RI bureau of mines report of investigations 6326

REMOVING METHANE (DEGASIFICATION) FROM THE POCAHONTAS NO. 4 COALBED IN SOUTHERN WEST VIRGINIA

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UNITED STATES DEPARTMENT OF THE INTERIOR
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UNITED STATES DEPARTMENT OF THE INTERIOR
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REMOVING METHANE (DEGASIFICATION) FROM THE POCAHONTAS NO. 4 COALBED IN SOUTHERN WEST VIRGINIA

by

W. M. Merritts,[1] C. R. Waine,[2] L. P. Mokwa,[3] and M. J. Ackerman[1]

ABSTRACT

Tests were conducted in the Pocahontas No. 4 coalbed in southern West Virginia to determine if successful degasification techniques, developed during an earlier study in the Pittsburgh coalbed, would be effective in removing methane from those usually more gaseous coals.

Specially designed long-hole drilling equipment and Bureau of Mines continuous recording devices were used to measure the quantity and methane content of the ventilating air during the investigation. Infusions were made with water at normal waterline pressures ranging from 200 to 220 psi or at pump pressures ranging from 400 to 650 psi.

Results of the tests showed that methane emissions from free-flow bleeder holes were as much as 131 cfm. Infusions of some of the holes increased the methane emission from other holes and from the exposed coal surfaces near the working areas as much as 1,540 cfm. The methane content in the main return-air currents after infusion was reduced more than 86 percent of the amount recorded before tests were made.

INTRODUCTION

Under a cooperative agreement between the Bureau of Mines and the Olga Coal Co., a field study was undertaken at the Olga No. 2 mine, operated in the Pocahontas No. 4 coalbed in southern West Virginia, to remove methane from virgin coal areas in advance of mining. This study concerns one phase of a broad Bureau program devoted to coal mining research. General opinion is that methane gas is entrapped in voids in the coalbed, in the pores of the coal,

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and in the adjacent strata. In many instances where the strata have been disturbed by geologic movements the gas has escaped from the coalbeds and is confined in pockets and void spaces in the overlying strata, usually under high pressure. Gas emissions during mining are neither regular nor continuous but usually vary with the rate of coal extraction. When methane is contained in crevices under considerable pressure, it may be released during mining in large gas outbursts.

High methane emissions in coal mines create production problems, causing operational downtime of highly productive mining machines, which in turn causes severe decreases in production rates and resulting increases in production costs.

Before continuous mining equipment was used in American mines, underground gas emissions that could not be diluted to a harmless mixture by increasing the volume of ventilating air occurred in only a few ultragassy mines. In areas where emissions were excessive, mining was discontinued until the methane could be diluted to a harmless mixture. Downtime caused by gas accumulations resulted in loss of production in such areas, but often other producing areas of the mine increased production to offset this loss.

The introduction of the multiple purpose continuous mining machines created two additional problems: (1) The rate of extraction was more rapid, usually increasing gas emissions and requiring greater quantities of ventilating air to dilute the gas and (2) the bulk of the continuous mining machines virtually precludes continuous flow of air across the face, where gas may accumulate.

Accumulations of methane in mine workings and subsequent ignition have caused many coal-mine explosions, resulting in loss of life and destruction of property. Unless the gas is removed from the face by some means, the emitted gas must be diluted and rendered harmless by the working face ventilating air current. Therefore, large volumes of air must be circulated in these areas so that the gas can be removed from the mine in a harmless mixture. The problem of maintaining a safe condition at the working faces requires constant monitoring of the methane content of the ventilating air.

If degasification techniques can be applied successfully to remove gas from virgin coal before and during mining, they would decrease operational downtime and minimize the hazards inherent to excessive gas emissions.

A survey of foreign publications showed that practical and economical degasification techniques have been developed in the United Kingdom, Europe, and other major coal-producing countries. A recent report stated that 135 collieries removed more than 17 billion cubic feet of methane in 1958.[4]

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[4] Boyd, W. T., and Harry Perry. Degasification of Coalbeds in Advance of Mining. Pres. 47th Nat. Safety Cong. and Exposition, Chicago, Oct. 20, 1959, Trans. Nat. Safety Cong., v. 7, 1959, pp. 21-34.

