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UP-DIP VERSUS DOWN-DIP MINING
An Evaluation

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UP-DIP VERSUS DOWN-DIP MINING

AN EVALUATION

by

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CINCINNATI, OHIO 45268

REVIEW NOTICE

FOREWORD

Man and his environment must be protected from the adverse effects of pesticides, radiation, noise and other forms of pollution, and the unwise management of solid waste. Efforts to protect the environment require a focus that recognizes the interplay between the components of our physical environment--air, water, and land. The National Environmental Research Centers provide this multidisciplinary focus through programs engaged in

- studies on the effects of environmental contaminants on man and the biosphere, and

- a search for ways to prevent contamination and to recycle valuable resources.

This report presents detailed results of a feasibility study of down-dip mining, a technique that appears to offer an alternative to sealing or permanant treatment of polluted effluents from underground coal mines after abandonment. Of special importance in the study were evaluation of pollution reduction effectiveness of the technique and the National water quality impacts of its widespread employment.

A. W. Breidenbach, Ph.D.
Director
National Environmental
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ABSTRACT

The report presents detailed results of a feasibility study of down-dip mining, a technique that appears to offer an alternative to sealing or permanent treatment of polluted effluents from coal mines after abandonment. The project included an evaluation of pollution reduction effectiveness, economic and engineering limitations, costs in varying situations, health and safety aspects, and National economic and water quality impacts of utilization of this technique.

Project efforts included location and evaluation of a pair of abandoned underground mines - one developed to rise, one developed to dip - closely similar in all other aspects. The principal goal of this portion of the project was confirmation of the theory that discharge water quality in down-dip mines is substantially better than that in up-dip mines. An active mine with units operating up-dip and down-dip was also evaluated to ascertain major advantages and disadvantages of each mode of operation. Key factors which could vary with different mining procedures were pumping and coal haulage. However, haulage cost differentials were found to be extremely small, and pumping costs, which are related to highly variable local infiltration rates, could not be predicted. Health and safety and National water quality and economic impacts were also evaluated. Findings of each phase of study are presented here.

This report was submitted in fulfillment of Grant Project Number 68-01-0465 by Skelly and Loy, Engineers-Consultants, under the sponsorship of the Environmental Protection Agency. Work was completed as of January 31, 1975.

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

ACKNOWLEDGMENTS

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Messrs. Norman Wetzel, Charles Merritt and Raymond Alvertro provided valuable data and technical support during the active mine site investigative portion of this study.

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Richard D. Thompson, Division of Industrial Waste and Erosion Regulation, Pennsylvania Department of Environmental Resources.

Principal investigators in this study were Jamison B. Warg and John W. Mentz.

SECTION I

CONCLUSIONS

Based upon the findings of this study, the following conclusions have been made:

- Acidity concentrations in the main discharge from the abandoned mine developed to the rise were consistently in excess of 2200 mg/l, while those of the discharge from the mine developed to the dip were generally less than 100 mg/l. Iron, sulfate, manganese and aluminum concentrations were also significantly higher in discharges from the up-dip mine.

- Since the abandoned mines evaluated in this study were similar in all other respects, the primary factor controlling water quality can only be the direction of mine development.

- Employment of down-dip mining techniques will cause no significant water quality improvements on a National scale, since recently proposed Federal effluent limitations guidelines for the coal industry will require all active mining operations to maintain acceptable water quality.

- Pollution formation reductions attributable to down-dip mining will be of major economic importance to mine operators who, as a result of proposed effluent guidelines, are faced with the prospect of perpetual treatment of discharges from their mines after closure.

- Implementation of down-dip mining in major acid-producing coal regions could have significant regional and National impact on treatment costs, which are tied directly to production costs.

- The minimal concentrations of acid mine drainage observed within the down-dip mine are attributed to local site conditions, and do not reflect any deficiency in the pollution reduction potentials of the down-dip mining technique.

- When all pertinent mining factors are considered, down-dip mining at the active site was no more or less productive than up-dip mining.

- Numerous physical factors (including geologic and hydrologic conditions), which are only marginally related to mine development or mining technique, can have combined effects that are sufficient to obscure any advantage or disadvantage of mining to the dip.

- Substantial declines in both total production and productivity per man-shift occurred at the active minesite after passage of the Health and Safety Act of 1969.

- In the active mine evaluated, coal haulage is by far the largest single power cost.

- For a constant length belt segment operating at optimum speed for coal haulage, cost differentials for haulage up-grade (as in a down-dip developing mine) over haulage down-grade (as in an up-dip mine) at any realistic grade angle is generally less than one half cent per metric ton - a relatively minimal economic impact.

- Despite extremely low conveyor belt haulage unit cost, percentage cost increases incurred by haulage up-grade rather than down-grade range from a few to several hundred percent.

- Pumping costs, which vary greatly according to geology, hydrology, techniques employed, and mining plan, may be substantially increased by down-dip mine development; but they will probably not reach the point of adversely affecting production economics.

- Effective sealing of mines in steeply dipping seams can only be assumed where mines have been developed to the dip and the lower outcrop or barrier integrity has been maintained.

- Since down-dip mining operations must remain in compliance with all facets of the Health and Safety Act of 1969, there are no adverse health and safety considerations which could affect utilization of this technique.

- Feasibility and applicability of down-dip mining techniques have already been established in numerous applications under various mining conditions.

- Alternative mining procedures are available to counter any increased water handling attributable to down-dip mining. The capital investment required to implement these techniques may, however, be higher than the benefits realized.

SECTION II

RECOMMENDATIONS

- Down-dip mining techniques should be implemented wherever they may have significant beneficial pollution reduction effects on discharge water quality.

- An in-depth evaluation of the alternate down-dip mining procedures proposed in this report should be conducted to determine their applicability, feasibility, productivity, and economics.

- Detailed analysis of down-dip barrier pillar and coal outcrop strength characteristics, permeability, variables controlling these factors, and procedures for estimating required thicknesses would prove beneficial in maximizing coal recovery while minimizing chances for outcrop or barrier failure.

SECTION III

INTRODUCTION

The fact that at least two thirds of all coal related acid mine drainage discharges from underground mines has been well documented by numerous studies. It has also been established that a very high percentage (75 - 90%) of this underground mine drainage pollution emanates from abandoned mine sites. Reasons for the high level of pollution from abandoned underground mines are largely related to mining methods utilized in the past. Historically, the oldest underground coal mines were generally drift mines; that is, they entered the coal seam where it intersected or outcropped on the land surface. It was also common to open the drift on a low side of the coal seam (the down-dip side), and mine toward the rise of the coal. This permitted easy haulage of loaded coal carts from the working coal face down-dip to the mine entry. In addition, as the mine was developed, most water infiltrating into the workings would drain by gravity, thereby minimizing the amount of pumping required to keep active areas dry.

Groundwater entering the mine workings frequently becomes polluted, and up-dip mining practices permit this mine drainage to flow freely from the mine both during active operations and after abandonment. Typically, mine atmospheres exhibit humidity near the saturation point. Thus, regardless of other sources of water, there is always moisture condensing on mine walls, roof and floor. This moisture, combined with oxygen in the mine atmosphere and pyritic materials in the coal, floor material, or roof rock, forms acid salts. The acid "weeps" down walls of the workings as condensation continues and newly condensed moisture replaces acid-bearing moisture. When acid reaches the floor of the workings, it is transported to another portion of the mine, or from the mine, by flowing water. This flowing water, usually the result of groundwater infiltration, may be steady or intermittent in nature depending on climatic conditions. In either case, if the mine has been developed to the rise and drains by gravity, acid drainage will eventually flow from the workings and subsequently degrade water quality of receiving streams.

Oxidation of pyrite and formation of acid mine drainage are not problems that decline rapidly in severity with time. Acid formation and discharge may continue for over a hundred years after closure as fresh pyritic materials are constantly exposed by minor roof collapses. In addition, groundwater infiltration and condensation of moisture from the mine atmosphere are constant sources of water. Water quality of discharges from acid-producing abandoned mines will improve only after the mine roof has completely collapsed into the workings, and pyritic materials have been leached out or sealed off from further oxidation. Thus, acid formation is a long-term continuing process and must be dealt with in a manner that will effectively keep oxygen from entering the mine.

Pollution emanating from active underground mines has continually decreased with increased effectiveness of water quality controls. In fact, active mines should soon cease to be a source of pollution when presently available control mechanisms of state and Federal government are fully employed. Most active mines are currently required to provide some form of treatment to contaminated drainage, and pollution from active underground mines is not considered a major problem today. However, upon closure of an underground mine, the question arises as to responsibilities of former mine operators in maintaining water quality, and duration of those responsibilities. The United States Environmental Protection Agency's effluent limitations guidelines for the coal industry, which could be enacted as early as July 1, 1975, would place permanent responsibility for maintenance of acceptable water quality in abandoned mine discharges upon the former operator.

Current technology for maintenance of acceptable water quality from underground mines is limited to mine sealing and treatment. Many mines developed to the rise and currently active can never be effectively sealed, because of tremendous hydraulic heads that would develop as the mines flooded. Heads in excess of 6 or 7 meters (20 to 23 feet) cannot be safely withheld, either by mine seals or outcrop barriers (which are often weakened by adjacent surface or underground mining). In such situations, constant seepage can occur through the seal or the adjacent coal outcrop barrier. In addition, there is always the possibility of failure of a seal or adjacent outcrop barrier, which could, depending upon the volume of water contained, have devastating effects. Discharges from mines located in acid-producing coal seams and exhibiting such deterrents to sealing will require constant treatment - an extremely costly solution to the mine drainage problem.

This study was initiated to investigate a specific mining technique which appears to offer an alternative to mine sealing or permanent treatment of effluents from abandoned coal mines. This method, referred to in this report as down-dip mining or mining to dip, is not really new, since it has been employed on a limited scale throughout the coal fields for many years.

The principle of down-dip mining involves entry to the coal seam at the highest point of mine development and mining toward lower coal elevations. In mines so developed, both coal and mine drainage must be removed from mine workings against the force of gravity. Until recent years, this technique was not utilized extensively since it often increased coal production costs substantially. However, improvements in mining, pumping, and haulage technology have greatly reduced cost differentials to the point where it is actually a viable and competitive alternative mining method.

In theory, one of the principal benefits of mining to dip is not actually realized until after mine abandonment. All water entering the workings drains down-dip, but there is no discharge point in the low side of the mine. Therefore, when a mine is abandoned and pumping ceases, the workings inundate naturally. Flooding isolates pyritic materials in the coal, overburden, and mine floor. Thus, pyritic oxidation virtually ceases and acid mine drainage production almost halts. When the mine has completely filled and begins to discharge from the entry on the up-dip side, water may still be acid in nature, until the previously formed pollutants drain from the workings. However, in a relatively short time, within a few years, discharge quality will substantially improve because no new acid is being formed in the workings. In this manner, mining to dip can greatly reduce costs of maintaining environmental quality after abandonment.

Specific goals of this evaluation of down-dip mining were cited in the project proposal:

1) Evaluate actual effectiveness of the technique by studying pollution formation within similar mines developed to rise and to dip.

2) Develop applicability of the technique and its economic and engineering limitations.

3) Determine cost variations that could be incurred by employing the technique in various situations.

4) Evaluate health and safety aspects of the technique.

5) Evaluate National water quality and economic impact of employing this technique.

6) Develop design criteria and general feasibility of utilizing this technique.

7) Generate recommendations for use by regulatory agencies to implement the technique, if feasible.

Water quality improvements attributable to use of down-dip mining (as opposed to up-dip mining) were determined by comparison and evaluation of abandoned underground mines. These mines were selected to be as similar as possible in all aspects, except that one mine was developed to rise and the other to dip. All available historical water quality data and mine mapping was obtained and closely scrutinized to ascertain mine conditions and extent of similarity between the two mines. To evaluate water quality, a periodic sampling and flow measuring program was initiated, with sampling stations at every point of discharge from both mines.

Initial project plans called for analysis of two active mines, one operating to rise and one operating to dip, with similar characteristics. Numerous contacts were made in an effort to locate a pair of suitable mines. State and Federal regulatory agencies concerned with underground coal mining and State mine inspectors from several regions were interviewed. Background knowledge and experience gained during completion of two nationwide studies of the coal industry were also utilized. Contacts with coal companies or firms operating captive coal mines included Consolidation Coal Company, Jones and Laughlin Steel Corporation, Bethlehem Mines Corporation, Peabody Coal Company, Southern Ohio Coal Company, Bradford Coal Company, and Lady Jane Collieries, Incorporated.

This search for active mining operations suitable for investigation revealed a number of important points. First, a cooperative pair of mines sufficiently similar to warrant further consideration could not be located. In many cases, mines are so large that they concurrently include both up-dip and down-dip development, and cannot be accurately evaluated precisely as up- or down-dip operations. On the other hand, mines small enough to be developed exclusively to rise or dip are uncommon and generally do not maintain sufficiently detailed, usable data on production or economics.

Based on these preliminary findings, consideration was given to evaluation of a single large mine in which different sections being developed to rise or to dip could be isolated. Production and economics in a large mine could be evaluated for either single production units or panel segments of the mine. Several coal producers contacted simply had no operations that adequately fit requirements of this study. Some did not have sufficiently detailed production and economic data breakdowns to permit analysis of different mine sections. Other mine operators were unwilling to divulge information they considered to be confidential, including specific production rates and detailed cost breakdowns. Some tentatively agreed to cooperate early in the study only to decline after discovering the level of detail required in data being sought. A recent deluge of environmentally oriented studies and surveys, many quite demanding of the coal industry, also made many companies hesitant to cooperate in yet another evaluation, regardless of its purpose. In fact, of the companies contacted, only Lady Jane Collieries, Incorporated, agreed to cooperate in this study. Fortunately, while not totally ideal, Lady Jane met most of the prerequisites established for a meaningful evaluation. Company representatives provided valuable production information and insights into economics of mining coal. Based on this information, other goals of the study were achieved and recommendations were developed. Findings of this study and all pertinent data are presented in this report.

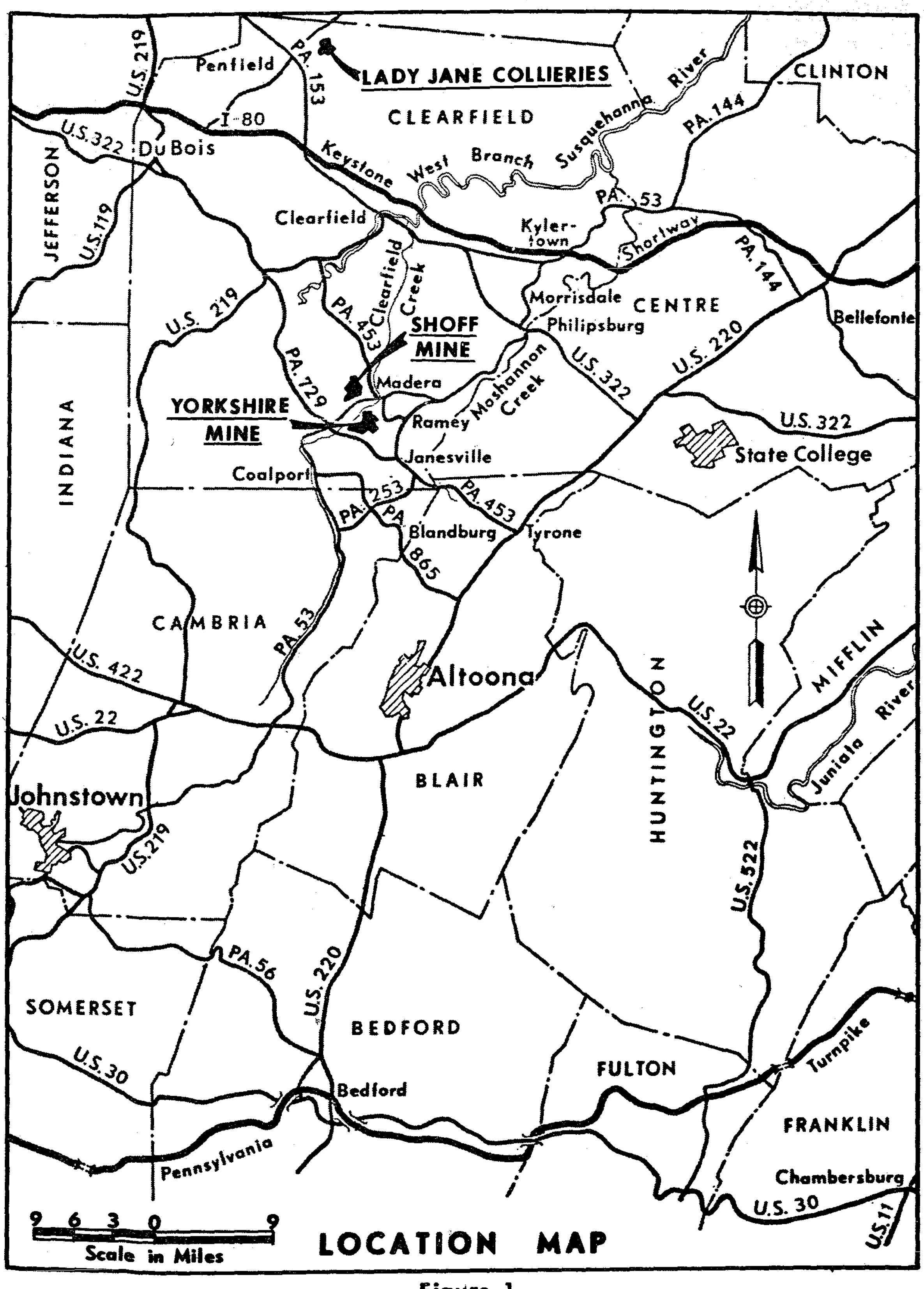

LADY JANE COLLIERIES
SHOFF MINE
YORKSHIRE MINE
CLEARFIELD
CLINTON
JEFFERSON
CENTRE
INDIANA
CAMBRIA
BLAIR
HUNTINGTON
MIFFLIN
SOMERSET
BEDFORD
FULTON
FRANKLIN
Penfield
DuBois
Clearfield
Kylertown
Morrisdale
Philipsburg
Madera
Ramey
Janesville
Coalport
Blandburg
Tyrone
Bellefonte
State College
Altoona
Johnstown
Bedford
Chambersburg
U.S. 219
U.S. 322
U.S. 119
I-80
PA. 153
PA. 144
PA. 53
PA. 453
PA. 729
PA. 253
PA. 865
U.S. 220
U.S. 422
U.S. 22
U.S. 522
PA. 56
U.S. 30
U.S. 11
Keystone
Shortway
West Branch Susquehanna River
Clearfield Creek
Moshannon Creek
Juniata River
Pennsylvania Turnpike
9 6 3 0 9
Scale in Miles
LOCATION MAP

Figure 1

SECTION IV

ABANDONED MINE SITE

EVALUATIONS

SELECTION CRITERIA

The primary basis for selection of the two abandoned mines studied in this project was their great degree of similarity. Such similarities were mandatory for this project in order to permit the most dependable evaluation of relative water quality. Both mines are similar in all of the following parameters: coal seam mined, coal quality (particularly sulfur content), approximate mine size, mining methods utilized, time period of operation, availability of mine history and mapping, geologic controls, hydrologic setting, topographic regime, and measureable discharges.

Abandoned mines selected for this in-depth study were the Shoff and Yorkshire No. 1 Mines in Clearfield County, Pennsylvania. The Shoff Mine was worked to rise, the most common mining procedure in past years, while the Yorkshire No. 1 Mine, hereafter referred to simply as the Yorkshire Mine, was operated to dip. Since abandoned mines worked to the rise are almost always greater sources of acid mine drainage, the Shoff Mine serves as a control to determine if the water quality of the down-dip Yorkshire Mine shows a difference in pollution load. Table 3, page 27, shows summary information.

