SOME PREDICTED RESULTS

For purposes of testing the computer program TARGET and to demonstrate the accuracy of results predicted thereby, a few test runs were made of a selected problem. The problem posed is that of unsteady convective dispersion in one space dimension.

Grid-dependency tests were first conducted to determine the effect of grid-size upon numerical accuracy. It was observed in that sufficiently accurate results may be obtained with a reasonable number of grid nodes. Further tests investigating the dependence of accuracy upon the chosen time-step were conducted. These are illustrated in FiguresJ-2 and J-3 which indicate that for accuracy a sufficiently small timestep must be chosen. Subsequently predictions of a moving solute front were made. For a given set of parameters, the predicted results for this case may be observed in Figure J-3to compare very favourably with the corresponding analytical solutions.

TARGET has undergone numerous other tests, not reported here, to ensure that the program is essentially correct and that the results predicted with it are both plausible and valid. The validation tests are being continued in parallel efforts.

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NOMENCLATURE

A_i^{φ}	coefficients representing hydraulic conductivity or dispersion coefficients etc., for variable Ø at position i;
C_j	chemical species concentration of species j;
$C_{j,d}$	concentration, of species j, of discharge;
$\bar{C}_p$	average specific heat capacity at constant pressure;
$C_{p,d}$	specific heat capacity at constant pressure of discharge;
$\bar{C}_v$	average specific heat capacity at constant volume;
h	hydraulic head;
k_z	vertical hydraulic permeability;
$\dot{q}'''$	flow rate;
S	storage coefficient;
s^{φ}	source term for variable Ø;
SO_p	component of linearized source term for variable Ø at node p;
SN_p	component of linearized source term for variable Ø at node p;
T	temperature;
T_d	temperdatue of discharge;
t	time;
U	x-direction velocity;
V	y-direction velocity;
W	z-direction velocity;
x	horizontal cartesian coordinate direction;
y	horizontal cartesian coordinate direction;
z	vertical cartesian coordinate direction;
Γ_x^{φ}	effective hydraulic conductivity or dispersion coefficient for variable, in direction x;
ρ	density;
ρ_o	reference density.

Source Term Description	Algebraic Form
Piezometric heat distribution	
Injection or extraction flow rate	$\pm \quad \dot{q}'''$
Bucyancy	$- \frac{\partial}{\partial z}\left\{k_z \left[1 - \frac{\rho}{\rho_o}\right]\right\}$
Temperature distribution	
Injection or extraction rate	$\dot{q}'''\left\{C_{p,d} T_d - \bar{C}_p T\right\}$
Concentration distribution	
Injection or extraction rate	$\dot{q}'''\left\{C_{j,d} - C_j\right\}$