Review of the literature indicates that European degasification techniques probably would not be adaptable to existing natural conditions or mining methods used by the U.S. coal industry. Longwall mining is practiced extensively in Europe, while room-and-pillar mining predominates in the United States. In longwall mining the roof subsides as mining progresses, resulting in redistribution of roof pressures, and breaking and cracking of the overlying strata. When the pressure on the longwall face is relieved by the breaking of the overlying strata, much of the gas in the coalbed is released, some coming out of the working face and percolating through cracks in the overlying strata. In room-and-pillar mining, the roof pressure remains fairly stable until the pillars are removed during retreat mining and the gas enters the mine workings, or remains in the coal pillars and immediate adjacent strata. Other differences in mining conditions and practices such as the number of beds worked simultaneously, interval between workable coalbeds, attitude of the beds, and the character of the adjacent strata have a marked effect on the success of all degasification techniques.

An initial study of degasification methods at the Humphrey No. 7 mine, operated in the Pittsburgh coalbed in northern West Virginia was completed and the results published.[5] Because of the encouraging results of these tests, it was decided to make a similar study in the Pocahontas No. 4 coalbed in southern West Virginia.

ACKNOWLEDGMENTS

The cooperation and assistance of the following mine officials of the Olga No. 2 mine are gratefully acknowledged: D. C. Ridenour, general manager; Homer Hickam, general superintendent; Adam Robinett, chief mining and industrial engineer; Clarence Hickam, general mine foreman; R. W. Wotring, safety director; and C. A. Beasley, mining engineer.

DEGASIFICATION TEST SITE

Location

The 5 Left section of the Olga No. 2 mine, a peninsular area of virgin coal, Olga Coal Co., Caretta, McDowell County, West Va., was selected as a study site. At this mine irregular peninsula-shaped areas of coal are nearly cut off from the main coalbed by want areas of shale and sandstone, and usually these areas are very gassy.

This area was partly mined before the degasification study was begun, but considerable operating-time delays were experienced because of high-gas emissions. Consequently the section was idled for about 6 months to allow gas to drain from the face areas. When mining was resumed, large gas emissions again caused operating delays, and soon thereafter the degasification work was begun.

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[5]Merritts, W. M., W. N. Poundstone, and B. A. Light. Removing Methane (Degasification) From the Pittsburgh Coalbed in Northern West Virginia. BuMines Rept. of Inv. 5977, 1962, 39 pp.