MINE LOCATIONS

The abandoned workings of the Shoff Mine and the Yorkshire Mine are located in Clearfield County, Pennsylvania, just west of Madera. Both mines are in Bigler Township, and lie on opposite banks of Clearfield Creek. Both Shoff and Yorkshire Mines have major discharges which enter Clearfield Creek just upstream from the mouth of Muddy Run, a tributary highly polluted by acid mine drainage.

The Yorkshire Mine, which underlies approximately 220 hectares (544 acres), has a pair of drift entries just south of Clearfield Creek.

There are also two slope entries one kilometer (two-thirds of a mile) southeast of the drifts, adjacent to the mouth of Banian Run. The mine extends approximately 3.2 kilometers (two miles) south and southwest from the drifts, ranges in width from 300 to 1830 meters (1000 to 6000 feet), and underlies several small hills and portions of Banian Run.

The Shoff Mine workings lie only about 610 meters (2000 feet) north of the Yorkshire drift entries on the north bank of Clearfield Creek. Slightly smaller than Yorkshire, the Shoff Mine occupies 170 hectares (428 acres) beneath a single, large hill. The mine extends 1980 meters (6500 feet) north from Clearfield Creek, and ranges from 490 to 1650 meters (600 to 5400 feet) in width. Thirteen drift entries to the mine complex are located along its southern and eastern edges.

Locations, general configurations and entries of the Shoff and Yorkshire Mines are shown in Figure 2.

GEOLOGY

The Shoff and Yorkshire Mines were both developed in the highly pyritic Clarion coal, or "A" seam, which outcrops about 12 meters above stream level on both the north and south banks of Clearfield Creek. Stratigraphically the lowest Allegheny Group coal, the Clarion is underlain by sandstones comprising the upper portion of the Pottsville Group. All of the other major Allegheny Group coal seams - the Lower Kittanning "B" coal, the Middle Kittanning "C" coal, the Upper Kittanning "C - Prime" coal, the Lower Freeport "D" coal, and the Upper Freeport "E" coal - outcrop on the hillsides above both mine workings. All but the Upper Kittanning coal are at least locally strippable in this area, but additional deep mining was limited to some small Lower Freeport workings on the hilltop above the Shoff Mine. Figure 3 is a stratigraphic column showing relative positions and types of strata found in the study area.

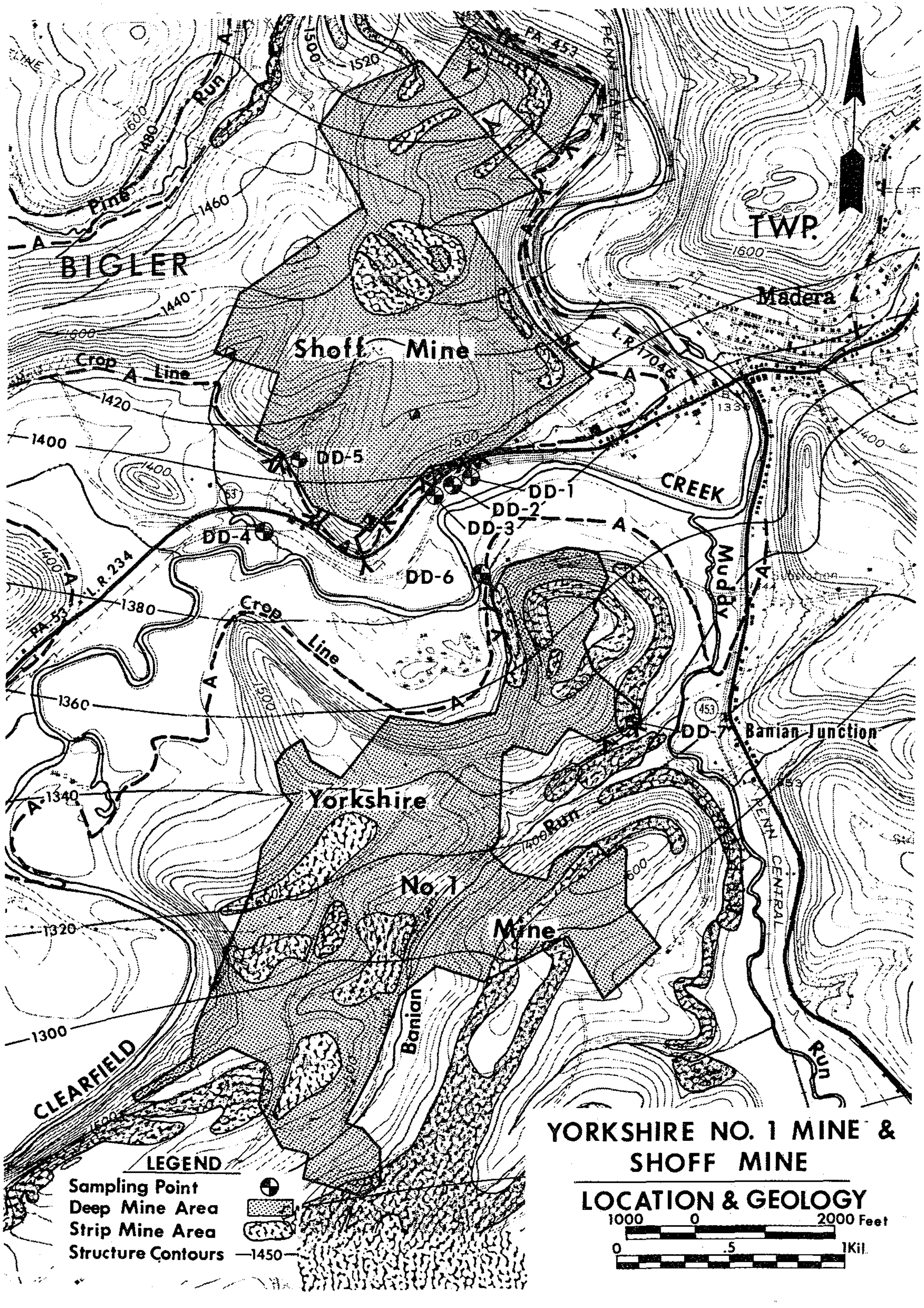
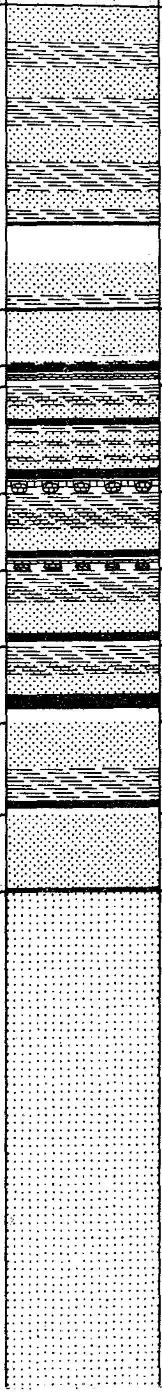

YORKSHIRE NO. 1 MINE & SHOFF MINE
LOCATION & GEOLOGY
1000 0 2000 Feet
0 .5 1Kil.
LEGEND
Sampling Point
Deep Mine Area
Strip Mine Area
Structure Contours —1450—
Shoff Mine
Yorkshire No. 1 Mine
BIGLER
TWP.
CLEARFIELD
Madera
Banian Junction
CREEK
Muddy Run
Banian Run
Pine Run
Crop Line
DD-1
DD-2
DD-3
DD-4
DD-5
DD-6
DD-7
PA 453
L.R. 234
L.R. 17046
PENN CENTRAL
1400
1380
1360
1340
1320
1300
1420
1440
1460
1480
1500
1520
1600

Figure 2



SYSTEM	SERIES	GROUP	FORMATION	COAL MEMBER	CHARACTERISTICS OF COAL MEMBER	GENERAL STRATIGRAPHY OF GROUP
CARBONIFEROUS	PENNSYLVANIAN	CONEMAUGH		Brush Creek	Very thin. Laterally discontinuous. No mining.	Erratic cyclic sequences of sandstone, siltstone, shale and thin coals. Lower 235 feet of group
				Mahoning	Average thickness = 1 foot. Thin and laterally discontinuous. Strip mined locally	
			Mahoning			
		ALLEGHENY	Freeport	Upper Freeport	Average thickness = 30 inches. Laterally continuous. Extensively strip mined with some deep mining.	Variable cyclic sequence of clay, claystone, carbonaceous shale, siltstone, sandstone, and mineable coals. Average thickness = 295 feet
				Lower Freeport Lower Freeport	Split into 2 benches with a 30 foot separation. Both are laterally continuous with average thickness of 30 inches and are extensively strip mined and deep mined	
			Upper Kittanning	Upper Kittanning	Average thickness = 24 inches. Laterally discontinuous but locally strip and deep mined.	
			Middle Kittanning	Middle Kittanning	Average thickness = 36 inches. Fair persistance. Mostly strip mined with some deep mining.	
			Lower Kittanning	Lower Kittanning	Average thickness = 58 inches including a 10 inch shale parting. Laterally continuous and extensively deep mined with some strip mining.	
			Clarion	Clarion	Average thickness = 42 inches. Laterally continuous. Deep mined and strip mined.	
		POTTSVILLE	Mercer	Mercer	Very thin and laterally discontinuous. Locally strip mined.	Poorly developed cyclic sequence of shales, siltstones, sandstones, and thin coals. Thickness = 50 feet
	MISSISSIPPIAN		Pocono			Fine-grained to conglomeratic sandstone up to 350 feet thick

VERTICAL SCALE

50 0 100 Feet

STRATIGRAPHIC COLUMN OF SURFACE ROCK

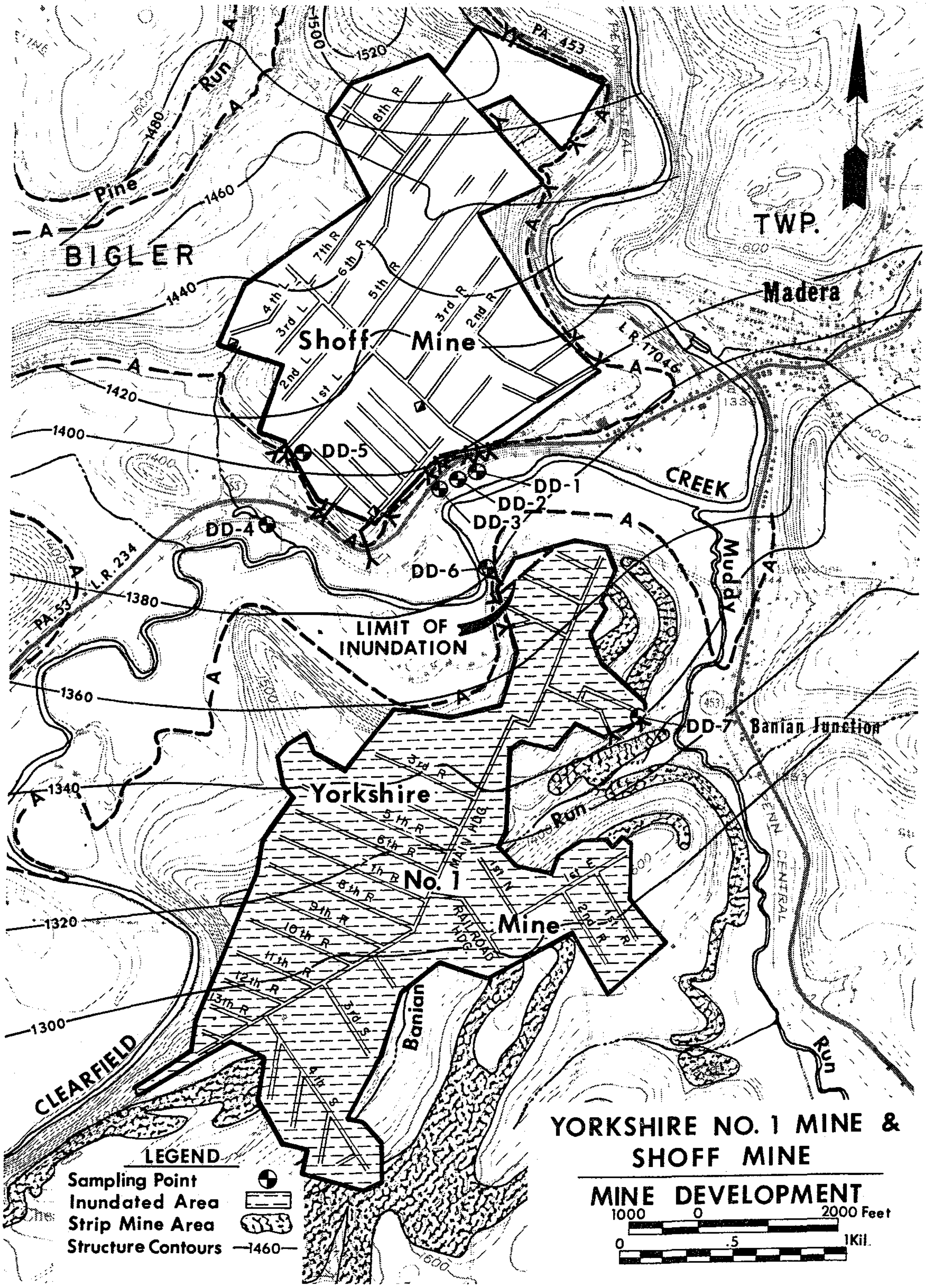
Figure 3

Geology of this portion of central Pennsylvania has been formally mapped by Mr. William Edmunds, Chief Coal Geologist for the Pennsylvania State Geological Survey. Mr. Edmund's mapping shows the Shoff and Yorkshire Mines are structurally situated on the southeast limb of the northeast-southwest trending Laurel Hill Anticline. The Shoff workings are centered only 1.6 kilometers (one mile) southeast of the anticlinal axis, and the Yorkshire workings are only 0.8 kilometer (one-half mile) farther away. As a result, strata in the vicinity of both mines dip south or southeast at about one degree, as shown by the structure contours in the Figure 2 mapping. There are numerous large faults just north of the abandoned mines, along the anticlinal axis. However, no major faults are known to exist in the immediate vicinity of either mine.

MINING HISTORY

Shoff Mine

The Clarion coal seam at the Shoff Mine site was mined at different times by both underground and surface techniques between the late 19th century and the early 1930's. During the 1940's, approximately 460 meters (1500 feet) of "A" coal outcrop adjacent to the workings, and 610 meters (2000 feet) of the overlying "B" coal outcrop east and southeast of the mine were stripped. In addition, several small "D" and "E" coal strip cuts were made above the Shoff Mine complex. Some of this stripping has since been backfilled, regraded and revegetated, and now supports well established plant growth. There are currently no active mines in the vicinity of the Shoff Mine. All mining information indicates the Shoff Mine was developed to the rise to facilitate gravity drainage from workings and to take advantage of rail facilities located adjacent to the creek. Although most entries have been destroyed by road building activities along Route 53, review of available mine maps show that there were 17 drift entries driven from the south and east. The southernmost drifts were primarily utilized for haulage of coal from the mine to the tipple and rail sidings along Clearfield Creek. Other entries provided access for personnel and supplies. A generalized view of the workings and mine development of the Shoff Mine complex is shown in Figure 4.



YORKSHIRE NO. 1 MINE &
SHOFF MINE
MINE DEVELOPMENT
1000 0 2000 Feet
0 .5 1Kil.
LEGEND
Sampling Point
Inundated Area
Strip Mine Area
Structure Contours 1460
Shoff Mine
Yorkshire No. 1 Mine
BIGLER
TWP.
Madera
CREEK
Muddy Run
Pine Run
Banian Run
Banian Junction
CLEARFIELD
LIMIT OF INUNDATION
DD-1
DD-2
DD-3
DD-4
DD-5
DD-6
DD-7
PA. 453
L.R. 17046
L.R. 234
PA. 53
MAIN HDG
RAILROAD HDG

Figure 4

The Shoff Mine consists of several blocks of interconnected workings which were apparently first mined independently from one another. The southernmost block, originally known as the Gatehouse Mine, measures approximately 460 by 550 meters (1500 by 1800 feet), with access through six drift entries. The extreme southern drift in the Shoff complex was the original point of entry to this block of workings. From this point, a main heading was driven 460 meters (1500 feet) to the northeast, roughly paralleling the coal outcrop in that area. Five headings were then driven to the left, or northwest, from that main heading, permitting extensive coal extraction.

An interconnected northeastern block of coal was also worked independently by the Greenwood Mine. There are three drift entries to the Greenwood Mine workings, but no discharge has been observed at any of these. It is suspected the workings are partially inundated and draining to the northern end of the Shoff Mine.

Most of the Shoff Mine is comprised of the remaining central block of coal. The mine's main heading, which is now evident only as the primary drainage discharge point (DD-1) extends approximately 1160 meters (3800 feet) northwest, slightly west of the maximum dip. From this main heading, four left headings were driven down-dip southwest. Beyond the workings of the previously mentioned Gatehouse Mine, about 550 meters (1800 feet) in from the drift entry, the first of these headings bore to the left. After breaking through the coal outcrop, the first of these left headings was utilized as an entryway. This was the westernmost drift in the Shoff Mine. The barrier between the ends of the three remaining down-dip headings and the coal outcrop apparently ranges from 15 to 150 meters (50 to 500 feet). Large barrier size here probably reflects a property or mineral rights ownership boundary, since such blocks of coal are not usually left unmined.

Although these 2nd, 3rd and 4th left headings were driven toward the dip, they did not break through to the surface, and it is not likely they are currently inundated. Rooms driven from these headings interconnect with one another, allowing drainage southwest to the first left heading. This heading is presently a significant source of acid mine drainage.

Seven right headings were driven to the rise from the Shoff Mine's main heading. These northeast-trending headings have a total combined length of about 6.5 kilometers (4 miles). Due to their orientation, they are probably not flooded. These up-dip headings were driven on 150 meter (500 foot) centers, while rooms or panels were driven on 15 meter (50 foot) centers and range in length from 30 to 120 meters (100 to 400 feet). Maximum recovery, or full pillar extraction, was practiced in the southern portions of the mine. The lowest recovery rate, about 35%, occurred beneath a maximum of 90 meters (300 feet) cover near the center of the mine. Shoff headings and their associated workings are extensively interconnected, and combined are the source of mine drainage emanating from the main entry.

Yorkshire No. 1 Mine

General mine development of the Yorkshire No. 1 Mine is illustrated in Figure 4, Page 15. Operation of this mine occurred during approximately the same time period as the Shoff Mine. Portions of mine mapping for this complex were dated by the original mining engineers, and indicate that mining began during or shortly before World War I. During early years of mine development, from about 1916 to 1920, the main heading was driven approximately 1830 meters (6000 feet) southwest, or down-dip. Eight right headings totalling roughly 4.4 kilometers (2 3/4 miles) were driven northeast on 90 meter (300 foot) centers. These right headings were driven to rise from the main heading with an average grade of 1.5 percent. As in the Shoff Mine, panels from these headings were driven on 15 meter (50 foot) centers, and there are indications of only local full pillar extraction. Available mapping suggests drift and slope entries to the Yorkshire workings were developed very early in the mine's history. Coal was transported from the mine through slope entries to a tipple and rail facilities adjacent to Banian Run. The drift entries were probably used for personnel and equipment access.