TABLE J-1

DESCRIPTION OF SOURCE TERMS

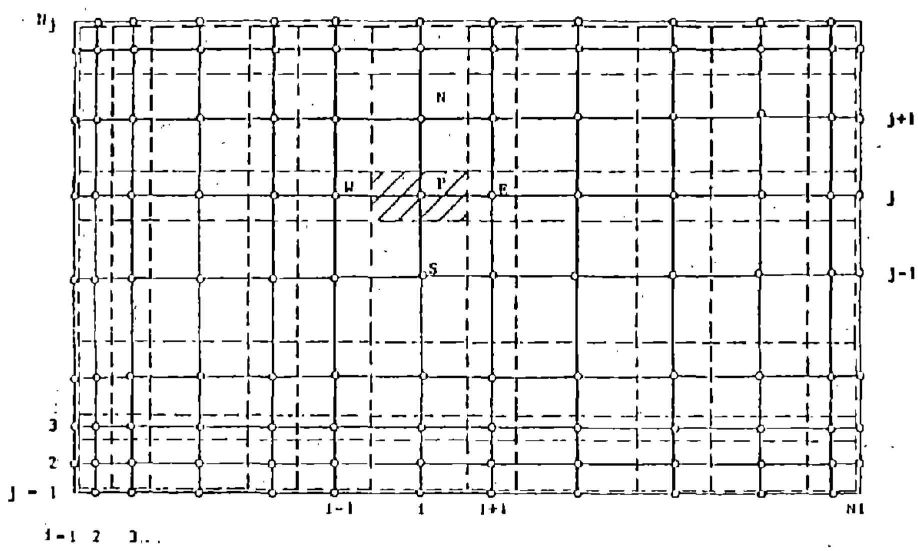

FIGURE J-1

SCHEMATIC ILLUSTRATION OF NUMERICAL GRID

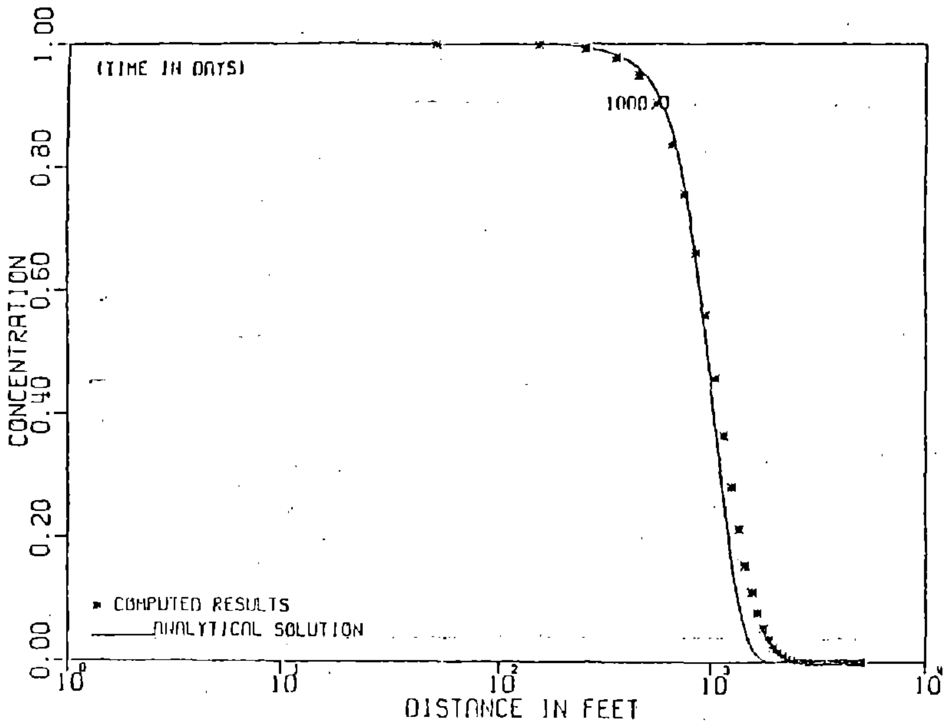

FIGURE J-2

TIME DEPENDENCE OF CONCENTRATION DISTRIBUTIONS

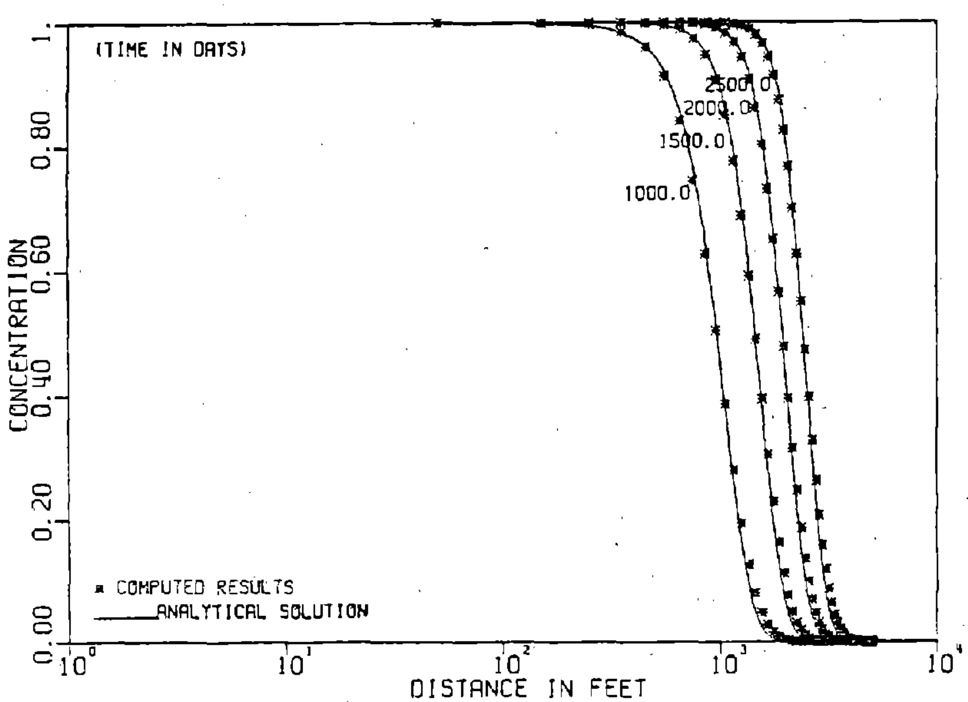

FIGURE J-3

FOR A MOVING SOLUTE FRONT
TIME DEPENDENCE OF CONCENTRATION DISTRIBUTIONS

APPENDIX K

TARGET INPUT GUIDE

TARGET INPUT GUIDE

Name	Dimension	Format	Description
ITEST	1	20I4	=1 print-out obtained before and after solution procedure =0 ordinary print-out procedure
NNVC	1	20I4	No. of variables to be calculated (set=3, ISOLVE will determine which variables are solved for.)
NNVP	1		No. of variables to be printed. Variables are solved and printed in the order determined by the next card input...if 1 to 3 are to be solved and are of interest, then set NNVP=3, etc.,
NVH	1		integer associated with hydraulic head
NMJ	1		integer associated with chemical concentration
NVT	1		integer associated with temperature
NVU	1		integer associated with X-direction velocity
NVV	1		integer associated with y-direction velocity
NGX	1	20I4	integer associated with x-direction "conductivity"
NGY	1		integer associated with y-direction "conductivity"
NSC	1		integer associated with storage coefficient
NST	1		integer associated with saturated thickness
NFT	1		integer associated with thickness of variables
ISTEP	1		initial step no.
NSTEPS	1		final step no. (time step)
IITER	1	20I4	initial iteration no.
NITERS	1		final iteration no.
ISTUN	1		=1: steady-state run =2: unsteady run
IREF	1	20I4	I index of reference node
JREF	1		J index of reference node
IRTAPE	1	20I4	"Read from" tape no.
IWTAPE	1		"Write to" tape no.
NLAYER	1	20I4	no. of layer in problem (up to 5)
NI	1	20I4	no. of I-direction nodes
NJ	1		no. of J-direction nodes
DX	NI x NJ	10F8.0	cell-widths in x-direction
DY	NI x NJ		cell-widths in y-direction
XMIN	1		minimum distance in x-direction
XMAX	1		maximum distance in x-direction
YMIN	1	10F8.0	minimum distance in y-direction
YMAX	1		maximum distance in y-direction
TIME	1		initial time
TIMAX	1		maximum time
DT	1		initial timestep
DTMAX	1	10F8.0	maximum timestep
TIMREF	1		reference time
EX	1		ratio by which to increase timestep
DELT	1		Δt by which it is permissible for time mismatches to be made