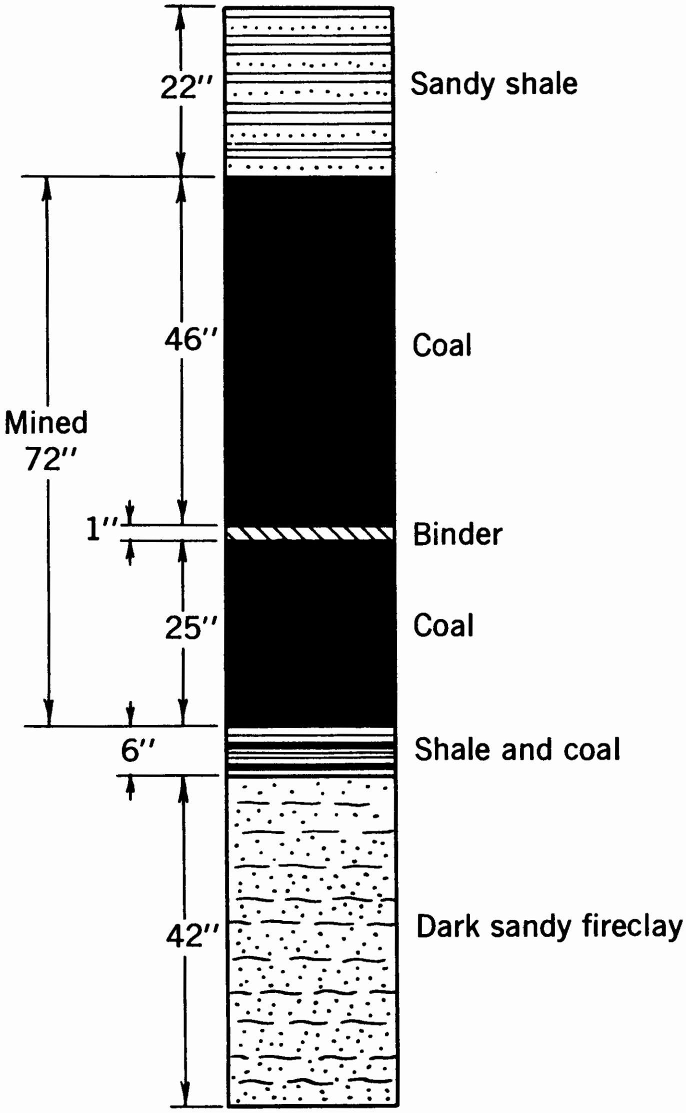

FIGURE 1. - Section of Pocahontas No. 4 Coalbed at Olga No. 2 Mine.

Coalbed

The Pocahontas No. 4 coalbed occurs in the Pottsville series of the Pennsylvanian period. It extends over an area of 155 square miles in Wyoming and McDowell Counties, West Va., where it is about 28 feet below the Pocahontas No. 5 and 85 feet above the Pocahontas No. 3 coalbed.

This coalbed is about 72 inches thick and has a slabby structure at the Olga No. 2 mine, where it lies about 600 feet below the surface in the valleys and approximately 1,600 feet below the tops of the mountains. The coal is soft, friable, and tends to slough from the ribs after being exposed to air. A 1/2- to 1-inch binder occurs in the bed from 12 to 46 inches below the roof. Figure 1 shows a typical columnar section of the immediate roof, the coalbed, and the bottom in a minable area.

In want areas the coal may be very thin or entirely replaced by shales and sandstones. Figure 2 shows a columnar section of the immediate roof, the coalbed, and the bottom in a want area. The main roof usually is sandy shale but may be laminated and contain coal partings near the want areas.

The floor is not affected by water but will break during development and will heave excessively during pillar extraction in some areas.

A typical analysis of coal at this mine is as follows:

Moisture	percent	1.5
Volatile matter	do...	16.0
Fixed carbon	do...	77.5
Ash	do...	5.0
Sulfur	do...	0.5
Calorific value	Btu per pound	14,600

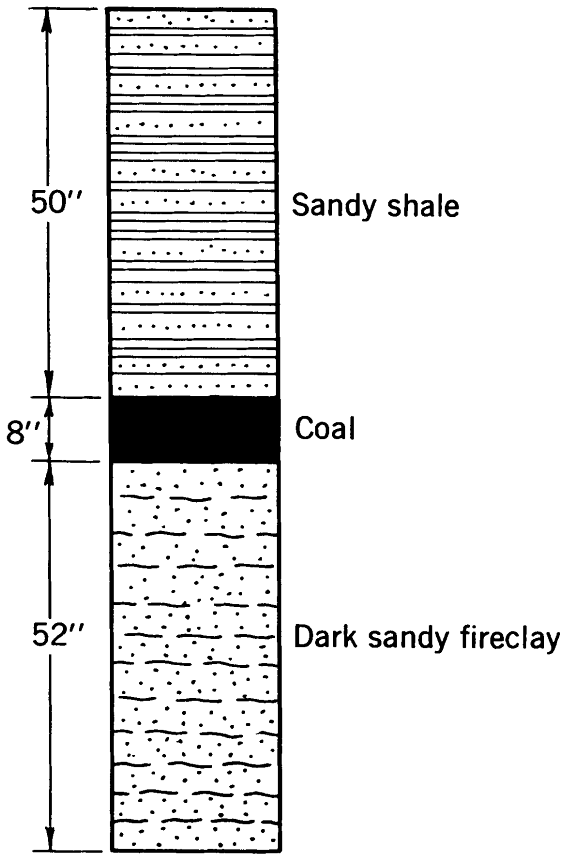

FIGURE 2. - Section of Pocahontas No. 4 Coalbed in a Want Area at Olga No. 2 Mine.

Mine Development

The coalbed was entered through five shafts, each approximately 680 feet deep, and a multiple-entry block system was used to develop the coalbed. Main entries were developed in sets of 8 to 12 except in areas between wants where fewer were driven. Entries were driven about 16 feet wide, on 90-foot centers; crosscuts were driven on 80-foot centers, and pillars were extracted on retreat.