Between 1921 and 1929, the mine's main heading was extended an additional 1460 meters (4800 feet), but was angled 30 degrees west from the original heading. Along this new portion of heading, six additional right headings - the 9th through 14th - with a combined length of 1.9 kilometers (1.2 miles) were driven. In this newer portion of the mine, the Clarion coal rolls gently, but the general direction of dip is still to the south.

Southeastern portions of the mine were last to be developed, between 1929 and 1941. Four south headings were advanced down-dip off the main heading. The 4th South heading was over eight-tenths kilometer (one-half mile) in length, and opened a large block of coal. A second block of coal beneath and southeast of Banian Run was also mined at this time. Such mining under the stream was limited to areas in which cover exceeded 21 meters (70 feet), thereby reducing possibility of excessive groundwater infiltration.

MINE DRAINAGE AND WATER QUALITY

One of the primary goals of this study was determination of effectiveness of mine flooding as a method to reduce or eliminate production of acid mine drainage in abandoned underground coal mines. The general theory behind this pollution reduction technique is discussed in the introduction to this report. However, the only accurate way to determine validity of this theory is to evaluate quality of effluent from a mine which employed down-dip mining as compared to water quality from an unflooded, up-dip mine's discharge. This was achieved by monitoring flow and quality of all mine drainage discharges from two selected abandoned mines - the Shoff Mine, unflooded with its mine workings above the points of discharge (up-dip), and the inundated Yorkshire Mine with its development to dip below the points of discharge. This water quality data, augmented by all available information on development of the abandoned mines and local structure of the coal, provided an excellent picture of the mines' drainage patterns and effects of those patterns on formation of acid mine drainage.

Historical monitoring data, as well as current sampling results, were utilized in this study to provide a comprehensive water quality data base. Historical data was obtained for both Shoff and Yorkshire Mines from previous sampling programs conducted during studies of the Clearfield Creek and Muddy Run watersheds. Grab type sampling was commonly used in both programs, and samples were analyzed for pH, hot acidity, total iron and sulfates. Every Shoff Mine discharge point was located and sampled during the Clearfield Creek study. Unfortunately, the main discharge from the Yorkshire Mine was not sampled in either study, although the small discharge from the slope entry was. The purpose of these watershed studies was elimination or reduction of acid emanating from various sources, including deep mines. Mine drainage with little or no acidity and relatively high pH, 5.0 or greater, was not monitored consistently or considered a major source of pollution. The historical water quality data obtained during those sampling programs is presented in Table 1.

For this study, monitoring stations were established at all discharge points for each mine, five for the Shoff Mine and two for the Yorkshire Mine. These sampling points are identified in Figure 2, Page 12. Eight semi-monthly sample runs were conducted for collection of discharge samples and metering flows. All water samples were analyzed for: pH, acidity, alkalinity, sulfates, ferric iron, ferrous iron, specific conductance, manganese, aluminum and calcium. Data obtained during the current sampling program is summarized in Table 2. Chemical analyses of samples taken early in the study period were examined closely to evaluate noticeable differences between field and laboratory pHs. This also raised questions as to the validity of other analysis data. Inconsistencies were corrected through comparisons with other quality control analyses made by a separate laboratory. The laboratory providing the most dependable work was retained for all further water quality analyses.

A comparison of quality of major discharges from the abandoned mines provides helpful insights into the hydrogeologic conditions controlling formation of acid in those mines. Acid concentrations in the Shoff Mine's main discharge were consistently in excess of 2200 mg/l, while those of the discharge from the Yorkshire Mine were generally less than 100 mg/l. Similar differences were noted in other mine drainage indicators. Total iron concentrations ranged from 600 to 800 mg/l and sulfates averaged 2000 mg/l in the Shoff Mine, while concentrations of 55 mg/l of iron and 600 mg/l of sulfate were observed in Yorkshire Mine discharges. Manganese and aluminum concentrations were also substantially higher in the Shoff Mine. Total pollutant loads for the Shoff and Yorkshire Mines were computed combining all discharges from each mine. These combined loadings, which were presented in Table 3, confirm the historical water quality data, and are in agreement with water quality analyses reflected in the main discharge samples.

Table 1
HISTORICAL WATER QUALITY DATA
SHOFF AND YORKSHIRE NO. 1 MINES

Discharge Number	Date	Lab pH	Flow cu.m./min. (cu.ft./sec)	Net Acidity mg/l	Net Acidity kg/day (lbs/day)	Total Iron mg/l	Total Iron kg/day (lbs/day)	Sulfates mg/l	Sulfates kg/day (lbs/day)
DD-1	9/14/72	2.9	0.66 (1.45)	2870	2737 (6030)	999	952 (2097)	1970	1878 (4137)
	10/20/72	2.6	0.63 (1.39)	3200	2895 (6378)	200	181 (399)	2300	2081 (4584)
	11/01/72	3.1	0.24 (0.53)	2690	921 (2029)	93	342 (753)	2500	856 (1886)
DD-2	10/02/72	3.3	0.03 (0.06)	1510	74 (163)	278	14 (31)	2460	120 (264)
	11/01/72	2.7	0.09 (0.20)	1580	193 (425)	288	35 (77)	1940	237 (522)
DD-3	10/02/72	3.4	0.10 (0.22)	2200	323 (712)	338	50 (110)	4000	587 (1293)
	10/20/72	2.4	0.17 (0.38)	1820	445 (930)	172	42 (93)	2800	685 (1509)
	11/01/72	2.9	0.09 (0.20)	1570	192 (423)	744	91 (200)	2500	306 (674)
DD-4	9/14/72	2.8	0.10 (0.22)	1320	194 (427)	376	55 (121)	1030	151 (333)
	10/20/72	2.6	0.03 (0.06)	1500	73 (161)	354	17 (37)	1450	71 (156)
	11/01/72	3.1	0.12 (0.26)	1390	238 (524)	328	56 (123)	2350	402 (896)
DD-5	10/02/72	3.6	0.03 (0.06)	7600	372 (820)	173	85 (187)	1330	65 (143)
	10/20/72	2.7	0.08 (0.18)	340	41 (90)	84	10 (22)	320	39 (86)
DD-7	7/30/70	3.7	0.02 (0.04)	62	2 (4)	0.7	0.0	470	11 (24)
	8/31/70	3.7	0.07 (0.15)	56	5 (11)	0.4	0.0	298	29 (64)
	9/30/70	3.7	0.03 (0.06)	80	4 (9)	0.5	0.0	470	23 (51)
	10/29/70	3.7	0.17 (0.38)	107	26 (57)	0.3	0.1 (0.22)	461	113 (249)
	11/29/70	3.9	0.34 (0.75)	100	49 (108)	0.3	0.1 (0.22)	509	249 (549)
	12/28/70	4.0	0.78 (1.72)	60	67 (148)	.0	0.0	461	518 (1141)
	1/23/71	4.2	0.17 (0.38)	102	25 (55)	.2	0.0	557	136 (300)
	3/05/71	3.9	1.05 (2.31)	66	100 (220)	0.3	0.4 (0.88)	154	233 (513)
	4/08/71	3.7	0.44 (0.97)	60	38 (84)	0.1	0.1 (0.22)	336	213 (469)
	5/11/71	3.6	0.29 (0.64)	48	20 (44)	0.08	0.0	288	120 (264)

Table 2

ABANDONED MINE WATER QUALITY DATA

SHOFF MINE

SAMPLING STATION DD-I

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	μmhos
7-30-74	0.65	–	2.3	2700	2508	0	0	740	688	728	676	3000	2787	11.4	11	7.4	6.9	8.7	8.1	2,900
8-20-74	0.20	2.8	2.6	2280	669	0	0	787	231	560	164	2100	616	12.7	3.7	7.1	2.1	9.2	2.7	3,150
9-08-74	0.20	–	2.9	2420	710	0	0	1335	391	773	227	3500	1027	17.7	5.2	6.9	2.0	10.3	3.0	4,350
9-24-74	0.36	2.5	2.9	4600	2362	0	0	1033	530	672	345	3750	1926	14.7	7.5	8.1	4.2	9.2	4.7	4,000
10-10-74	0.09	2.6	2.7	2600	318	0	0	691	84	616	75	2450	299	14.5	1.8	7.9	1.0	94.5	1.2	3,050
10-31-74	0.17	2.5	2.5	2560	626	0	0	612	150	392	96	2225	544	14.9	3.6	9.0	2.2	75.7	1.9	2,900
11-27-74	0.34	2.5	2.9	3050	1491	0	0	750	367	507	248	2490	1218	32	16	7.8	3.8	88	4.3	3,275
12-02-74	0.19	2.8	2.7	2700	726	0	0	720	194	463	125	2180	586	100	27	7.8	2.1	89	2.4	3,235

SHOFF MINE

SAMPLING STATION DD-2

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	μmhos
7-30-74	0.10	–	2.3	1700	249	0	0	168	25	7.84	1.2	1925	283	16.3	2.4	10.1	1.5	4.6	0.7	2,700
8-20-74	0.03	2.4	2.4	1408	69	0	0	116	5.7	2.24	0.1	1650	81	14.1	0.7	9.3	0.5	4.2	0.2	4,550
9-08-74	0.09	–	2.7	1460	178	0	0	187	23	5.60	0.7	2100	257	15.9	1.9	11.2	1.4	5.1	0.6	4,450
9-24-74	0.05	2.4	2.7	2500	183	0	0	497	37	0	0	2450	180	19.5	1.4	11.2	0.8	3.9	0.3	3,900
10-10-74	0.03	2.2	2.6	2000	98	0	0	271	13	12.3	0.6	1875	92	23.9	1.2	13.2	0.6	88.5	4.3	3,150
10-31-74	0.02	2.6	2.5	1820	44	0	0	227	5.5	7.84	0.2	2275	56	64.7	1.6	13.2	0.3	59.6	1.5	2,800
11-27-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
12-02-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–

Table 2 (Cont.)

ABANDONED MINE WATER QUALITY DATA

SHOFF MINE

SAMPLING STATION DD-3

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	µmhos
7-30-74	0.01	–	2.5	1200	8.6	0	0	25.1	0.2	0	0	1250	9.2	16.5	0.1	10.4	0.1	6.2	0.04	2050
8-20-74	0.03	3.0	2.6	788	39	0	0	33.8	1.6	5.6	0.3	1375	67	15.5	0.8	11.5	0.6	5.8	0.3	2540
9-08-74	0.02	–	2.9	940	23	0	0	32.9	0.8	1.12	0.03	1575	39	18.3	0.4	11.3	0.3	7.1	0.2	3150
9-24-74	0.03	2.7	2.8	2100	103	0	0	28.9	1.4	2.24	0.1	1650	81	17.1	0.8	10.4	0.5	6.9	0.3	2500
10-10-74	0.02	3.0	2.7	1300	32	0	0	57.7	1.3	2.24	0.05	1675	41	24.7	0.6	15.8	0.4	95.5	2.3	2150
10-31-74	0.02	3.5	2.6	1120	27	0	0	73.5	1.8	4.48	0.1	1725	42	64.6	1.6	14.2	0.3	63.2	1.5	2500
11-27-74	0.03	3.0	2.9	1800	88	0	0	69	3.4	2.0	0.1	1475	72	152	7.4	18.4	0.9	78	3.8	2340
12-02-74	0.03	3.1	2.8	1120	55	0	0	54.4	2.7	1.92	0.1	1350	66	144	7.0	15	0.7	73	3.6	2355

SHOFF MINE

SAMPLING STATION DD-4

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	µmhos
7-30-74	0.01	–	2.3	700	8.6	0	0	12.7	0.2	10.08	0.1	475	5.8	15.2	0.2	3.7	0.05	0.4	0.005	1025
8-20-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
9-08-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
9-24-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
10-10-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
10-31-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
11-27-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
12-02-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–

SHOFF MINE

SAMPLING STATION DD-5

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	µmhos
7-30-74	0.12	–	2.4	900	154	0	0	170	29	39.2	6.7	1050	180	8.9	1.5	4.3	0.7	3.3	0.6	2225
8-20-74	0.05	3.2	2.5	724	53	0	0	149	11	90.7	6.7	1100	81	7.4	0.5	5.1	0.4	2.7	0.2	2975
9-08-74	0.09	–	2.8	340	103	0	0	282	34	94.1	12	1425	174	10.1	1.2	4.5	0.5	2.9	0.4	2950
9-24-74	0.09	2.1	2.7	1600	196	0	0	410	50	80.6	9.8	1500	183	19.1	2.3	4.2	0.5	0.5	0.1	2475
10-10-74	0.02	3.1	2.7	1200	29	0	0	529	13	448	11	1425	35	19.0	0.5	6.7	0.2	50.9	1.2	1800
10-31-74	0.02	2.8	2.6	1400	34	0	0	297	7.3	134	3.3	1700	42	30.5	0.7	7.9	0.2	43.4	1.1	2300
11-27-74	0.03	5.1	2.8	1650	81	0	0	366	18	155	7.6	1600	78	104	5.1	7.5	0.4	50	2.4	2620
12-02-74	0.002	2.8	2.6	1280	3	0	0	332	0.8	89	0.2	1410	34	76	0.2	6.0	0.1	49	0.1	2490

Table 2 (Cont.)
ABANDONED MINE WATER QUALITY DATA

SAMPLING STATION DD-6 — YORKSHIRE NO. I MINE

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	Cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	μmhos
7-30-74	1.00	–	3.7	26	38	0	0	52.8	76	52.64	76	525	758	33.4	48	1.5	2.2	7.2	10	850
8-20-74	0.85	4.9	3.8	16	20	0	0	53.8	66	40.2	49	400	489	29.7	36	2.0	2.4	0.5	10	950
9-08-74	0.66	–	4.8	92	83	10	9.5	59.4	57	56.0	54	575	548	34.0	32	1.7	1.6	7.1	6.3	875
9-24-74	2.96	–	4.1	68	290	0	0	51.2	218	63.8	271	550	2340	33.9	144	1.6	6.8	6.9	29	900
10-10-74	0.46	4.9	4.7	32	21	0	0	59.0	39	58.2	39	500	330	48.8	32	1.9	1.3	0	0	700
10-31-74	4.34	4.9	3.5	80	499	0	0	58.8	367	58.24	363	550	3429	52.6	328	2.0	12	1.3	8.1	800
11-27-74	0.22	5.2	5.7	54	17	6	1.9	55.0	17	53.0	17	455	145	116	37	2.1	0.7	0.6	0.2	925
12-02-74	0.56	5.8	5.6	116	94	22	18	57.2	46	60.0	48	440	355	120	97	1.8	1.5	0.5	0.4	975
12-20-74	1.00	–	3.6	70	101	0	0	57.0	82	55.0	79	475	684	117	168	1.9	2.7	0.3	0.4	1,010

SAMPLING STATION DD-7 — YORKSHIRE NO. I MINE

DATE	FLOW	pH		ACIDITY		ALKALINITY		TOTAL IRON		FERROUS IRON		SULFATES		CALCIUM		MANGANESE		ALUMINUM		SPEC. COND.
	Cu m/min	FIELD	LAB	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC. mg/l	LOAD kg./day	CONC mg/l	LOAD kg./day	μmhos
7-30-74	0.01	–	3.2	52	0.4	0	0	1.4	0.01	0	0	300	2.2	17.4	0.1	3.4	0.02	5.1	0.04	600
8-20-74	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
9-08-74	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
9-24-74	0.01	4.4	3.7	72	0.9	0	0	0.3	0.01	0	0	525	6.4	15.3	0.2	2.9	0.04	4.9	0.06	725
10-10-74	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
10-31-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
11-27-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
12-02-74	Dry	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–

Shoff Mine

Despite higher total flow rates attributed to its slightly greater area, pollutant loadings emanating from the Yorkshire Mine were almost always lower than those from the Shoff Mine. There are several important factors that contribute to this significant water quality difference. A large percentage of Shoff Mine headings and workings were driven to the rise or intercepted by other headings being driven to the rise. As a result, most of the mine is not inundated, but drains steadily by gravity to one of several discharge points. Pyritic coal and fallen roof material is continually exposed to air, groundwater and moisture condensing from the saturated mine atmosphere. Three key factors which affect production of acid mine drainage - accessibility of pyritic material, availability of that material for reaction, and contact time - all play important roles in explaining the relative water quality of the two mines. Since all discharges are located in the southern end of the mine, drainage distance for much of the mine water is quite long. This extends contact time between acid-forming materials, and can substantially increase concentrations of mine drainage pollutants. In addition, roof rock above the workings does not appear to be extremely strong, as evidenced by collapse of all drifts and surface signs of caving that appear along much of the coal outcrop. Continual minor roof collapses within Shoff workings constantly expose fresh, unweathered pyritic materials for reaction. This also substantially increases pollution production within the mine.

Yorkshire No. 1 Mine

Mine mapping and associated geologic and water quality information all suggest that the Yorkshire Mine is largely inundated, with its overflow point being the drift entry discharge adjacent to Clearfield Creek. Assuming this statement is true, it is extremely difficult to explain the source of acid water that periodically discharges from the slope entry. The mouth of the slope is approximately 21 meters (70 feet) above the coal seam at that point, and is 11 meters (35 feet) higher than the drift discharge point. In fact, the slope entry appears to be at a higher elevation than any portion of the Yorkshire Mine workings. It is, therefore, highly unlikely that the drainage emanating from the slope actually originates in any portion of the mine workings.

Field observations and the Mine Development map, Figure 4, offer a feasible explanation for existence of the slope entry discharge. As the map illustrates, the Lower Kittanning "B" seam has been strip mined to a point directly adjacent to the slope entry. This seam and its associated overburden are known acid producers in this area, and dip of the coal is such that all drainage collected in the strippings would be channeled to its southern end, where the slope is located. Available geologic information also indicates that overburden material above the Clarion coal, through which the slope was originally driven, was a relatively unstable roof rock. Since closure of the Yorkshire Mine, the roof of the slope has probably completely collapsed, blocking the slope and preventing extensive passage of water.

Thus, water channeled through the surface strippings could enter and pool in the upper portion of the slope, gradually seeping downward into the mine workings. In wet weather periods, when water pools in the slope faster than it can seep downward, the discharge appears. Acidity in this water could result from contact with pyritic materials in any of several places - Clarion coal overburden in the slope, Lower Kittanning overburden in adjacent strippings, and refuse from Yorkshire workings, which is spread throughout the tipple area just below the slope. This explanation could account for existence of what appears to be a mine discharge at an elevation far above the predicted mine pool level.

Assuming the above explanation is valid and that there is only one actual point of discharge from the Yorkshire Mine workings, hydrology and mine drainage in the mine can be accurately assessed. The Yorkshire Mine's major discharge point is DD-6 (see Figure 2), a drainage pipe from the mine's collapsed drift entry. As previously mentioned, water quality in this mine is significantly better than that observed in the Shoff Mine, although there is a marginal acid production problem. The improved water quality is largely due to the inundated condition of most of the mine, which isolates pollution forming materials from oxygen required in the acid formation process. Neutralization and aeration of some acid may also be occurring fairly close to the point of discharge, as suggested by large volumes of ferric hydroxide (yellow boy) which intermittently discharge in "slugs."

There are several possible explanations for production of the acid that slightly degrades the Yorkshire Mine water. First, and probably most important, is the fact that the northernmost tip of the mine complex is up-dip from the drift entry, and is therefore not inundated. Thus, in this portion of the mine, pyritic materials are exposed to air and moisture, and acid mine drainage can form. Since the unflooded area is relatively small and contact times are short, relatively little acid forms. Some of this is neutralized, as previously mentioned, near the mine opening. However, since acid production occurs fairly close to the discharge point, there may be insufficient time for neutralization of all acid prior to discharge. A second factor is the small amount of dissolved oxygen trapped in infiltrating groundwater which would be available for reaction with acid-producing materials. Additional acid waters could be infiltrating downward from surface mines in four of the Allegheny Group coal seams that overlie Yorkshire workings. Several of these and their associated overburden materials are known acid mine drainage producers in this portion of Pennsylvania.