ISOLVE	3	20I4	in the same order as NVH, NMJ, NVT are numbered, set ISOLVE =0 do not solve for variable =1 solve for variable
NSWP	3	20I4	no of solution sweeps made during the solving of each variable

TARGET INPUT GUIDE (continued)

Name	Dimension	Format	Description
ISWP	1	20I4	index to determine positive or negative I-direction sweep 1st
JSWP	1		index to determine positive or negative J-direction sweep 1st
IXY	1		index indicating I or J direction sweep 1st
RELAX	3	10F8.0	relaxation factor
CRIT	1	10F8.0	criterion by which no. of iterations is automatically chosen
IPRINT	16	20I4	integer which specifies whether or not the variable is to printed (in the same order as NVH, NMJ etc.,) =1 print =0 no print
IPJUMP	1	20I4	not used at present
NPJUMP	1		step intervals at which print out is required
ITYP	1	20I4	I index of "typical" node
JTYP	1		J index of "typical" node
PTIME	50	10F8.0	times at which print-out is required (these are also used to specify pumping periods when wells are specified ie., 5 pumping periods for each of up to 20 wells)
HCX	5	10F8.0	hydraulic conductivity in x-direction
HCY	5	10F8.0	hydraulic conductivity in y-direction
STC	5	10F8.0	storage coefficient
POR	5	10F8.0	porosity
RTC	5	10F8.0	retardation coefficient
TCX	5	10F8.0	thermal conductivity in x-direction
TCY	5	10F8.0	thermal conductivity in y-direction
ADIS	1	10F8.0	coefficients for the following equation $D(V) = A + B\|v\|^C$ A=ADIS; B=BDIS, C=CDIS, D=dispersion coefficient
BDIS	1		
CDIS	1		
HFIX	1	10F8.0	fixed hydraulic head value at nodes with GFLAG = -1
AMJFIX	1		fixed concentration value at nodes with GFLAG = -1
TFIX	1		fixed temperature value at nodes with GFLAG=-1
PTIME	1	20I4	initial pumping schedule number
NPUMPS	1		number of pumps
IP	1	20I4	I index of 1st pump node
JP	1		J index of 1st pump node
PUMP	5	10F8.0	pumping rate at 1st pump node
AJPUMP	5	10F8.0	concentration injection rate at 1st pump node
TPUMP	5	10F8.0	temperature injection rate at 1st pump node
*	*	*	THE LAST 4 CARDS ARE REPEATED NPUMP TIMES
H	NI x NJ	10F8.0	initial hydraulic head array
AMJ	NI x NJ	10F8.0	initial concentration array
T	NI x NJ	10F8.0	initial temperature array
GFLAG	NI x NJ	20I4	geometry flag =0 ordinary node =1 constant flux (fixed gradient boundary) =-1 zero flux (fixed value boundary etc.,)
PGLAG	NI x NJ	20I4	property flag set = 1
TITLE	6 x 16	6A6	