The 5 Left section was developed by either five or seven entries (fig. 3). After the main entries were advanced to the want area, a set of five entries, designated as Right Side entries, were driven to the right from entry 1 of the 5 Left Main entries. When the Right Side entries were extended to the want area, a set of five entries, designated as Left Side entries, were driven to the left from entry 5 of the 5 Left Main entries. After the Left Side entries were developed to the want area, five rooms were driven to the left from entry 1 of the Left Side entries. After the rooms were driven 160 feet, pillar extraction was begun.

One ripper-type continuous mining machine mined the coal, loaded it directly into cable-reel shuttle cars that transported the coal to a ramp, and then discharged it into 10-ton steel mine cars. Trolley locomotives moved the mine cars to the shaft bottom, where self-dumping skips hoisted the coal to the surface.

During development, the roof was supported by 4-foot-long roof bolts on 4-foot centers across the face and in the direction of advance with rib bolts 2 feet from each rib. Percussion drills were used to drill the roof-bolt holes. After the continuous miner advanced 20 feet, it was moved to another working place while roof bolts were installed.

During pillar mining, posts with crossbars were set on 5-foot centers to support the roof instead of using roof bolts. After a pillar cut was mined, the roof was supported temporarily by two rows of posts on 4-foot centers across the face, with two or more steel safety jacks between the last row of posts and the face.

The mine was classed gassy by the West Virginia Department of Mines and, at the time of the study, was liberating about 6 million cubic feet of methane