In summary, water quality data presented in Tables 1 and 2 clearly shows that there is substantial difference in water quality of the Shoff and Yorkshire Mines. The Shoff, with its up-dip development and unflooded workings, is a major source of acid mine drainage. Discharge quality of the Yorkshire Mine ranges from marginal to slightly acid, and reflects the hydrologic effectiveness of inundation as a deterrent to formation of acid mine drainage.

Table 3
SUMMARY

ABANDONED MINE SITE CHARACTERISTICS

	Yorkshire No. 1 Mine		Shoff Mine	
Direction of Operation	To Dip		To Rise	
Coal Seam	Clarion "A" coal		Clarion "A" coal	
Percent Dip	10 South or Southeast		10 South or Southeast	
Period Mined	1900 to 1942		Late 1800's to early 1930's	
Area Mined	220 hectares (540 acres)		170 hectares (428 acres)	
Percent Extraction	50%		Variable 35% to 100%	
Percent Sulfur in Coal	3% approximate		3% approximate	
Percent of Workings Inundated	90%		Less than 10%	
Maximum Thickness of Overburden	90 meters (300 feet)		90 meters (300 feet)	
Number of Discharges	2		5	
Total Flow (cu.m/min.)				
Maximum - Date	4.34	10/31/74	0.89	7/30/74
Minimum - Date	0.22	11/27/74	0.16	10/10/74
Average	1.34		0.39	
Total Acidity Load (kg/day)				
Maximum - Date	499	10/31/74	2,928	7/30/74
Minimum - Date	17	11/27/74	477	10/10/74
Average	129		1,408	
Total Iron Load (kg/day)				
Maximum - Date	367	10/31/74	742	7/30/74
Minimum - Date	17	11/27/74	111	10/10/74
Average	107		365	
Ferrous Iron Load (kg/day)				
Maximum - Date	363	10/31/74	684	7/30/74
Minimum - Date	17	11/27/74	86	10/10/74
Average	110		252	
Sulfate Load (kg/day)				
Maximum - Date	3,429	10/31/74	3,265	7/30/74
Minimum - Date	145	11/27/74	467	10/10/74
Average	1,009		1,398	
Aluminum Load (kg/day)				
Maximum - Date	29	9/24/74	10.5	11/27/74
Minimum - Date	0	10/10/74	3.4	8/20/74
Average	7.2		6.8	

SECTION V

ACTIVE MINE SITE EVALUATION

MINE LOCATION

Lady Jane Collieries, Incorporated, operates the Stott No. 1 Mine in Huston Township, Clearfield County, Pennsylvania, about 1.6 kilometers (one mile) south of Penfield (see Figure 5). The mine's workings lie just east of Bennett Branch Sinnemahoning Creek and two of its tributaries, Moose Run and Horning Run, beneath approximately 6.5 square kilometers (2.5 square miles) of Moshannon State Forest. Field offices, drift entry, tipple, treatment facilities and discharge point for this operation all lie just northeast of Moose Run, in the southwestern corner of the mine.

From the drift entry, the main heading extends nearly 2300 meters (7600 feet) to the northeast. The Second North heading then extends from the main heading to the northwest for 2600 meters (8600 feet), and actually serves as a main heading for the entire northwest portion of the mine. Several other drifts also intersect the mine workings, but are maintained only for ventilation and emergency escape. In addition, there is a shaft entry to the workings, located north of Horning Run along the 2nd North heading. This 60 meter (190 foot) shaft serves as an entryway for mine personnel and supplies.

Specific location, configuration, mine development and general geology of the Stott No. 1 Mine are shown in Figure 5.

GEOLOGY

The Stott No. 1 Mine is operated in the Lower Kittanning "B" coal, which is part of the Pennsylvanian age Allegheny Group. As the stratigraphic column in Figure 3 shows, rocks overlying the "B" seam are predominantly sandstones and shales, with two thin, unmined coal seams 21 and 51 meters (68 and 166 feet) above. The "B" coal ranges from 81 to 122 centimeters (32 to 48 inches) thick, averaging 94 centimeters (37 inches). Analytical values for Lower

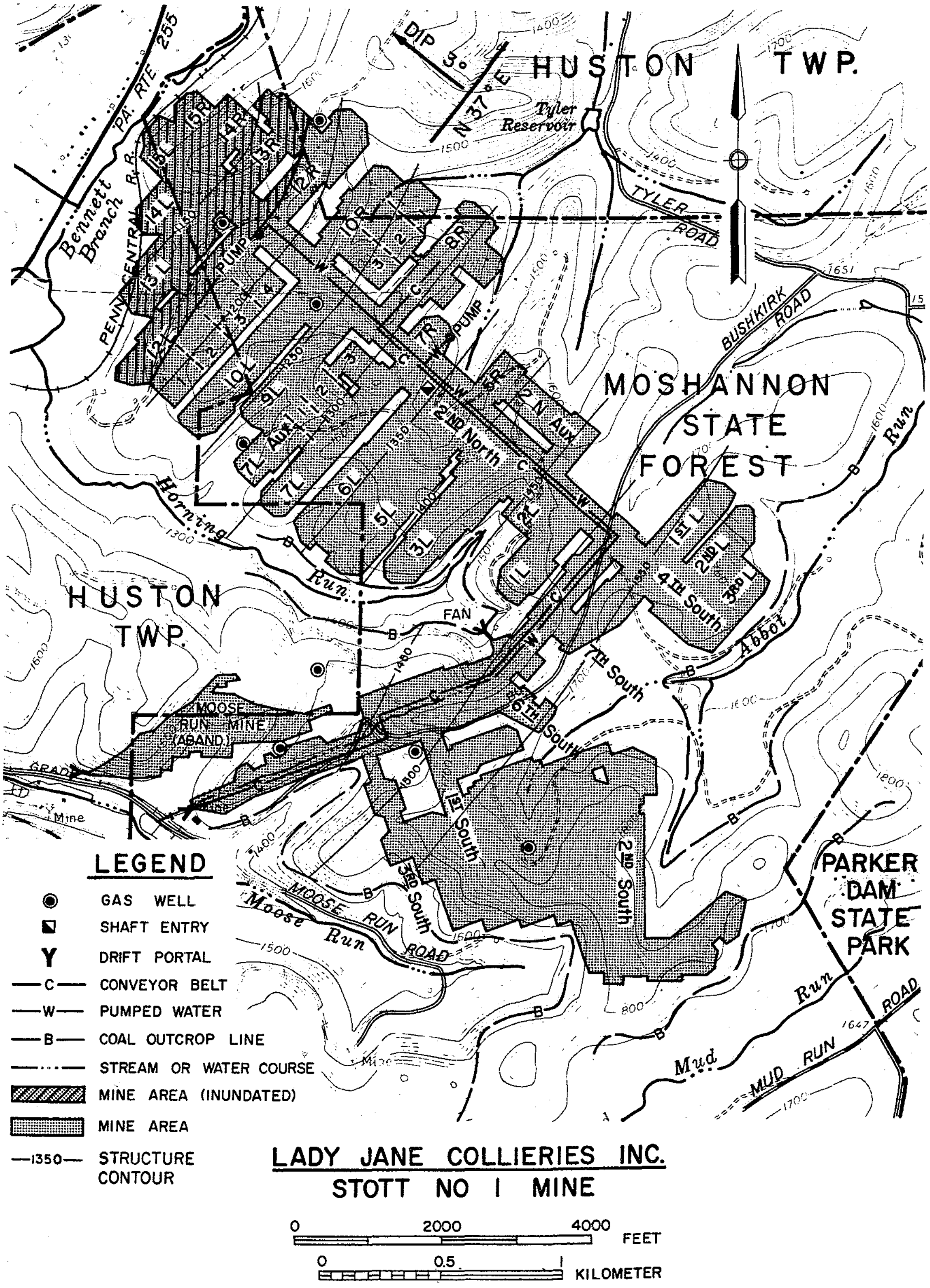

HUSTON TWP.
DIP 3° N 37° E
Tyler Reservoir
TYLER ROAD
BUSHKIRK ROAD
MOSHANNON STATE FOREST
PA. RTE 255
Bennett Branch
PENN CENTRAL R.R.
Horning Run
HUSTON TWP.
MOOSE RUN MINE (ABAND.)
Moose Run
MOOSE RUN ROAD
Abbot Run
PARKER DAM STATE PARK
Mud Run
MUD RUN ROAD
PUMP
FAN
2ND North
2 N Aux
7L Aux
4TH South
7TH South
6TH South
1ST South
2ND South
3RD South
LEGEND
GAS WELL
SHAFT ENTRY
DRIFT PORTAL
C CONVEYOR BELT
W PUMPED WATER
B COAL OUTCROP LINE
STREAM OR WATER COURSE
MINE AREA (INUNDATED)
MINE AREA
1350 STRUCTURE CONTOUR
LADY JANE COLLIERIES INC.
STOTT NO I MINE
0 2000 4000 FEET
0 0.5 1 KILOMETER

Figure 5

Kittanning coal produced by the Stott Mine average, on an as received basis, 3308 Kilogram calories, three percent total sulfur, 12 percent ash and 3.8 percent moisture.

In this area, the Lower Kittanning coal is structurally situated on the northern flank of the Chestnut Ridge Anticline. Strata here generally strike about 37 degrees east of North and dip northwest about 3 degrees. No major faults have been identified in the vicinity of the mine, although minor structural irregularities have been encountered during mining.

Roof rock in the mine consists of dark shales and sandy shales. Relatively few fractures are present and good roof conditions exist throughout most of the mine. The mine floor, or bottom is comprised of a relatively soft underclay which, when wet, softens and presents problems for mobile equipment.

MINE DEVELOPMENT

Stott No. 1 Mine began production in the early 1950's when its drift entry adjacent to Moose Run was driven into the Lower Kittanning coal. This drift entry has a surface elevation of 422 meters (1384 feet) above sea level and is located roughly 365 meters (1200 feet) southeast of the entry to the abandoned Moose Run Mine. From the drift entry, the Stott Mine was advanced to rise, but not directly perpendicular to the maximum dip angle, in a northeasterly direction for about 1293 meters (4240 feet). The coal seam rose approximately 40 meters (130 feet) in that distance, to an elevation of 461 meters (1514 feet) above sea level. Several south headings were driven directly up-dip, perpendicular to this main heading, and the southeastern portion of the mine was subsequently worked out.

The second phase of mining in the Stott No. 1 Mine extended the main heading about 730 meters (2400 feet) northeast, along the strike of the coal seam. Panels driven from the north side of this portion of the main heading were, therefore, down-dip, while panels driven south were up-dip. While panels in this northeastern portion of the mine were being worked, 2nd North heading, which serves as the main heading for the entire northwestern segment of the mine, was driven down-dip northwest. This heading eventually attained a length of

2560 meters (8400 feet) and dropped 125 meters (413 feet) along its length.

Original mining plans called for 15 left and right headings driven parallel to strike from the 2nd North heading. Most of the left headings were eventually driven, and large panels of coal were worked out along the headings. In addition, it was possible in some areas to mine as many as three butts, or smaller blocks of rooms and pillars, from the sides of the panels, thereby increasing their size. Since headings were driven along strike, differences in coal elevation only range from 1.2 to 2.4 meters (4 to 8 feet) although heading lengths average 850 meters (2800 feet). However, differences in elevation across individual panels, perpendicular to left and right headings, are as great as 9.1 meters (30 feet).

Although 15 right headings from 2nd North were anticipated during early mine development, several were never driven. Instead, large butts were extended from existing mined out panels to permit maximum extraction of coal where no headings were to be driven. A recent addition to the mine, located northeast or right of 2nd North heading, is an auxiliary 2nd North heading, which was driven from 5th Right heading. Coal adjacent to this auxiliary heading will be mined at some future date.

MINING AND PRODUCTION

Mining Technique

As the Stott No. 1 Mine mapping in Figure 5 shows, current mining activities are restricted to the 8th Right panel from 2nd North heading. Conventional room-and-pillar mining methods are used, and coal is transported from the mine by a conveyor belt system. A working coal face averages about 6 meters (20 feet) in width. This basic sequence of operations is employed to extract and transport coal from the active mining area:

1) The roof bolter drills and bolts the roof with bolts of 0.9 to 1.2 meter (3 to 4 foot) lengths. Eight bolts are normally required for each cut of coal removed.

2) After roof control is provided, a face drill is used to drill four 2.4 meter (8 foot) deep holes. The holes are equally spaced across the 6 meter wide working face.

3) A cutting machine is used to undercut the coal face. The cutting machine has a 2.7 meter (9 foot) cutting bar which permits undercutting as much as 2.4 meters (8 feet) of coal.

4) After undercutting, the four holes are loaded for multiple shooting. Two inner holes are fired first with the same time delay. Shortly afterward, outer holes are fired, using a different delay. The central blast increases available space in the middle of the face and provides an opening for coal to be thrown into when outer holes are fired.

5) A loading machine loads coal into shuttle cars. In the Stott No. 1 Mine, approximately 16 to 18 metric tons (18 to 20 short tons) are loaded from a single cut.

6) Shuttle cars transport coal to a conveyor or belt feeder. These cars are 67 meters (22 feet) long and haul 1.8 metric tons (2 short tons) of raw coal under normal mine conditions. Eight to ten shuttle car loads are required to transfer all the coal from coal face to belt feeder. Haulage from face to belt feeder requires approximately 30 seconds.

7) The belt feeder loads a shuttle car load of coal onto the belt in about 90 seconds. Coal is then transported out of the mine to the tipple via conveyor belt system.

Total conveyor haulage distance through the Stott No. 1 Mine is roughly 3360 meters (11,000 feet) or over 1.8 kilometers (two miles). Portions of the belt which convey coal from working face through 8th Right panel to main haulageway – the 2nd North heading – can be added or removed in 36 meter (120 foot) increments as mining advances or retreats. The belt is thus always relatively close to the working face. The main belt then extends 1280 meters (4200 feet) through 2nd North heading to the mine's main drift heading, and from there an additional 2070 meters (6800 feet) through the drift entry to the tipple.

Production

Seventy-four people are currently employed (above and below ground) at the Stott No. 1 Mine. Three shifts are operated - two production shifts and one maintenance shift. In 1973, production from this mine totalled 235,000 metric tons (260,000 short tons), and averaged 190 to 209 metric tons (210 to 230 short tons) of raw coal per production shift. To obtain this production level in a shift, ten to twelve cuts of coal must be blasted and loaded as previously described.

The 8th Right panel, which is currently being mined, trends parallel to strike. Therefore, rooms developed on the left side of the panel are down-dip while those on the right are up-dip. Two production units are currently in operation, and since both advance and retreat mining are practiced, each unit periodically operates both to rise and to dip. For example, as one unit advances to the right from 8th Right heading, it mines to the rise. Then, as that unit retreat mines, it is actually mining to the dip.

In order to characterize production achieved by these two units, as related to their mode of mining (up-dip versus down-dip), production records were obtained for 1973. This was an average year in terms of mine operation and production. These records, compiled by the Mine Superintendent, contain valuable summaries of production from each of the two units. The following information is presented: tons of raw coal mined per month, tons of raw coal per man-shift (both production and maintenance shifts are considered), tons of clean coal for each production unit per month, monthly activities of each production unit, mining conditions encountered, equipment problems, coal thickness or quality variations, and water conditions. Flow data representing monthly averages of daily weir flow measurements at the settling pond effluent point are also included. This 1973 production record is presented in Table 4.

Additional production records are also presented for 1967 to characterize the Stott No. 1 Mine's activities prior to passage of the Health and Safety Act of 1969. As a result of tighter mining restrictions, coal production generally decreased after passage of the Act. Data presented for 1967 - 68 production includes most of the same parameters already described for the 1973 production log. Production by the two individual units, however, is not presented separately.

Table 4

STOTT NO. 1 MINE – 1973 PRODUCTION RECORD

MONTH	RAW COAL PRODUCTION METRIC TONS (SHORT TONS)	RAW COAL PRODUCTION (METRIC TONS / MANSHIFT) (SHORT TONS / MANSHIFT)	CLEAN COAL PRODUCTION METRIC TONS (SHORT TONS) UNIT NO. 1	CLEAN COAL PRODUCTION METRIC TONS (SHORT TONS) UNIT NO. 2	PRODUCTION DAYS	AVERAGE MONTHLY FLOW m^3/min (cfs)
January	23,712 (26,143)	13.7 (15.2)	9,816 (10,822)	9,053 (9,981)	22.0	1.934 (1.138)
February	19,045 (20,998)	11.2 (12.4)	9,439 (10,407)	8,277 (9,126)	20.0	1.735 (1.021)
March	19,947 (21,992)	11.2 (12.3)	9,602 (10,586)	10,220 (11,268)	21.7	1.489 (0.876)
April	21,012 (23,167)	13.6 (15.0)	9,028 (9,954)	10,454 (11,526)	21.0	1.671 (0.983)
May	21,591 (23,805)	13.2 (14.5)	10,965 (12,089)	9,739 (10,738)	21.5	1.799 (1.058)
June	23,898 (26,348)	15.9 (17.5)	10,352 (11,414)	11,730 (12,933)	20.0	1.617 (0.951)
July	15,806 (17,427)	11.7 (12.9)	7,222 (7,963)	7,481 (8,248)	14.0	1.016 (0.598)
August	21,733 (23,961)	13.7 (15.1)	9,651 (10,641)	10,565 (11,648)	--	--
September	20,794 (22,926)	14.7 (16.2)	8,552 (9,429)	10,884 (12,000)	19.0	0.840 (0.494)
October	25,324 (27,921)	15.1 (16.6)	11,502 (12,681)	13,120 (14,465)	23.0	0.373 (0.219)
November	20,117 (22,180)	13.9 (15.3)	8,955 (9,873)	10,513 (11,591)	19.0	1.610 (0.947)
December	20,844 (22,987)	14.7 (16.2)	9,782 (10,785)	9,519 (10,495)	19,0	2.528 (1.487)

Table 4

STOTT NO.1 MINE-1973 PRODUCTION RECORD

PROGRESS REPORT		COMMENTS
UNIT NO. 1	UNIT NO. 2	
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	Unit #1 encountered some water
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	New roof bolter put into service
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	
Completed advancing to dip, retreated 800 feet to rise. 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	
Advancing to dip 7th L. Panel off 2nd North Heading	Advanced to rise, then retreated to dip 10th R. Panel off 2nd North Heading	
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	Flow reduced due to sludge pumping
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise, then retreated to dip 10th R. Panel off 2nd North Heading	No flow due to sludge pumping
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	Unit #1 encountered wet floor
Advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise 10th R. Panel off 2nd North Heading	Low flow due to sludge pumping
Retreat to dip, then advanced to dip 7th L. Panel off 2nd North Heading	Advanced to rise, then retreated to dip 10th R. Panel off 2nd North Heading	
Advanced to dip 7th L. Panel off 2nd North Heading	Retreated to dip 10th R. Panel off 2nd North Heading	

This 1967 - 68 production data is included in Table 5. An in-depth evaluation of this data is included in Section VI.

MINE DRAINAGE AND WATER QUALITY

Water is removed from the Stott No. 1 Mine workings by a combination of pumping and natural gravity flow. As the mapping in Figure 5 shows, the entire northern portion of the mine has been developed to the dip. Therefore, lowest portions of the mine, which have already been worked out, have been allowed to flood to an elevation of 360 meters (1180 feet) above sea level. The mine pool must be maintained at this elevation to prevent interference with current mining activities. Pool level is controlled primarily by two 50 kilowatt (75 horsepower) pumps, each with a capacity of 32 liters per second (500 gallons per minute). Location of the pumps is also shown in Figure 5. One is located at the mine pool itself while the other is located in a sump farther up-dip, where it removes drainage from the mine before it can enter the mine pool.

Mine drainage is then pumped up 2nd North heading through a 20 centimeter (8 inch) pipe, and then for roughly 760 meters (2500 feet) along strike in the main heading to a point near 6th South heading. This drainage then flows by gravity to the main entry. In addition to the pumped drainage, all other mine workings up-dip from the main entry, including the large southern section of the mine, also discharge by gravity.

All water discharged from the mine is treated before it reaches Moose Run. Discharge flow rates range from 26 to 31 liters per second (417 to 486 gallons per minute), or about 1.7 cubic meters per second (one cubic foot per second). During and immediately after sludge is pumped from the treatment facility's settling basins, flows are substantially lower, ranging from four to eight liters per second (64 to 128 gallons per minute). This decrease in flow results from the increased holding capacity of settling basins immediately after cleaning.

The mode of formation of the Stott No. 1 Mine's acid drainage is identical to that discussed for the abandoned Shoff Mine. Large worked out portions of the mine accumulate infiltrating groundwater and condensation moisture, and drain by gravity toward lower areas. Gravity drainage permits an extended contact time between the water and the highly pyritic Lower Kittanning coal, roof and floor materials. As a result, extremely acid mine drainage is formed and must be treated to avoid environmental degradation.

The untreated mine drainage pumped from the "B" seam Stott No. 1 Mine is highly acid, and is somewhat similar in quality to water discharging from the previously discussed Shoff Mine. A typical water quality analysis is presented below:

pH	2.45
Total Acidity	2800 mg/l
Total Iron	1009 mg/l
Ferric Iron	742 mg/l
Aluminum	103 mg/l
Sulfate	3700 mg/l

This drainage must be chemically treated in order to comply with Pennsylvania's effluent standards (pH 6.0 to 9.0, net alkalinity, and total iron less than 7.0 mg/l). This is achieved using approximately 7.9 kilograms (17.3 pounds) of hydrated lime per 1000 gallons of drainage. Neutralization is followed by aeration and settling to reduce total iron content of the effluent.

In addition to required treatment of mine effluent, there are control measures employed in the mine to reduce infiltration of groundwater and subsequent acid formation. One such measure is channelization of water that does enter the mine to avoid prolonged contact with pollution forming materials. Channelization is also necessary to maintain suitable, relatively dry working conditions for mine personnel and equipment. This can be effectively achieved by employing small diversion ditches between or along barrier pillars within the mine.

Infiltration control or reduction is augmented in the Stott No. 1 Mine by a number of measures. Under certain geologic conditions, drilling for roof bolts may provide access for groundwater to enter the workings. Where such conditions are encountered in portions of the mine that will be open or used for extended periods, resin type roof

Table 5
STOTT NO. I MINE
1967-68 PRODUCTION RECORD

MONTH / YEAR	RAW COAL PRODUCTION		PRODUCTION DAYS
	METRIC TONS (SHORT TONS)	METRIC TONS/MAN SHIFT (SHORT TONS/MAN SHIFT)	
September 1967	28,960 (31,929)	20.77 (22.9)	21
October 1967	29,861 (32,923)	20.5 (22.6)	22
November 1967	22,059 (24,321)	14.3 (15.8)	19
December 1967	27,306 (30,106)	18.8 (20.7)	22
January 1968	25,269 (27,860)	17.1 (18.9)	20
February 1968	17,191 (18,954)	14.5 (16.0)	17.75
March 1968	23,465 (25,871)	16.3 (18.0)	21
April 1968	24,600 (27,123)	17.1 (18.8)	22
May 1968	25,794 (28,439)	16.8 (18.5)	23
June 1968	24,854 (27,403)	16.1 (17.8)	20
July 1968	26,223 (28,912)	18.7 (20.6)	20
August 1968	16,847 (18,574)	15.4 (17.0)	15

Table 5
STOTT NO. I MINE
1967-68 PRODUCTION RECORD

PROGRESS REPORT		COMMENTS
UNIT NO. I	UNIT NO. 2	
Advanced to rise 3rd L. Panel off 4th South Heading	Advanced to dip 6th L. Panel off 2nd North Heading	
Advanced to rise until Oct. 17, then Retreated to dip 3rd L. Panel off 4th South Heading	Advanced to dip 6th L. Panel off 2nd North Heading	Late Oct., Unit #2 encountered low coal, some water
Retreated to dip 3rd L. Panel off 4th South Heading	Retreated to rise 6th L. Panel off 2nd North Heading	Fire in Unit #1 undercutting saw
Retreated to dip 3rd L. Panel off 4th South Heading	Retreated to rise 6th L. Panel off 2nd North Heading	Unit #1 utilized three reduced production crews
Retreated to dip 3rd L. Panel off 4th South Heading	Retreated to rise 6th L. Panel off 2nd North Heading	Unit #2 had limited working space
Retreated to dip 3rd L. Panel off 4th South Heading	Retreated to rise 6th L. Panel off 2nd North Heading	Limited working space poor mining conditions
Advanced to dip 1st L. Panel off 2nd North Heading	Advanced to dip 5th L. Panel off 2nd North Heading	Limited working space for both units in new locations. Poor roof Galis 4100 face drill introduced to Unit #1
Advanced to dip 1st L. Panel off 2nd North Heading	Advanced to dip 5th L. Panel off 2nd North Heading	Poor roof required additional bolting and timbering
Advanced to dip 1st L. Panel off 2nd North Heading	Advanced to dip 5th L. Panel off 2nd North Heading	Unit #2 encountered low coal (less tons per cut, smaller shuttle loads, poor maneuverability, increased breakdowns)
Retreated to rise 1st L. Panel off 2nd North Heading	Advanced to dip 5th L. Panel off 2nd North Heading	Unit #2 encountered low coal. Belts down for 11.5 hours (refuse blockage at transfer points)
Retreated to rise 1st L. Panel off 2nd North Heading until 7/22, then advanced to dip 2nd L. Panel off 2nd North Heading	Advanced to dip 5th L. Panel off 2nd North Heading	
Advanced to dip 2nd L. Panel off 2nd North Heading	Advanced to dip until 8/20, then retreated to rise 5th L. Panel off 2nd North Heading	300 man-shifts of vacation time this month. Unit #1 encountered wet roof, bad floor. Unit #2 encountered low coal

bolts are utilized. These bolts completely seal the drill hole an prevent subsequent infiltration, whereas conventional, or expand roof bolts do not.

Variation of the mining plan is also utilized as required to avoid zones of major or minor faulting which could permit increased infiltration of groundwater. At the same time, such alterations can avoid poor roof or floor conditions and lower quality or thinner coal, thereby increasing productivity.

SECTION VI

ANALYSIS OF DOWN-DIP MINING TECHNIQUES

WATER QUALITY

A detailed discussion of the water quality evaluation of abandoned up-dip versus down-dip underground mines is presented in Section IV. Water quality data obtained in this study generally confirms preliminary predictions that mining to the dip could be implemented as a pollution control measure. Sample data for the Yorkshire Mine, which was developed to the dip, indicated water quality substantially better than that discharging from the Shoff Mine, which was developed to the rise. The discharge from the down-dip mine was still, however, slightly acid in nature and not of acceptable quality. Inundation of the Yorkshire Mine's workings isolated most pyritic materials in the coal, roof and floor, thereby reducing their potential oxidation to form acid mine drainage.

There are several factors which might explain the confirmed formation of acid in the Yorkshire Mine. A very small portion of the abandoned workings lie above the elevation of the discharge points and are, therefore, not inundated. Groundwater infiltrating into these up-dip workings could oxidize pyrite and form acid. Since the point of acid formation is relatively close to the point of discharge, there may not be sufficient time to completely neutralize the acid prior to discharge, even though the mine waters have that potential. A second source of acid production could be the reaction of dissolved oxygen in the mine water with the pyritic materials. This could yield limited acid formation and low acidity concentrations in the effluent. Still another source of acid could be the abandoned surface mines above the Yorkshire workings. Spoils of the overlying Lower Kittanning "B" seam, in particular, are a major source of acid in this area. Acid could be formed in the strip cuts by surface run-off contact with spoil materials, and could subsequently infiltrate downward into the Yorkshire workings. Thus, some of the water infiltrating into the mine workings could already be slightly acid in nature.

Despite the minimal pollution evident in the Yorkshire Mine water quality, it is many times improved over the quality of any of the Shoff Mine discharges. Since the mines were specifically selected for the many similarities they evidenced, the primary factor controlling water quality can only be attributed to the direction of mine development. Therefore, the water quality benefits to be accrued from utilization of the down-dip mining techniques can be substantial and the technique should be given fullest recognition as a pollution control process.

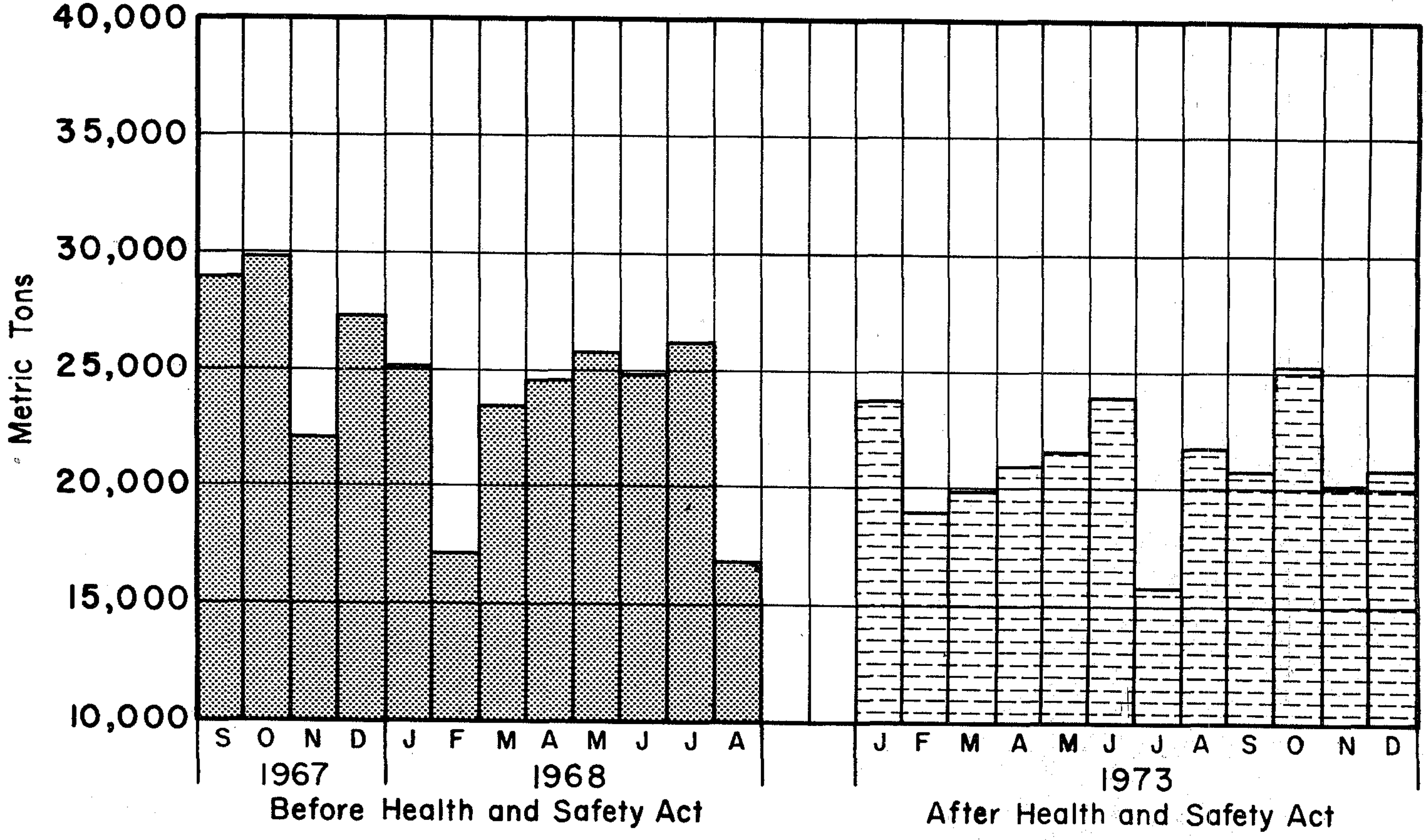
PRODUCTION

Production data obtained for the Stott No. 1 Mine was presented in Tables 4 and 5 and discussed to some extent in Section V. Interpretation and analysis of that information yields important conclusions concerning effects of down-dip mining on coal production. Three charts have been developed from available data in an attempt to visually illustrate production and productivity trends at the Stott Mine as related to up-dip and down-dip operation. It must be stressed that this data was obtained in only one mine and may not be typical of all underground mines.

As previously mentioned, data was obtained for a year's production before and after passage of the Health and Safety Act of 1969. Figure 6 illustrates a decline in production at the Stott Mine resulting from implementation of that Act. Also indicated are wide variations that occur in total monthly production from factors such as: equipment breakdowns, poor roof or floor conditions, excessive water, and variations in coal thickness or quality.

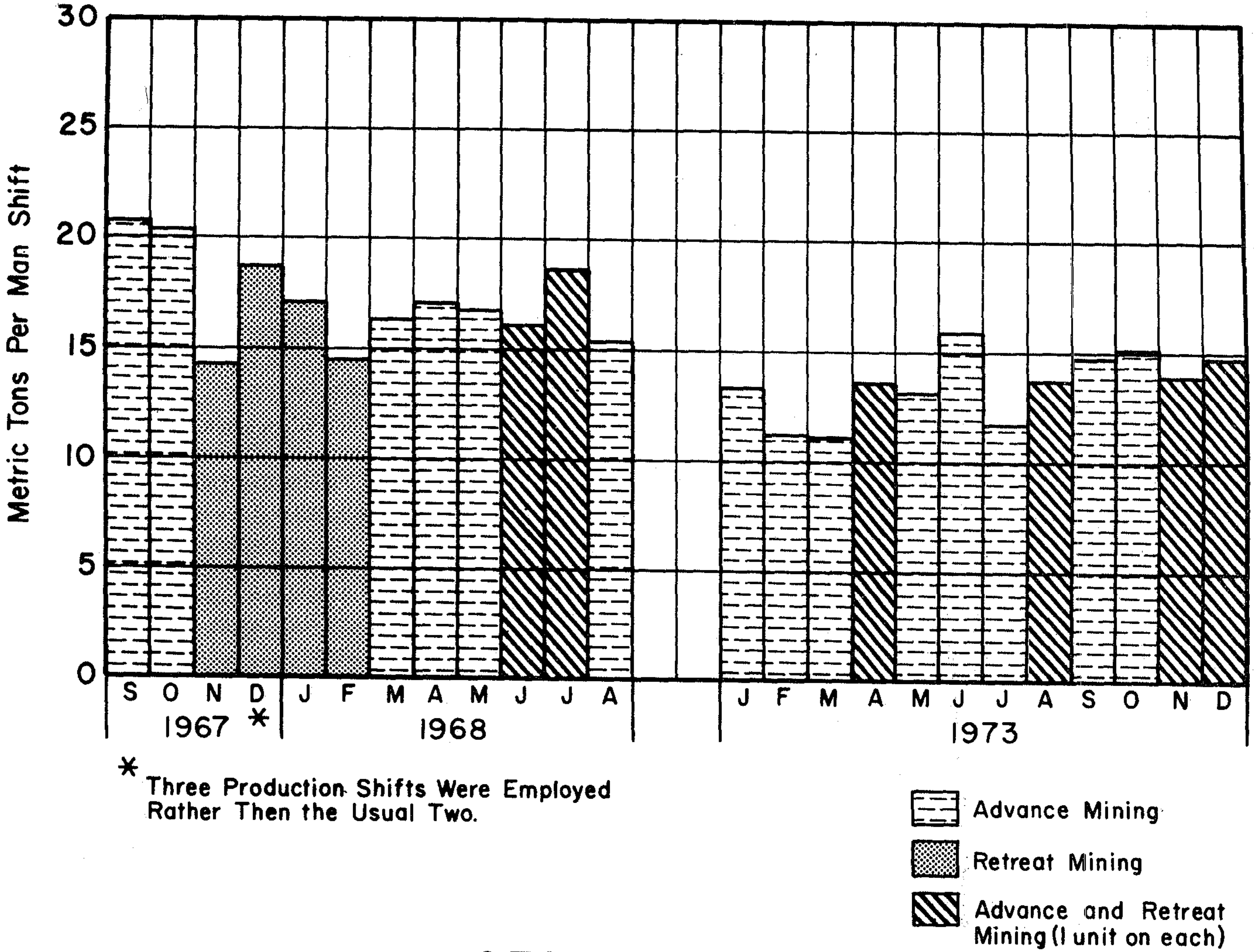
Trends evident in raw coal productivity data, graphed in Figure 7, match almost perfectly for both production years with the Raw Coal Production Chart (Figure 6) previously described. Raw coal productivity on a tons per man-shift basis also showed a significant decline after passage of the Health and Safety Act. One reason for decline in productivity was the required addition of non-production mine personnel for health and safety reasons, thereby increasing the number of personnel while overall production stayed the same or declined.