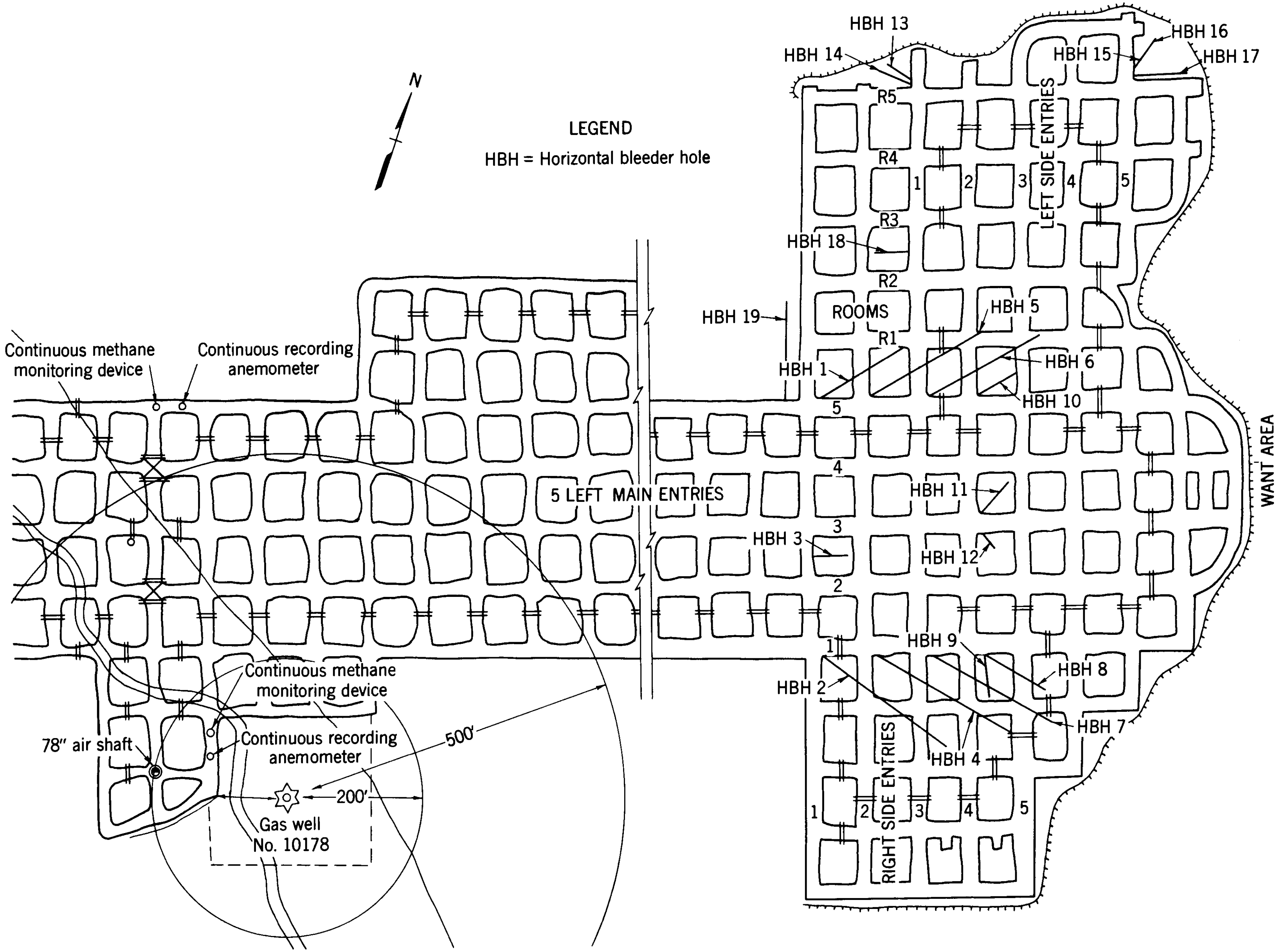

FIGURE 3. - Plan of 5 Left Section Where Degasification Tests Were Conducted.

per day. About 872,000 cfm of air was circulated through the mine by two exhaust fans and two forced-air fans.

A split system of ventilation was used to deliver 9,300 to 50,000 cfm of air to the last crosscut in each section and from 28,000 to 75,000 cfm of air to the intake end of the pillar lines.

In the 5 Left section, 120,000 cfm of air was available for ventilation. Intake air was coursed through the center entries, split near the active faces, and returned through the outside entries to a 78-inch-diameter upcast shaft (borehole) shown in figure 3. This shaft was 700 feet deep and equipped on top with an exhaust fan.

Air currents were directed to the working faces by means of brattice cloth faced with thin sheets of plastic on the high-pressure side of the cloth to minimize leakage. By installing line brattices on the return side instead of the intake side of the working places, the input of ventilating air at the faces was disturbed less frequently, as mining equipment generally could travel to and from the working places without passing under a line brattice, or check curtain.

Figure 4 shows the ventilating system, general arrangement of mining equipment, and sequence of 36 cuts for advancing entries before degasification tests. At the time degasification tests were begun, alternate entries 1, 3, and 5 were being driven 80 feet and entries 2 and 4 were advanced in sequence with crosscuts to connect all entries. The more desirable practice of advancing entries consecutively had been discontinued, because too many operational delays had occurred due to excessive gas liberations at the faces.

While greater production normally was realized when entries were driven consecutively than when alternate entries were driven, production by the latter plan increased, because downtime to dilute gas was reduced appreciably. From the method used, the following advantages were obtained: A limited number of places were worked at one time; the amount of ventilating air at the working faces could be controlled more effectively because line brattices and check curtains were disturbed less frequently; and gas emissions at the faces could be diluted more readily.

PREVIOUS DEGASIFICATION

Previous degasification at this mine was limited to (1) the normal drainage of methane from large blocks of coal outlined by bleeder entries, and (2) in very gassy sections, drainage from vertical boreholes, ranging from 6 to 78 inches in diameter, drilled from the surface to the entries and used later for ventilating upcast shafts.[6]

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[6]Ridenour, D. C. Core Drilling of Air Shafts and Manway Portals. Min. Cong. J., October 1957, pp. 99-101.

_____. Ventilation by Large-Diameter Boreholes. Paper pres. at joint meetings of West Virginia Coal Min. Inst. and Central Appalachian Section, AIME, Nov. 1, 1958. Mechanization, v. 23, No. 3, March 1959, pp. 69-70.