Figure 7 also attempts to show relationships between advance and retreat mining and productivity. Valid conclusions can only be drawn from this chart when both production units are either advancing or retreating. As the chart shows, this occurred during the first nine months of 1967-68, but not at all during 1973. Advance mining generally permitted greater productivity under typical mining con-



STOTT NO. I MINE — RAW COAL PRODUCTION
BEFORE AND AFTER HEALTH AND SAFETY ACT

Figure 6



STOTT NO.1 MINE
RAW COAL PRODUCTIVITY

Figure 7

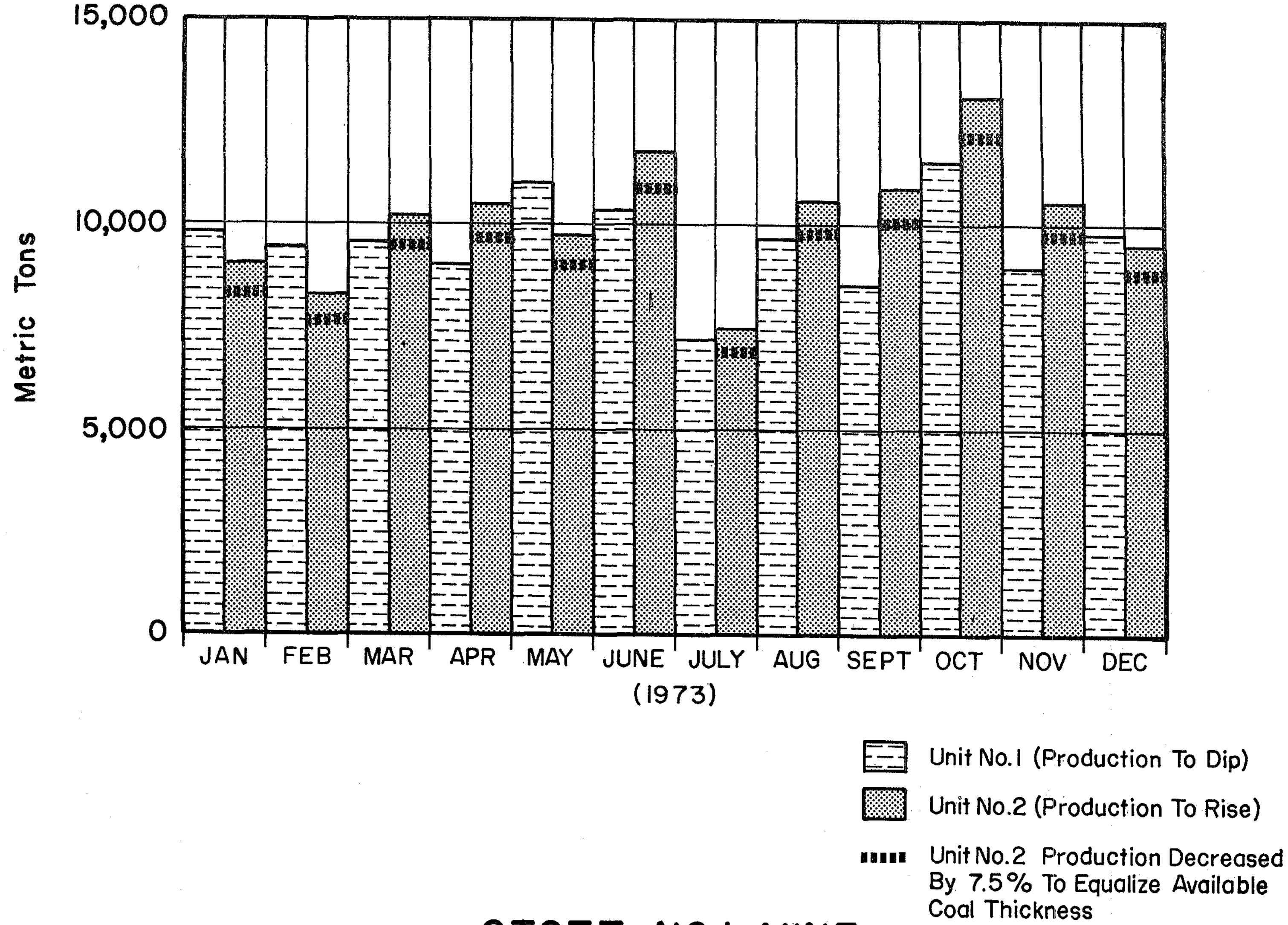
ditions. Retreat mining productivity was only comparable for one month, when skeleton production crews were used on each shift to permit three production shifts (instead of the normal two).

Greater productivity attributable to advance mining is logical, since it has several obvious advantages over retreat mining. Retreat mining generally consists of either splitting or completely removing pillars left by the advancing unit, there is a smaller volume of coal available for mining than there was during the advancing phase. In addition, more equipment maneuvering is generally required during retreat mining, further decreasing productivity.

The third chart (Figure 8) derived from Stott No. 1 Mine data shows monthly unit production rates for 1973. This information could not be graphed for the earlier production year because no unit production breakdowns were available. Originally the 1973 data was graphed in an attempt to illustrate trends for mining to the rise versus mining to the dip. Production Unit No. 1 mined to the dip for most of the year, while Unit No. 2 worked to the rise; thus, if one mode of operation yielded increased production, it would be evident as a trend on this chart. However, no apparent trends can be distinguished, and it appears that, at least for the Stott No. 1 Mine, mining to the dip has no substantial production benefits or deficiencies.

Unit No. 2 did produce 5.8 percent more coal in 1973. However, geologic and mining information reveals that coal mined by Unit No. 2 was thicker by an average of three inches, which would account for a 7.5 percent increase in volume of available coal. To equalize the monthly production of each unit in terms of coal thickness mined, the production figures for Unit No. 2 were decreased by 7.5 percent. These adjusted production figures suggest or estimate production rates for Unit No. 1 operating to the dip, and Unit No. 2 operating to the rise, in a coal seam of uniform thickness. Both actual and revised production rates for each unit are shown in Figure 8.

Assuming an equal volume of available coal, production for Unit No. 1 exceeded that of Unit No. 2 by 2.2 percent. The Stott Mine's mining records, presented in Table 4, also show that Unit No. 2 spent 1.5 more months in retreat mining, with its lower productivity, than did Unit No. 1. Therefore, when all of these factors are combined and evaluated, production from the two units was approximately equal for 1973; and mining to the dip was no more or less advantageous than mining to the rise.



STOTT NO.1 MINE
MONTHLY UNIT PRODUCTION RATES

Figure 8

There are numerous factors generally unrelated to type of mine development which can have adverse effects on production and economics in underground mines. Combined effects of these factors can sometimes be so great that they obscure advantages or disadvantages of mining to the dip. These factors are frequently beyond control of the operator, and cannot be accurately predicted beforehand. Mine roof and floor conditions, for example, are extremely important. Highly fractured roof rock requires increased roof bolting, possibly augmented by timber cribs, props, crossbars and beams. Additional temporary supports are required while these permanent supports are being placed. These roof conditions or fracture zones can also be associated with excessive groundwater infiltration, resulting in increased pollution formation and pumping requirements. Resulting wet mine floor conditions, especially where the floor consists of relatively soft clay, can also adversely affect production. Mobile equipment and coal shuttle cars may bog down and small pillars and timbers may sink into the mine floor, causing adjacent portions of the floor to heave. This infringement on available vertical space in the workings can hamper production, or even force early closure of affected portions of the mine.

Geologic and hydrologic conditions can vary sufficiently to hamper production locally regardless of mining techniques employed. Zones of foreign mineral veins such as pyrite or clay, decreases in coal thickness, locally steep dips, and major or minor fault offsets of the coal seam either decrease the amount of available coal or increase difficulty of extracting that coal by reducing efficiency of mining, loading, or shuttle equipment. Some mines are extremely dry, while others encounter volumes of water sufficient to force closure. The amount of groundwater encountered in mines is usually dependent upon a number of factors, all of which relate to infiltration and capacity of mine workings to intercept that infiltration. Important factors include orientation and configuration of workings, mine size, local mining history, mining methods employed, roof and floor rock conditions, depth and type of overburden, degree of rock faulting and fracturing, amounts and rates of precipitation, and proximity to aquifers.

Coal barrier and pillar placement can also have a significant effect on productivity, since this determines the amount of unmined coal that must remain. Pillars and barriers vary in size and number according to mine size, orientation, geologic and hydrologic conditions, and depth. Permanent barrier pillars must be left around such underground obstacles as gas, oil, and water wells, adjacent mine workings, and heavily used entryways. Large pillars must also be left beneath surface structures which cannot tolerate subsidence. In addition, the size of normal pillars increases with depth of overburden above the seam.

It can be seen that most production-reducing factors discussed above do not relate specifically to mining techniques employed. In fact, most of them are independent variables. Productivity from any specific mining technique, including down-dip mining, is highly dependent on these factors. Based on this information, and on production records reviewed for the Stott No. 1 Mine, no significant production decreases have been noted that can be attributed to down-dip mining.

ECONOMICS

There are numerous factors in the coal mining industry that contribute to production costs of a ton of coal. Particular attention in this section is given to economic considerations which may differ for mines developed to the dip versus mines developed to the rise. Mention of specific dollar costs incurred at the Stott No. 1 Mine is avoided to protect competitive interests. Instead, cost items are presented as a percentage of approximate total production cost per ton of coal.

Information obtained at the Stott No. 1 Mine is supplemented in this report with information presented in a recently published United States Bureau of Mines study of estimated capital investments and operating costs for bituminous coal underground mines. The smallest mines evaluated in that study produced about one million tons of coal annually, substantially more than the Stott No. 1 Mine. However, much data presented for that production rate is valid and relevant in describing smaller operations. According to the Bureau's report, production costs in this tonnage range breakdown in approximately the following percentages:

Total labor and supervision	40%
Payroll overhead	14%
Union welfare	12%
Operations supplies	14%
Total power consumption	3%
Indirect and fixed costs	17%
TOTAL	100%

Most primary production cost factors listed above are not affected at all by utilization of down-dip rather than up-dip mining techniques. The same mining equipment is utilized; thus, there is no change in number of mine personnel. This means total labor, supervision, payroll overhead and union welfare remain unchanged. Since actual coal extraction techniques do not vary substantially, operations supplies and indirect and fixed costs also remain approximately the same. Thus, the parameter showing greatest degree of variation reflecting production economics of up-dip versus down-dip mining is power consumption.

The portion of production costs relating to power consumption represents the most critical cost variable in a comparative evaluation of down-dip and up-dip mining. As underground mines increase in size, power consumption percentage of total production costs declines below the 3 percent shown above. Total power costs in the Bureau of Mines study ranged from 11 to 17 cents per metric ton (12 to 19 cents per short ton). One active mine using continuous miners and rail haulage reported power consumption costs of $0.56 per metric ton ($0.50 per short ton), but even that represented only 4 or 5 percent of the total production cost.

Since power consumption is a key variable in assessment of down-dip mining economics, a closer examination of the relative power consumption percentages of various pieces of mining equipment is in order. Equipment utilizing substantial amounts of power includes continuous miners, undercutting saws, loading machines, shuttle cars, roof bolters, belt feeders, secondary conveyor belt systems used in headings, gathering pumps, ventilation fans, rock dusters, and shaft and slope hoists. There are other smaller consumers of electrical power too numerous to mention. The Stott Mine, which is smaller than any of the mines evaluated in the Bureau of Mines' eport, had a slightly higher power consumption cost percentage. lectrical costs at the Stott Mine account for about five percent of otal production costs. The following table of equipment and power onsumption percentages provides a relative comparison of electrical osts for normal operations at the Stott No. 1 Mine.

Table 6

STOTT NO. 1 MINE

POWER CONSUMPTION BREAKDOWN

Equipment	Hours/Week	Percentage of Total Mine Power Consumption
Main belt drives	80	37.9%
Ventilation fan	168	16.6
Main pumps	75	11.1
Drift to stacker belt drive	80	8.6
Side belt drives	80	6.3
Treatment plant	168	5.0
Roof bolter	30	3.6
Tipple	35	3.5
Loaders	30	1.8
Shuttle cars	40	1.4
Face drill	40	1.2
Undercutting saw	10	1.0
Miscellaneous	–	2.0
		100.0%

As Table 6 shows, haulage of coal constitutes by far the single largest power cost, despite the fact that the Stott Mine only produces coal on 2 shifts per day. Haulage during those two shifts accounts for 53 percent of all power consumed by the mine, or about 2.5 percent of total production cost. The Stott Mine accomplishes haulage exclusively through primary and secondary conveyor belt systems. This means of coal transport can, depending on mine conditions, be much more efficient than rail haulage, and has gained almost exclusive use in newer underground mines. Belts have a significant advantage over rails in that rails are restricted to grades of less than three degrees, while belts can be efficiently utilized where grades are as high as 14 to 16 degrees.

Terminology used in discussions of conveyor haulage in up-dip and down-dip mines is potentially confusing, thus a brief explanation is presented. In a down-dip mine, where development progresses toward lower elevations and the coal face is down-grade from the mine entry, conveyors must haul coal up-grade. The opposite is true of mines developed to the rise, where coal must be hauled down-grade from the working face to the mine entry.

Unfortunately, cost and power consumption figures for the Stott Mine's two production units could not be separated to the extent necessary to effectively evaluate coal haulage up-dip versus down-dip. To compensate for this data gap, a conveyor belt system manufacturer was contacted to determine how belt system operating costs were computed during planning and development of mine haulage systems. Information obtained from this source was invaluable for computation of conveyor haulage costs under various conditions.

There are a number of variables which must be considered when attempting to determine belt haulage costs. Factors such as anticipated production and haulage rates, layout of the mine workings, angle of grade to the rise or dip, belt segment length, and the width, composition, and velocity of the belt are all important in computing horsepower, energy consumption and costs of belt haulage. Table 7 includes many of these variables, and defines their effects on up-grade and down-grade conveyor haulage costs per metric ton.

The primary purpose of Table 7 is evaluation of unit cost differentials for up-grade conveyor haulage, as would be required in a mine developed to the dip, and down-grade haulage, which would be employed in an up-dip mine. To permit this, a constant belt length of 305 meters

Table 7

CONVEYOR BELT OPERATION COSTS
UP-GRADE VS. DOWN-GRADE HAULAGE

BELT WIDTH	HAULAGE RATE KKG/HOUR (SHORT TPH)	HAULAGE DIRECTION	ELEVATION CHANGE IN 1000 METERS (DEGREES DIP OF BELT)												
			5m (0.30°)	10m (0.57°)	15m (0.86°)	20m (1.15°)	25m (1.43°)	30m (1.70°)	40m (2.30°)	50m (2.86°)	60m (3.43°)	80m (4.57°)	100m (5.70°)	125m (7.12°)	150m (8.50°)
61 cm. (24 in.)	45.4 (50)	Rise	0.46	0.46	0.48	0.48	0.50	0.50	0.52	0.54	0.56	0.60	0.64	0.68	0.74
		Dip	0.44	0.44	0.42	0.42	0.42	0.42	0.40	0.38	0.36	0.34	0.30	0.28	0.24
	90.7 (100)	Rise	0.27	0.28	0.29	0.30	0.31	0.32	0.33	0.35	0.37	0.41	0.45	0.49	0.54
		Dip	0.25	0.25	0.24	0.23	0.22	0.22	0.20	0.19	0.17	0.15	0.12	0.08	0.05
	136.1 (150)	Rise	0.21	0.21	0.22	0.23	0.24	0.25	0.27	0.29	0.31	0.35	0.38	0.43	*
		Dip	0.19	0.18	0.17	0.17	0.15	0.15	0.14	0.12	0.11	0.08	0.05	0.02	
	181.4 (200)	Rise	0.17	0.18	0.19	0.20	0.21	0.22	0.24	0.26	0.28	0.31	0.35	*	*
		Dip	0.16	0.15	0.14	0.14	0.13	0.12	0.11	0.09	0.08	0.05	0.02		
	226.8 (250)	Rise	0.15	0.16	0.17	0.18	0.19	0.20	0.22	0.24	0.26	0.29	0.33	*	*
		Dip	0.14	0.13	0.12	0.12	0.11	0.10	0.08	0.07	0.06	0.03	0.01		
76 cm. (30 in.)	272.1 (300)	Rise	0.16	0.16	0.18	0.19	0.19	0.20	0.22	0.24	0.26	0.30	0.33	*	*
		Dip	0.14	0.13	0.13	0.12	0.11	0.11	0.09	0.08	0.06	0.07	0.01		
	317.5 (350)	Rise	0.15	0.15	0.16	0.17	0.18	0.19	0.21	0.23	0.25	0.29	*	*	*
		Dip	0.14	0.12	0.11	0.11	0.10	0.09	0.08	0.07	0.05	0.02			
	362.8 (400)	Rise	0.14	0.15	0.16	0.17	0.17	0.18	0.20	0.22	0.24	0.28	*	*	*
		Dip	0.12	0.11	0.10	0.10	0.09	0.09	0.07	0.05	0.04	0.02			
	408.2 (450)	Rise	0.13	0.14	0.15	0.16	0.17	0.18	0.20	0.22	0.23	0.27	*	*	*
		Dip	0.11	0.11	0.10	0.09	0.08	0.08	0.06	0.05	0.04	0.01			
	453.5 (500)	Rise	0.12	0.13	0.14	0.15	0.16	0.17	0.19	0.21	0.23	0.26	*	*	*
		Dip	0.11	0.10	0.09	0.09	0.08	0.07	0.06	0.04	0.03	0.01			

* Down-dip haulage yielded negative values, which are not valid. In such cases, the length of the belt segment would be reduced to permit computation of accurate costs.

NOTE: Costs are in cents per kkg of coal hauled per 305 meter belt segment.

(1000 feet) and velocity of 122 meters per minute (400 feet per minute) were assumed. Ten different production rates, ranging from 45.4 to 453.5 metric tons per hour (50 to 500 short tons per hour) were considered, as were two different belt widths - 61 and 76 centimeters (24 and 30 inches). A range of slope or grade angles between 0.3° and 8.5° is evaluated, and costs per metric ton for belt transporting coal up and down those grades are computed.

Table 7 exhibits several noteworthy trends concerning costs of utilizing conveyor belts to transport coal. As would be expected, costs for haulage up-grade increased with greater slope angles and the opposite was true for down-grade haulage. Second, the cost per metric ton for transporting coal over a constant belt length at a constant velocity was extremely low (always less than one half cent per ton) regardless of other mine conditions. The table also shows that use of larger belts and higher haulage rates can significantly lower the cost per metric ton of belt haulage.

This table also illustrates cost variations between up-grade and down-grade haulage over a predetermined slope angle. The percentage variation was computed for each pair of up-grade and down-grade unit haulage costs evaluated. These percentages are shown in Table 8 and are graphically illustrated in Figures 9 and 10. The table and figures show percentage increase in haulage costs attributable to up-grade rather than down-grade coal haulage ranges from a few percent in shallowly sloping settings to several hundred percent in steeper slopes. However, the table and the figure can be somewhat misleading if not carefully considered. While cost increases of 300 percent or 400 percent for up-grade haulage appear quite substantial, the actual variations in haulage costs are generally much less than one half cent per metric ton.

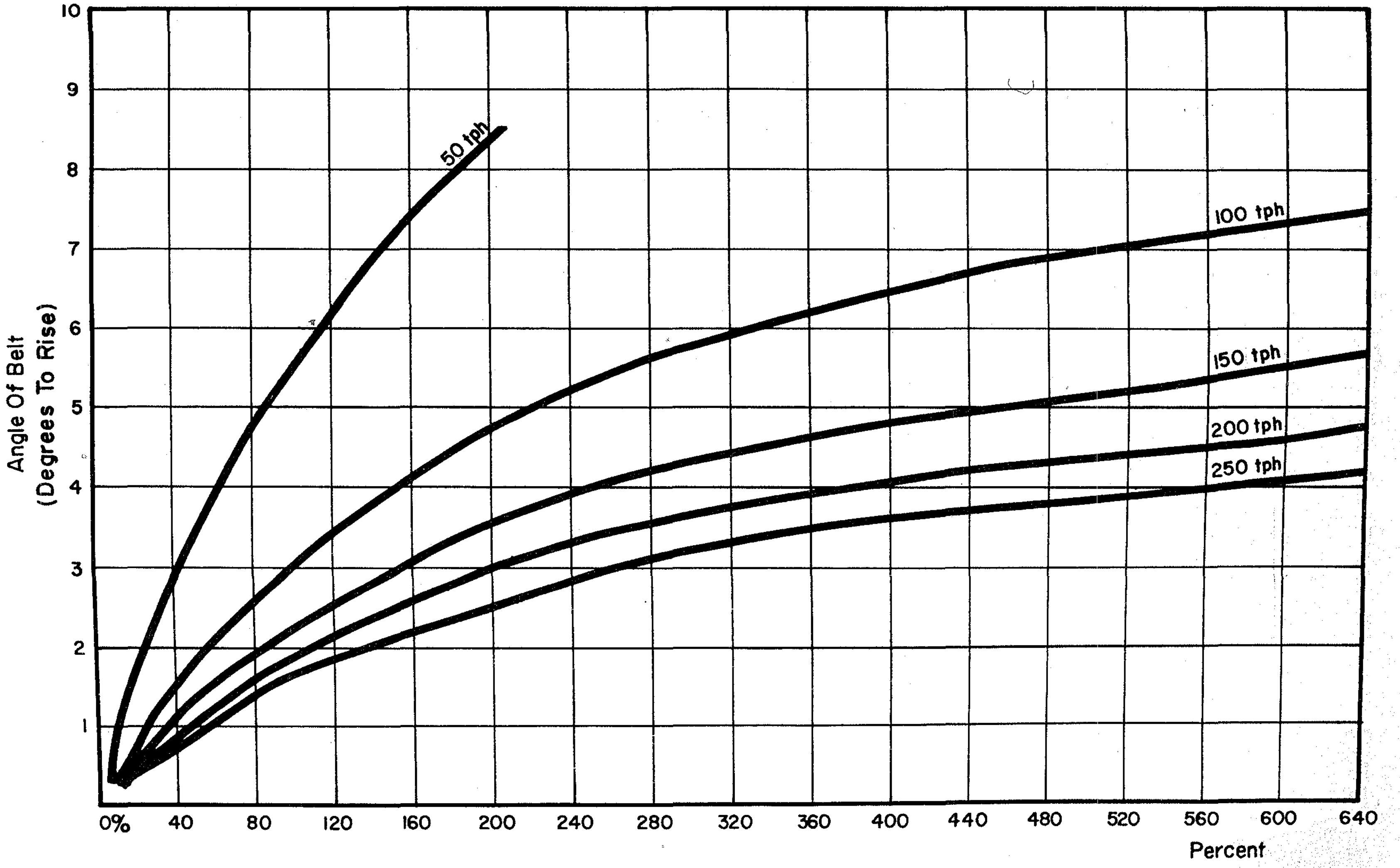
To summarize the conveyor haulage statistics that have been presented here, it appears the economic impact of up-grade versus down-grade coal haulage is relatively minimal. There may be a significant haulage cost increase in terms of percentage of down-grade haulage, but unit costs for both modes of haulage are so low (generally from 0.1 to 0.5 cents per metric ton) that these increases are insignificant. Thus, direction of belt coal haulage in any mining situation, including mining to the dip, does not appear to be a significant economic factor.

Table 8

PERCENTAGE COST INCREASES IN UP-GRADE OVER CONVEYOR HAULAGE DOWN-GRADE

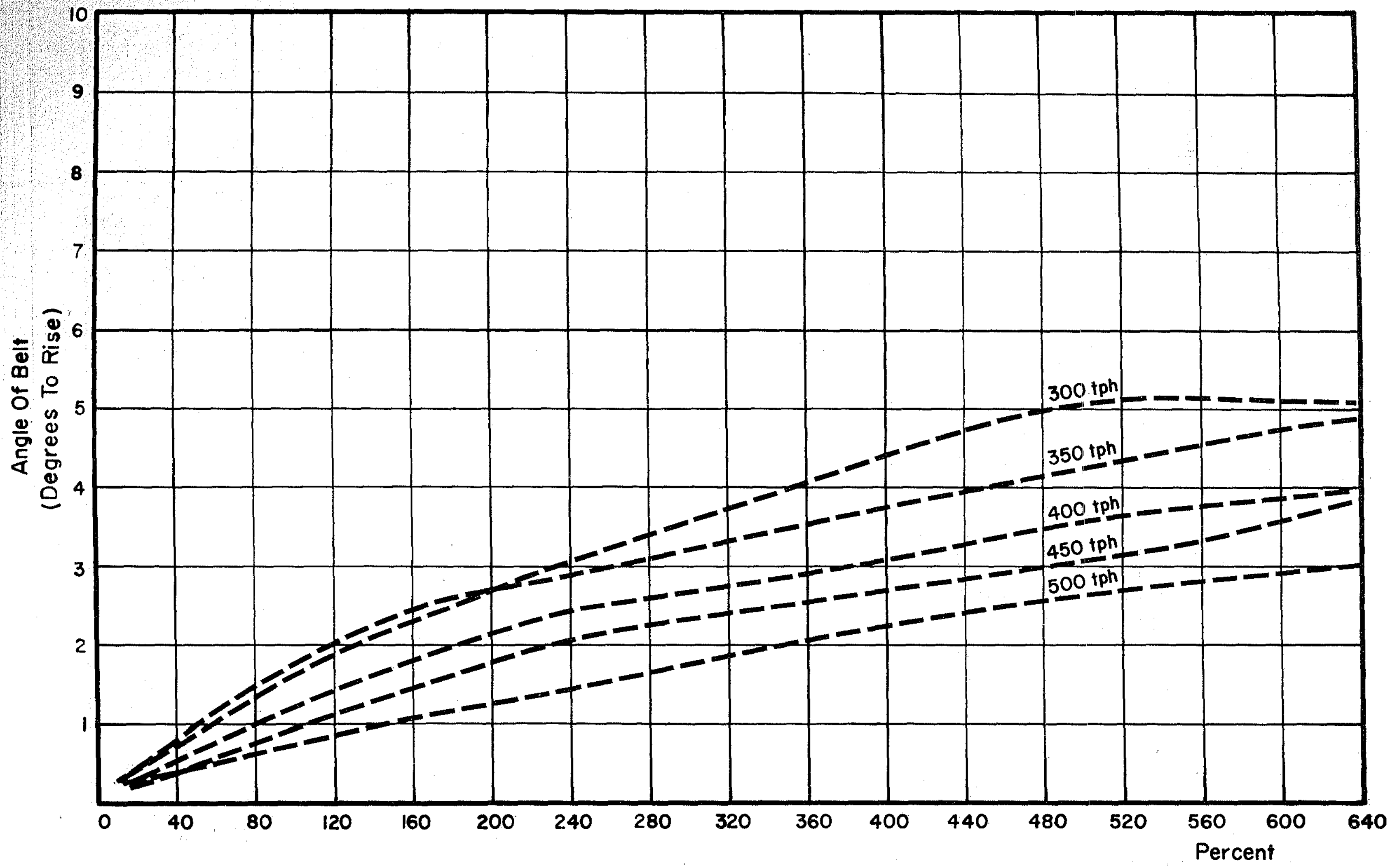
BELT WIDTH	HAULAGE RATE KKG/HOUR (SHORT TPH)	ELEVATION CHANGE IN 1000 METERS (DEGREES DIP OF BELT)												
		5m (0.30°)	10m (0.57°)	15m (0.86°)	20m (1.15°)	25m (1.43°)	30m (1.70°)	40m (2.30°)	50m (2.86°)	60m (3.43°)	80m (4.57°)	100m (5.70°)	125m (7.12°)	150m (8.50°)
61 cm. (24 in.)	45.4 (50)	4.5	4.5	14.3	14.3	19.0	19.0	30.0	42.1	55.6	76.5	113.3	142.8	208.3
	90.7 (100)	5.9	12.0	20.8	30.4	40.9	45.5	65.0	84.2	117.6	173.3	275.0	512.5	980.0
	136.1 (150)	10.5	16.7	29.7	35.3	50.0	66.7	92.9	141.7	181.8	337.5	660.0	2000	*
	181.4 (200)	6.3	20.0	35.7	42.9	61.5	83.3	128.6	188.9	250.0	520.0	1650	*	*
	226.8 (250)	7.1	23.1	41.7	50.0	72.7	100.0	175.0	242.8	333.0	867.0	3200	*	*
76 cm. (30 in.)	272.1 (300)	14.3	23.1	38.5	58.3	72.7	81.8	144.4	200.0	333.0	329.0	3200	*	*
	317.5 (350)	7.1	25.0	45.4	54.5	80.0	111.0	163.0	229.0	400.0	1350	*	*	*
	362.8 (400)	16.7	15.4	60.0	70.0	88.9	100.0	186.0	340.0	500.0	1300	*	*	*
	408.2 (450)	18.2	27.3	50.0	77.8	112.5	125.0	233.0	340.0	475.0	2600	*	*	*
	453.5 (500)	9.1	30.0	55.6	66.7	100.0	142.9	216.7	425.0	667.0	2500	*	*	*

* Down-dip haulage yielded negative values, which are not valid. In such cases, the length of the belt would be reduced to permit computation of accurate costs.



PERCENT INCREASES IN COST PER KKG
OF 63 cm. BELT COAL HAULAGE UP-GRADE
VERSUS DOWN - GRADE

Figure 9



PERCENT INCREASES IN COST PER KKG
OF 76 cm. BELT COAL HAULAGE UP-GRADE
VERSUS DOWN-GRADE

Figure 10

Pumping is a second key parameter in underground mine power consumption that could be seriously affected by employment of down-dip mining techniques. All of the workings in any mine drain by gravity to the lowest point. In most down-dip mining plans, that lowest point is located in the active portion of the mine. Since there is no discharge point there, and all mine water is draining toward the coal face, continuous or at least intermittent pumping is frequently required to maintain a dry working area.

Pumping is a highly variable parameter, dependent on geologic and hydrologic conditions, type of equipment utilized, and the mining plan. A mine developed exclusively to the rise would obviously have a distinct advantage over one developed to the dip, since the up-dip mine would drain largely by gravity to the discharge point. Pumping would still be required in some actively worked panels, but equipment and time expenditures would be relatively small. In an advancing down-dip mine, where virtually all water must be removed to permit continued production, pumping requirements are totally dependent upon mine size, geology and hydrology.

Another factor which may have a minimal effect on a mine's pumping requirements is utilization of continuous miners. These machines utilize a low volume water spray to control dust at the coal face. If the mine is operated in an acid-producing coal seam, all drainage within the workings may be too highly acid and corrosive to use in the sprays. Additional water must then be pumped from the surface to the continuous miners for spraying purposes, thereby increasing the volume of water that must be eventually pumped. Depending on the number of continuous miners in operation, this additional volume of water may be very small or fairly large.

Information available for pumping is general in nature, however, and can lead to no specific conclusions. It can only be stated that pumping costs may be substantially increased by down-dip development, but will probably not reach the point where they adversely affect production economics. The Stott Mine might exemplify this, since it can be considered an average mine in terms of amount of pumping required. Even here, energy consumption costs for pumping represent less than one percent of the mine's total production cost.

Many of the other power-consuming processes or pieces of equipment listed in Table 6 will not be affected by utilization of down-dip rather than up-dip mining procedures. Undercutting saws, roof bolters, face drills, loaders and shuttle cars all operate equally well in any grade or direction of grade normally encountered during mining. No changes in mining operations, their sequence or numbers of equipment operating personnel are required. Ventilation fans and tipple facilities will also maintain the same power consumption levels regardless of the manner of mine development.

One of the most critical mining cost factors and potential advantages of down-dip mining is realized only after mine closure. As previously mentioned, closure of mines developed to the rise requires expensive sealing of entire mines or many segments of some mines. The ability to effectively seal a mine and prevent discharges is determined by three factors: 1) hydraulic head that will ultimately affect the seal; 2) integrity of the coal adjacent to the seal; and 3) strength and integrity of coal barriers surrounding the mine. Mine seals frequently cannot be constructed to dependably withstand more than 6.1 to 7.6 meters (20 to 25 feet) of hydraulic head. Many underground mines in Appalachia, however, have total elevation changes many times greater than this. Such mines would have to be sealed in segments, each segment having less than the maximum permissible head. This can be extremely dangerous because failure of any one of the multiple seals could place an undue stress on consecutively lower seals, resulting in their failure and subsequent failure of the entire sealing operation.

One of the principal problems in seal construction lies not with the seal, which can be designed to withstand almost any amount of pressure, but with integrity of coal adjacent to the seals. As a result of mining, this coal is frequently in a fractured condition, and seals cannot easily be tied to or interlocked with the mine walls. The result is slowly increasing seepage around the seal as hydraulic pressure increases.

The coal barrier itself must also be of sufficient strength to maintain anticipated pressures. Where a mine is developed to the rise, the barrier on the down-dip or entry side of the mine will be a portion of the coal outcrop. If mine workings came too close to the outcrop, as was frequently the case when "grass roots" mining was common years ago, the outcrop coal barrier can be substantially weakened, rendering it incapable of withstanding any great amount of pressure. This, too, can result in failure of a mine sealing project, either by steadily increasing seepage rates or by complete outcrop failure.

Coal barrier integrity is no longer a major sealing factor in some of the states which have some form of barrier pillar requirements. However, it is not uncommon for such requirements to be excessive. Excessive barrier restrictions serve no valid safety purposes, and only tie up valuable coal reserves. Thus, poorly established requirements can result in an economic and energy loss.

When mines are developed to the dip, the first of these problems concerning strength of the seal and its bond to mine walls, are relatively unimportant. Since mine entries are on the up-dip side of the workings, they will be subjected to very little or no hydraulic head following closure. The question of coal barrier integrity becomes extremely important, however, because the workings will naturally inundate. If the low side of the mine workings is adjacent to a coal outcrop or the workings of another active or abandoned mine, care must be taken to leave enough coal in the barrier pillar to withstand anticipated heads upon closure. Since all coal left in the barriers is potential profit which can never be realized, it is also important to assure that only the barrier thickness required to safely maintain mine inundation remains unmined. This allows a maximum amount of coal to be extracted during down-dip mining without reducing safety or effectiveness of mine inundation.

In the past, both industry and government were concerned more with barrier pillar widths for reasons of safety, mine subsidence and property rights. Until recently, little or no consideration has been given to the importance of barrier pillars in maintaining post-mining inundation, which in turn minimizes water pollutant production and preserves environmental quality. Attention has only recently been focused on evaluation of barrier pillar requirements.

The Commonwealth of Pennsylvania has been a leader in assessing importance of adequate barrier pillar thicknesses. Pennsylvania's current barrier pillar restrictions, which are actually "rules of thumb" rather than legislative doctrines, are based primarily on predictions from past mining experiences. Prior to 1967, this State's barrier pillar widths were determined by thickness of the seam mined, but this proved inadequate when hydraulic heads exceeded 15 meters (50 feet). Post-1967 regulations require mines developed to the dip maintain at least a 15 meter (50 foot) barrier along the coal outcrop and between adjacent workings. Mines developed to the rise are governed by regulations which vary according to the anticipated hydraulic head. Hydraulic heads of less than 30 meters (100 feet) require at least one half meter (1.5 feet) of barrier pillar per foot of head. Where anticipated hydraulic heads exceed 91 meters (300 feet), mining is generally discouraged.

Several other states also have barrier pillar restrictions, some of which are based upon Pennsylvania's "rule of thumb." Requirements in West Virginia and Maryland are similar to those in Pennsylvania. Kentucky permits mining to within 7.6 meters (25 feet) of adjacent property lines or mines. Alabama requires a 152 meter (500 foot) barrier pillar around each mine, regardless of conditions or size. Virginia requires a 61 meter (200 foot) property line barrier only where the adjacent property is mined. Tennessee and Ohio have no definite barrier pillar requirements.

The wide range of variability in these regulations points to the fact that the problem of assessing barrier pillar conditions and determining thickness requirements has not yet been solved. With the increase of down-dip mine development for pollution control, this problem will require extensive consideration, hopefully culminating in development of an effective formula or equation for predicting adequate but not exaggerated barrier pillar requirements. Dependable regulations would maximize economics of mining to the dip while assuring that environmental degradation is prevented.

HEALTH AND SAFETY CONSIDERATIONS

Down-dip mining is not a new and untried mining technique, but one which has been locally implemented for many years. Therefore, the health and safety aspects of this technique have been well established, eliminating need for speculation. There is no significant difference in any aspect of down-dip mining as opposed to up-dip mining. The same equipment, personnel, coal extraction methods and haulage techniques are employed. The only alteration of normal mining procedures may be pumping increases required to minimize water impoundment at the coal face, but this is highly variable and does not affect health and safety. In addition to these points, both up-dip and down-dip mining must remain in compliance with the stringent, effective requirements of the Health and Safety Act of 1969. Thus, there are no noteworthy health and safety aspects of down-dip mining that could adversely affect its implementation.

NATIONAL IMPACTS

National impacts of mining to the dip rather than to the rise are difficult to assess in terms of water quality improvements, because mine drainage pollution is not really a nationwide problem. Production of mine drainage pollutants (acid, iron, sulfates, etc.) is a regional and local phenomena, dependent upon regional character, local geology, specific coal seams mined and extent of mining. The most significant pollutant attributable to coal mining is acid mine drainage, which is widespread only in the Eastern or Appalachian Coal Province. States in this region which are faced with degradation of streams by acid mine drainage include Pennsylvania, Ohio, Maryland, Virginia, West Virginia, Kentucky, Tennessee, and Alabama. There are also local acid pollution problems in other portions of the Nation, as well as non-acid areas in which iron or other mine drainage pollutant concentrations are above acceptable levels. In all of these cases, only a few of the coal seams present are pollution sources.

Regional structure of the Nation's various coal basins is also a relevant factor in determining national impacts. Much of the coal west of the Appalachian Basin dips very shallowly or is flat-lying. In such areas, mines may extend in any or all directions with no consideration for the minimal dip that does exist. Down-dip mining is, therefore, only a valid mining technique in those coal basins in which dips of the seams exceed one or two degrees.

The extent and type of mining conducted in an area is second in importance only to the chemical quality of coal and overburden materials in determing acid production extent and severity. Highly pyritic coal seams which have not been exploited by mining are seldom significant sources of mine drainage. Numerous studies conducted in the Appalachian coal field have attributed 70 to 90 percent of all acid mine drainage pollution to abandoned underground mines. The vast majority of these mines were originally developed to the rise to permit gravity drainage and coal haulage.

Pollution significance of previously abandoned underground mines cannot, however, be construed to imply that underground mines abandoned now or in the future will be similar pollution sources. Recently proposed effluent limitations guidelines for the coal industry require that mine operators be held permanently responsible for the quality of effluent from their mines, during mining and after abandonment. They must meet certain effluent standards regardless of the mining techniques they employ. Thus, in terms of water quality improvement on a national scale, employment of down-dip mining will have little effect, because all mine discharges must be of acceptable quality.

Down-dip mining will, however, be of major economic importance to mine operators who, as a result of these proposed effluent limitations, are faced with the prospect of perpetual treatment of discharges from their abandoned operations. Such permanent treatment will be absorbed as a production cost increase in the operator's other active operations, and will therefore have a significant national impact. The U. S. Environmental Protection Agency's recently prepared Development Document for Effluent Limitations Guidelines and Standards of Performance for the Coal Mining Point Source Category presents treatment cost data for several of the coal industry's best acid mine drainage treatment plants all of which employ lime neutralization. That study indicates costs of mine drainage treatment are dependent on several factors. Construction costs are largely determined by the volume of water that will be treated. Facilities that must be included in computation of construction costs for a treatment plant are land, holding basin, control building, lime storage, lime and feed mixer, aeration facilities, settling basins, fencing and roads, sludge disposal equipment and basins, instruments and electrical apparatus, and pumps.

Since most treatment plants employ earthen settling basins in their treatment process, land requirements can become very significant. Treatment facilities are generally confined to land requirements of less than ten acres. Even this may present problems where underground mines are concerned, since surface ownership is frequently confined to the mine access areas. However, land acquisition is generally not a significant aspect of treatment facility cost.

Disposal of sludges produced in treatment of mine drainage is an increasing problem which must be considered. Recently constructed treatment plants are providing settling basins with capacities of many millions of liters and provide sludge storage for several years. In some plants, sludge is intermittently removed for disposal in abandoned portions of the underground mine workings.

In addition to plant construction costs, treatment costs must also be considered. Costs in the previously cited EPA document ranged from $0.03 to $0.11 per thousand liters ($.11 to $.40 per thousand gallons) regardless of the acidity concentration. Lime is the most commonly used neutralizing agent, but other chemicals that can be used include limestone, soda ash and caustic soda. Limestone, the raw material, is readily available for production of lime; however, there is presently a tight market for availability of lime due to closing of several plants for air pollution problems. Availability of lime and other alkalies is a factor that could significantly increase costs of future treatment of underground mine discharges.