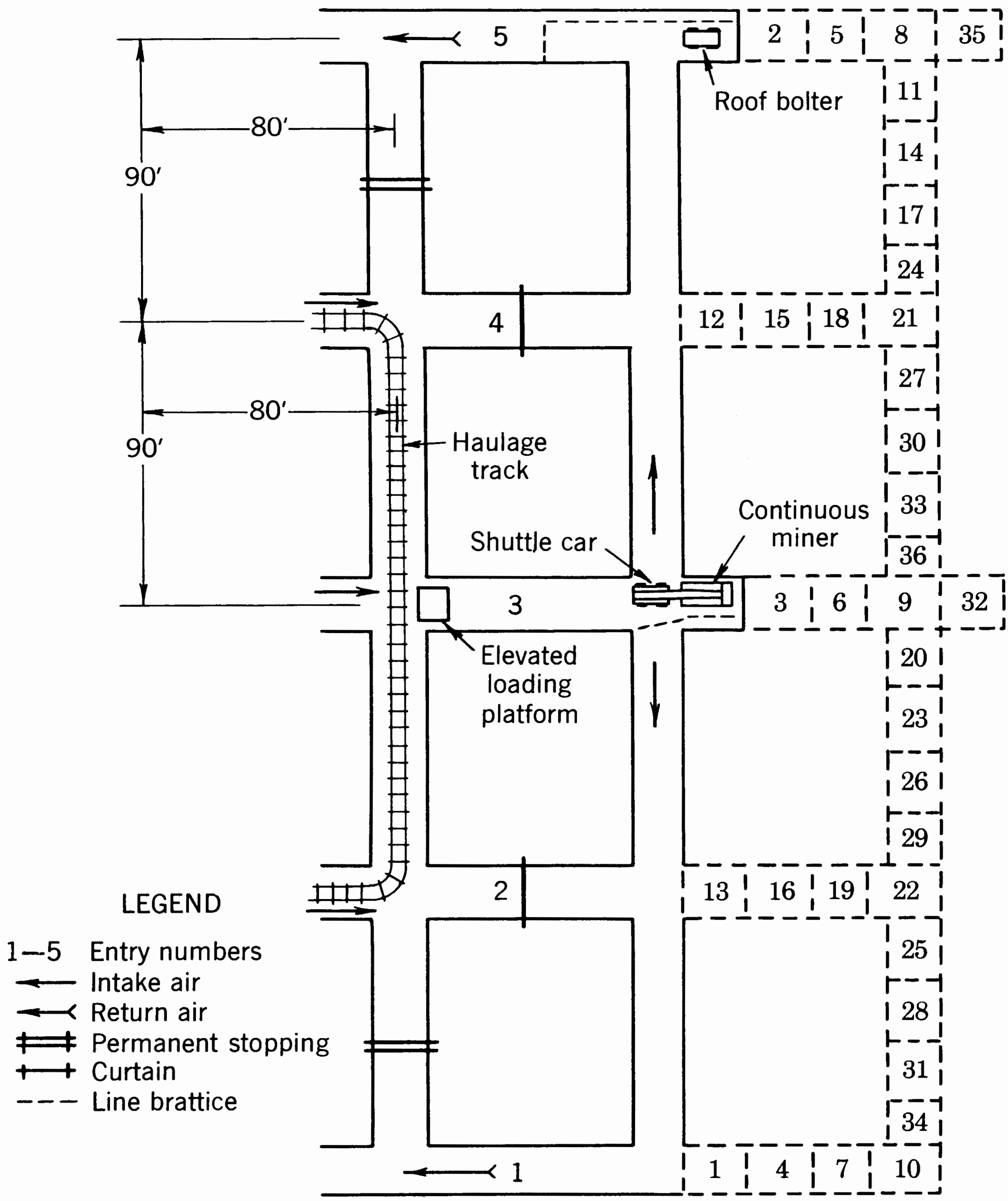