Even where all acid has been removed from the mine effluent by treatment, water quality may be unacceptable for certain applications. Such parameters as sulfates and total dissolved solids cannot be economically removed by any treatment process, while others such as hardness and total suspended solids may actually be increased during treatment of the mine drainage.

This extensive discussion of the complexities of acid mine drainage treatment is to amplify potential advantages of the down-dip mining technique. Water quality evaluation in this study has revealed that mining to the dip can yield significantly improved effluent water quality after mine closure. Natural inundation decreases pyrite oxidation and less acid is formed. Even if the effluent from down-dip mines is still slightly acid, as was the case with the Yorkshire Mine, the subsequent costs of perpetual treatment of that discharge are substantially lower than they would be if the same mine had been developed to the rise.

Thus, implementation of down-dip mining in major acid-producing regions could have significant regional and national impact on production costs. The only noteworthy production cost increases attributable to down-dip mining are pumping costs which, as previously mentioned, are so dependent on local geology and hydrology they cannot be predicted on a regional scale. Post-mining treatment cost savings attributable to down-dip mining are impressive, when computed on a perpetual basis, and will be more than sufficient to offset any increased pumping costs. Then, National impact of increased utilization of the down-dip mining technique would actually be favorable rather than adverse.

FEASIBILITY AND APPLICABILITY

Feasibility and applicability of the down-dip mining technique have already been established in numerous applications under various mining conditions. These applications have revealed that there are no real technological limitations to use of the technique, although production economics must be evaluated at each potential minesite. Evaluation of the technique in this study has also revealed there are no significant production cost increases attributable to actual coal extraction or haulage processes. The study shows that pumping is the only significant mining variable that can be adversely affected by down-dip mining. Increased pumping can lead to increased water treatment, and can severely effect production economics at a minesite. However, hydrological and geological conditions that relate to groundwater infiltration vary on a local basis and must be evaluated at each proposed minesite, regardless of mining technique being employed. To counter any increased water handling attributable to down-dip mining, several alternative mining procedures are available.

Since down-dip mining is not an entirely new or experimental technique, its range of applicability has already been fairly well established through actual use. Normal down-dip mining procedure has been to enter a coal seam from an up-dip coal outcrop and expand from there toward the dip. As the mine develops to the dip, all infiltrating water entering mined out areas, which are up-dip from the advancing coal face, drains by gravity toward the working face. Depending on local conditions, extensive pumping may be required at the coal face to permit mining to continue. As mining proceeds, roofs of older abandoned portions of the workings may settle and fracture, causing increased infiltration and necessitating even more pumping.

There are several alternatives or options to this conventional technique that can help reduce pumping requirements. The first of these is merely a modification that is already employed in many mines. Main headings and general trend of workings is still developed toward the dip, with all secondary headings developed parallel to the strike. Rooms or panels driven from these secondary headings are then oriented either to the rise or to the dip. If down-dip panels are developed first, water to be handled will be only that in the immediate vicinity of the active face. Infiltrating groundwater can be allowed to accumulate in these completed panels as up-dip panels are being developed. Local inundation of these down-dip panels can eliminate some flow of mine waters to the lowest portions of the workings, and thus reduce pumping requirements.

Another alternative down-dip mining technique also has potential to substantially reduce production costs by decreasing pumping and treatment requirements. This involves development of a down-dip mine in such a manner that actual mining would generally advance to the rise. To accomplish this, the main heading or headings are initially driven from the up-dip coal outcrop to the lowest anticipated point in the proposed mine. Mining would then expand from this lowest point, at the end of the main heading, and advance to the rise pulling pillars as mining progressed. Initial full-scale production would then occur from the down-dip workings and progress to the rise.

Use of this alternative has several potential advantages and disadvantages which would have to be weighed with local minesite conditions to determine coal production economics. Drainage reaching the newly developing lower portions of the mine would be only that which infiltrated through roof and walls of the main heading, and would, therefore, be minimal in volume. This would also be the case in a normally developed, new down-dip mine, where the relatively small active and mined out areas permit only limited infiltration. Thus, little or no advantage in pumping is realized in earlier stages of mining.

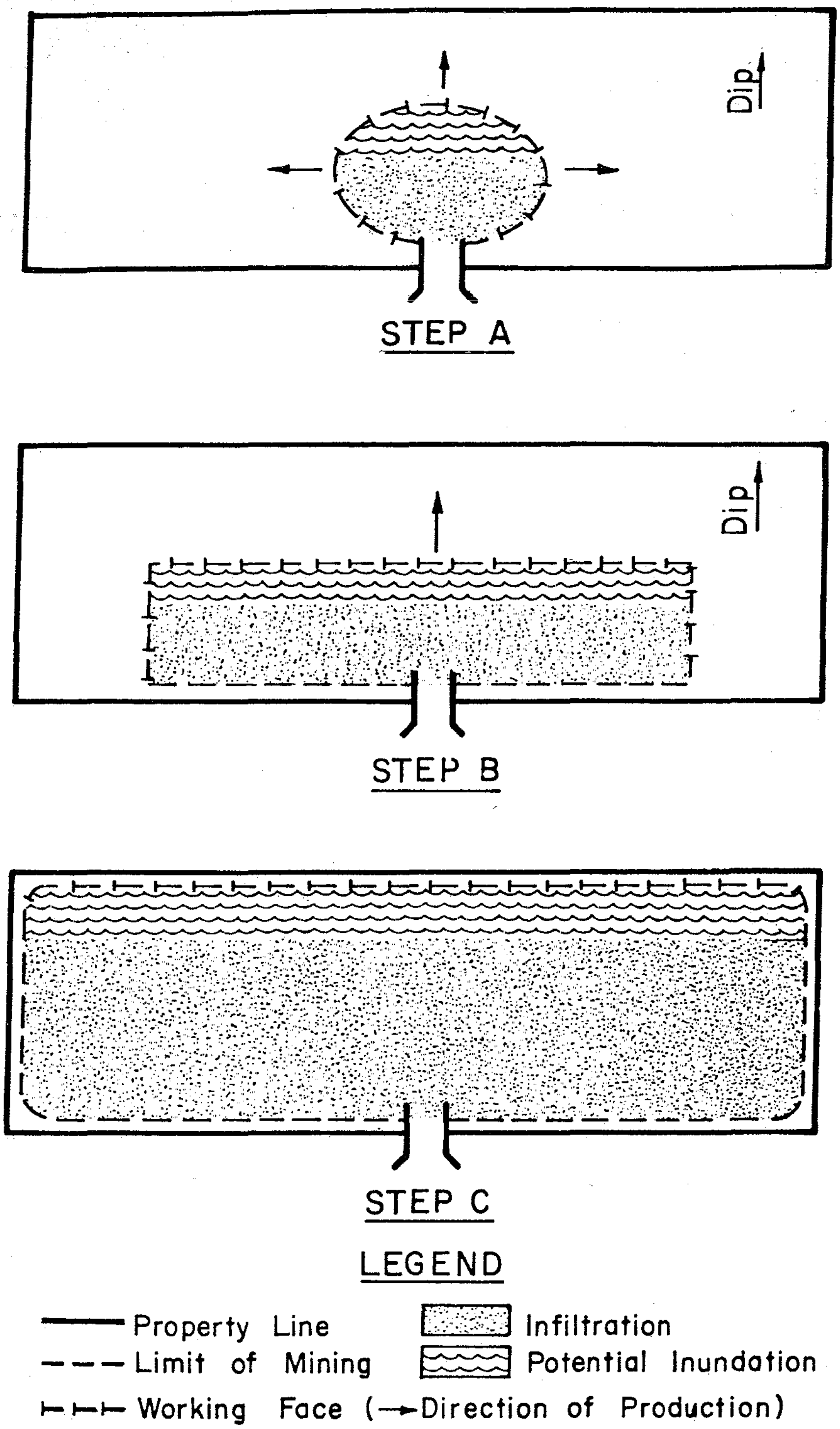
Pumping advantages of this alternative down-dip mining technique are increasingly realized as the mine expands, pillars are split or pulled, and portions of the workings are abandoned. In many down-dip operations, these abandoned areas may be higher in elevation than the active working areas, and therefore, infiltrating water drains by gravity to the active face, where increased pumping is required. If mining is advancing to the rise from the mine's lowest point, infiltrating water can be allowed to inundate low portions of the mine after pillars are removed or split. Thus, as mining is progressing to the rise, the extent of inundation of lower worked out areas is also increasing, and little or no pumping of water from the workings is required. Adoption of this mining alternative would decrease pumping costs, thereby decreasing overall production costs.

Use of belt haulage in these mines will also require some modification, but this should not affect overall economics of mine operation. In normal down-dip mining, main haulage belt segments are added as mining progresses. This proposed up-dip development technique, however, would require the entire length of main haulage belt be constructed to the lower limits of the mine prior to initiation of full scale production in that area. Therefore, initial investment in mines with this type of development plan must include all required main belt segments, since they must all be in position before full scale production can begin. Then, as mining proceeds to the rise, main haulage distances decrease and belt segments must be removed. The number of belt segments required would be the same, but the sequence of addition or removal of those segments would be altered. This would cause an increase in haulage and production costs from the newly developing operation, but both costs could be expected to decline as mining progressed. Thus, the average haulage production costs over the life of the mine would not be significantly altered by implementation of this alternate down-dip mining technique.

A possible disadvantage of the technique might be that it takes some time before full production can be realized. In normal mines expanding from a single entry point, many production crews can be simultaneously advancing in different directions. In this alternative technique, however, the long main heading must first be completed; and this generally requires one production unit. Other production crews could not be efficiently utilized until the main haulage heading and conveyor belt were completed. During this time, coal production would also be relatively low. A possible solution to this is development of several main headings, fully utilizing capabilities of all available production crews.

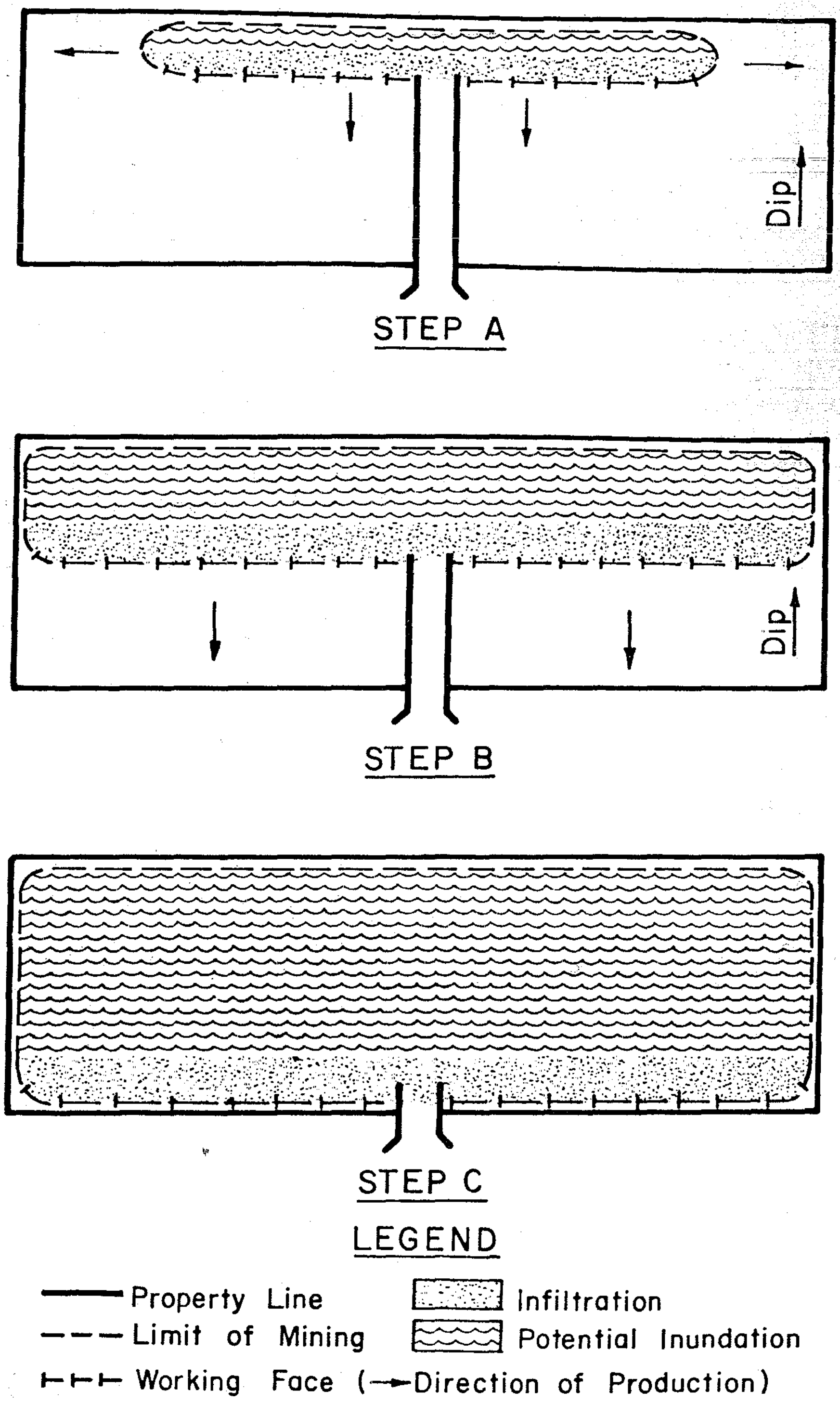
It is also important to note that approximately the same mining alternative technique can be employed in shaft or slope mine to minimize pumping during active operations. The surface elevation of the shaft or slope must, however, be located at a surface elevation greater than the anticipated head that will develop when the abandoned mine is completely inundated. Down-dip headings can be initially developed from the shaft or slope to the anticipated limits of mining. From this point, workings initially expand toward the dip, leaving low lying mined-out areas to inundate naturally and eliminating much of the required pumping. Mining can then proceed toward the rise past the slope or shaft entry to the highest limits of the mine. All drainage will flow toward the lower end of the workings and, if the active workings extend up-dip beyond the shaft or slope, in-mine haulage for that area will be down-grade toward those exit points. Thus, careful mine planning and slope or shaft placement can effectively reduce both haulage and pumping costs for those mines.

Figures 11 and 12 show the primary advantages of use of one of these alternative mining techniques as compared to "standard" down-dip operation.



NORMAL DOWN-DIP MINING PROCEDURE

Figure 11



ALTERNATE DOWN-DIP MINING PROCEDURE

Figure 12

SECTION VII

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SECTION VIII

DEFINITION OF TERMS

Abatement (Mine Drainage Usage) - The lessening of pollution effects of mine drainage.

Acid Mine Drainage - Any acid water draining or flowing on, or having drained or flowed from, any area affected by mining.

Aeration - The act of exposing to the action of air, such as, to mix or charge with air.

Anticline - An upfold or arch of stratified rock in which beds dip in opposite directions from a crest or axis.

Butt - A small section of rooms and pillars, generally an extension of an existing panel, in an underground mine.

Down-dip - In the direction of slope of a non-horizontal coal seam.

Effluent - Any water flowing out of the ground or from an enclosure to the surface flow network.

Heading - A passage or tunnel within an underground mine.

Hydrology - The science that relates to the water systems of the earth.

Infiltration - Water entering the ground water system and, subsequently, underground mine workings through the land surface.

Leaching - Solution of the soluble fraction of a material by flowing water.

Load (Water Quality Usage) - The quantity of material (acid, iron, sulfates, etc.) carried by flowing water in solution - generally expressed in pounds per day.

mg/l - Abbreviation for milligrams per liter, a weight to volume ratio commonly used in water quality analysis. It expresses the weight in milligrams of a substance occurring in one liter of liquid.

Neutralization - The process of adding an acid or alkaline material to waste water to adjust its pH to a neutral position.

Outcrop - The surface exposure of bedrock or strata.

Overburden - Nonsalable material that overlies a mineable mineral.

Oxidation - The removal of electrons from an ion or atom.

Panel - A large block of interconnected rooms and pillars worked either side of a mine's main heading.

pH - Negative logarithm to the base ten of hydrogen ion activity. pH 7 is considered neutral. Above 7 is basic, below 7 is acidic.

Refuse - Rock that has high carbon content - usually referring to dark colored coal mining waste material.

Room - A single block of extracted coal.

Slope - An underground mine entry which slants to a coal seam from the overlying land surface, not from the coal outcrop.

Sludge - Precipitant or settled material from a wastewater.

Slug - Sudden increase in concentration of stream pollutants resulting from heavy rainfall rapidly washing leached pollutants from land surfaces and underground mines.

Stratigraphy - Science of formation, composition, sequence and correlation of stratified rocks.

Up-Dip - In the direction of maximum rise of a non-horizontal coal seam.

Table 9

CONVERSION TABLE

ENGLISH TO METRIC

MULTIPLY (ENGLISH UNITS) by TO OBTAIN (METRIC UNITS)

ENGLISH UNIT	ABBREVIATION	CONVERSION	ABBREVIATION	METRIC UNIT
acre	ac	0.405	ha	hectares
acre - feet	ac ft	1233.5	cu m	cubic meters
British Thermal Unit	BTU	0.252	kg cal	kilogram - calories
British Thermal Unit/pound	BTU/lb	0.555	kg cal/kg	kilogram calories/kilogram
cubic feet/minute	cfm	0.028	cu m/min	cubic meters/minute
cubic feet/second	cfs	1.7	cu m/min	cubic meters/minute
cubic feet	cu ft	0.028	cu m	cubic meters
cubic feet	cu ft	28.32	l	liters
cubic inches	cu in	16.39	cu cm	cubic centimeters
degree Fahrenheit	°F	0.555(°F-32)*	°C	degree Centigrade
feet	ft	0.3048	m	meters
gallon	gal	3.785	l	liters
gallon/minute	gpm	0.0631	l/sec	liters/second
horsepower	hp	0.7457	kw	kilowatts
inches	in	2.54	cm	centimeters
inches of mercury	in Hg	0.03342	atm	atmospheres
pounds	lb	0.454	kg	kilograms
million gallons/day	mgd	3,785	cu m/day	cubic meters/day
mile	mi	1.609	km	kilometer
pound/square inch (gauge)	psig	(0.06805 psig +1)*	atm	atmospheres (absolute)
square feet	sq ft	0.0929	sq m	square meters
square inches	sq in	6.452	sq cm	square centimeters
ton (short)	ton	0.907	kkg	metric ton (1000 kilograms)
yard	yd	0.9144	m	meter

* Actual conversion, not a multiplier

TECHNICAL REPORT DATA

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PLEMENTARY NOTES

TRACT

The report presents detailed results of a feasibility study of down-dip mining, a technique that appears to offer an alternative to sealing or permanent treatment of polluted effluents from coal mines after abandonment. The project included an evaluation of a pair of nearly identical abandoned underground mines - one developed to rise, one developed to dip - to confirm the theory that discharge water quality in down-dip mines is substantially better than that in up-dip mines. An active mine with units operating up-dip and down-dip was also evaluated to ascertain economic and engineering limitations, costs in varying situations, and other major advantages or disadvantages of each mode of operation. Health and safety and National water quality and economic impacts of widespread use versus non-use of the technique were also assessed.

KEY WORDS AND DOCUMENT ANALYSIS

DESCRIPTORS		b. IDENTIFIERS/OPEN ENDED TERMS	c. COSATI Field/Group
r quality mines comparison r pollution waters nd water	Drainage Evaluation	Mining methods	13B

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