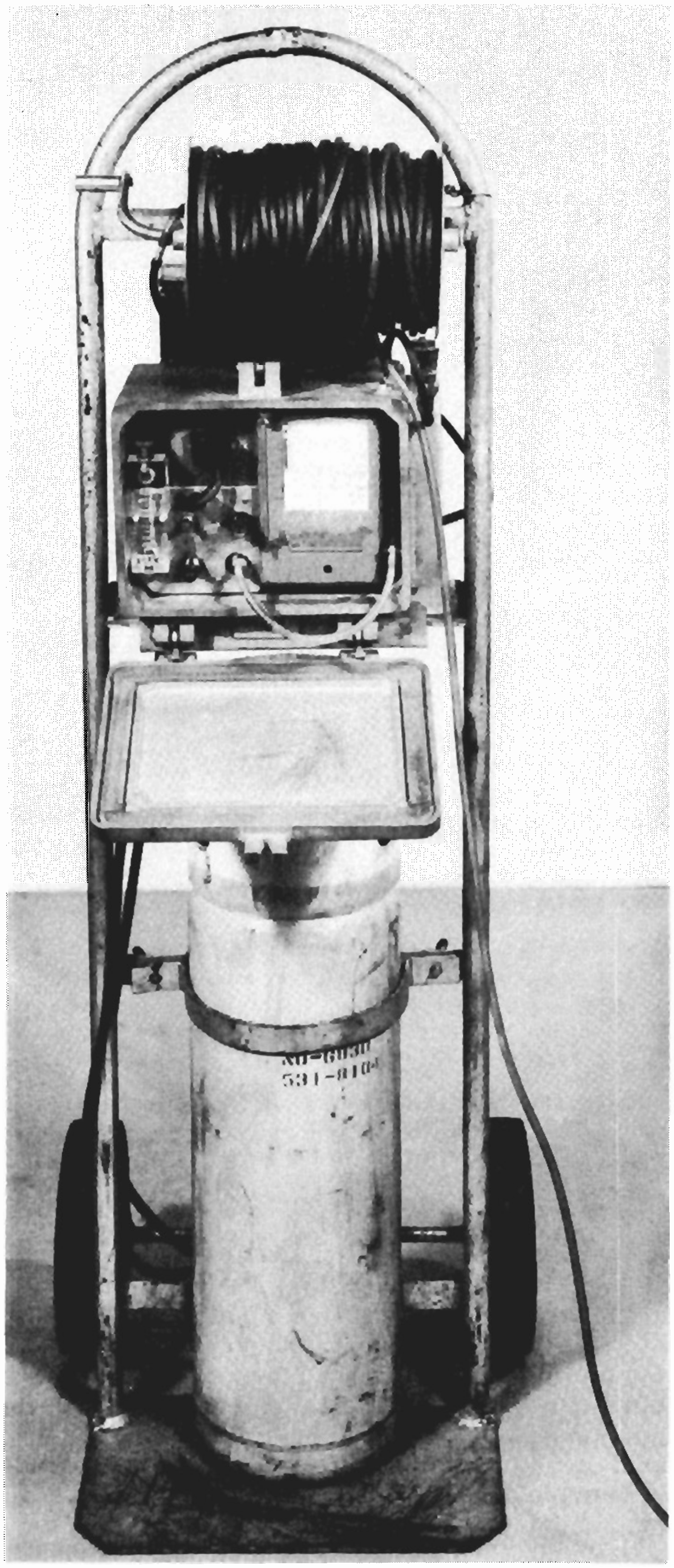


FIGURE 4. - Sequence of Cuts to Advance a Set of Five Entries Before Degasification.



By using a force system of ventilation, additional ventilation was obtained in highly gaseous areas and mixtures were coursed directly to the surface through boreholes, thereby reducing the quantity of gas and air moved in the main return airways. These methods, however, did not effectively reduce methane liberations during subsequent mining. When the blocks of coal were mined, methane liberations at the faces still were high and mining operations frequently were stopped to permit the ventilating air to dilute the methane to a safe concentration.

DEGASIFICATION EXPERIMENTS

Free-Flow Drainage by Horizontal Bleeder Holes

During the degasification study, long horizontal bleeder holes were drilled into the coalbed in advance of the working faces to determine the amount of methane that could be released from the holes by free flow.

Equipment

Drill

A hand-held pneumatic coal drill capable of drilling long holes rapidly was used to bore the holes. The drill stem consisted of a 10-foot fluted auger having a 2-inch-diameter double-prong bit followed by hollow 10-foot pipe extensions through which water flowed to flush cuttings from the hole.

FIGURE 5. - Bureau of Mines Continuous Methane Monitoring Device.

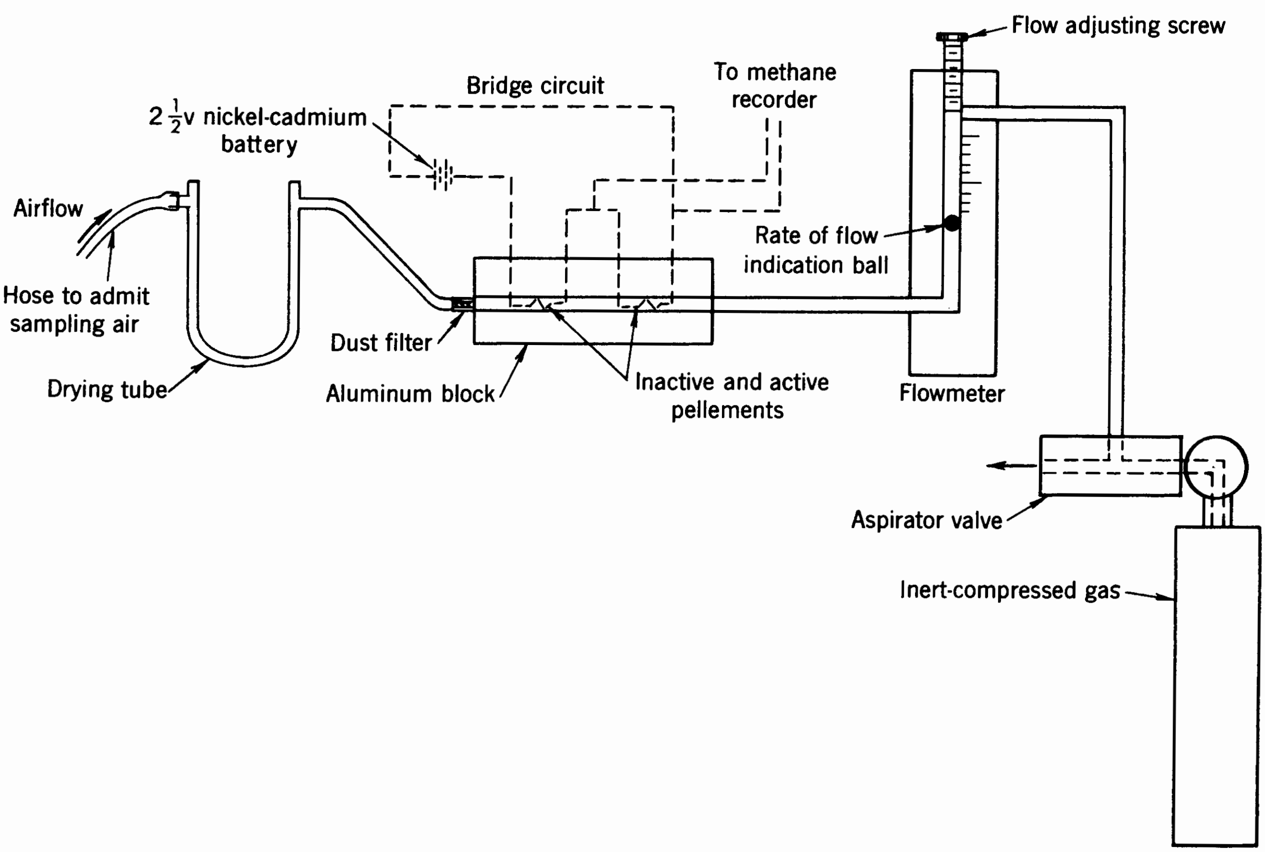


FIGURE 6. - Flow of Air Samples Through Continuous Methane Monitoring Device.

Continuous Methane Recorders

Methane concentrations in the return-air currents were recorded on continuous methane analyzing and recording devices. These continuous recording units were designed and built by Bureau engineers. Figure 5 shows an overall view of one of the methanometers.

The recording unit is battery powered. Air samples are pulled through flexible rubber tubing to the methanometer by the vacuum resulting from the controlled expansion of dichlorodifluoromethane from a cylinder. Analysis and recording of the methane concentration was continuous.

The detecting principle of the instrument depends on the change in resistance resulting from the combustion of the methane by an electrically heated catalytic filament imbalancing a bridge circuit. Two filaments (pellements) are combined in a wheatstone-bridge circuit in the methane detector (fig. 6). One pellement is active and one inactive. The inactive pellement is identical with the active one except that it is not impregnated with a catalyst. Upon application of current to the bridge circuit both pellements heat to the same temperature. In the presence of methane the temperature of the active pellement increases and produces a change in resistance

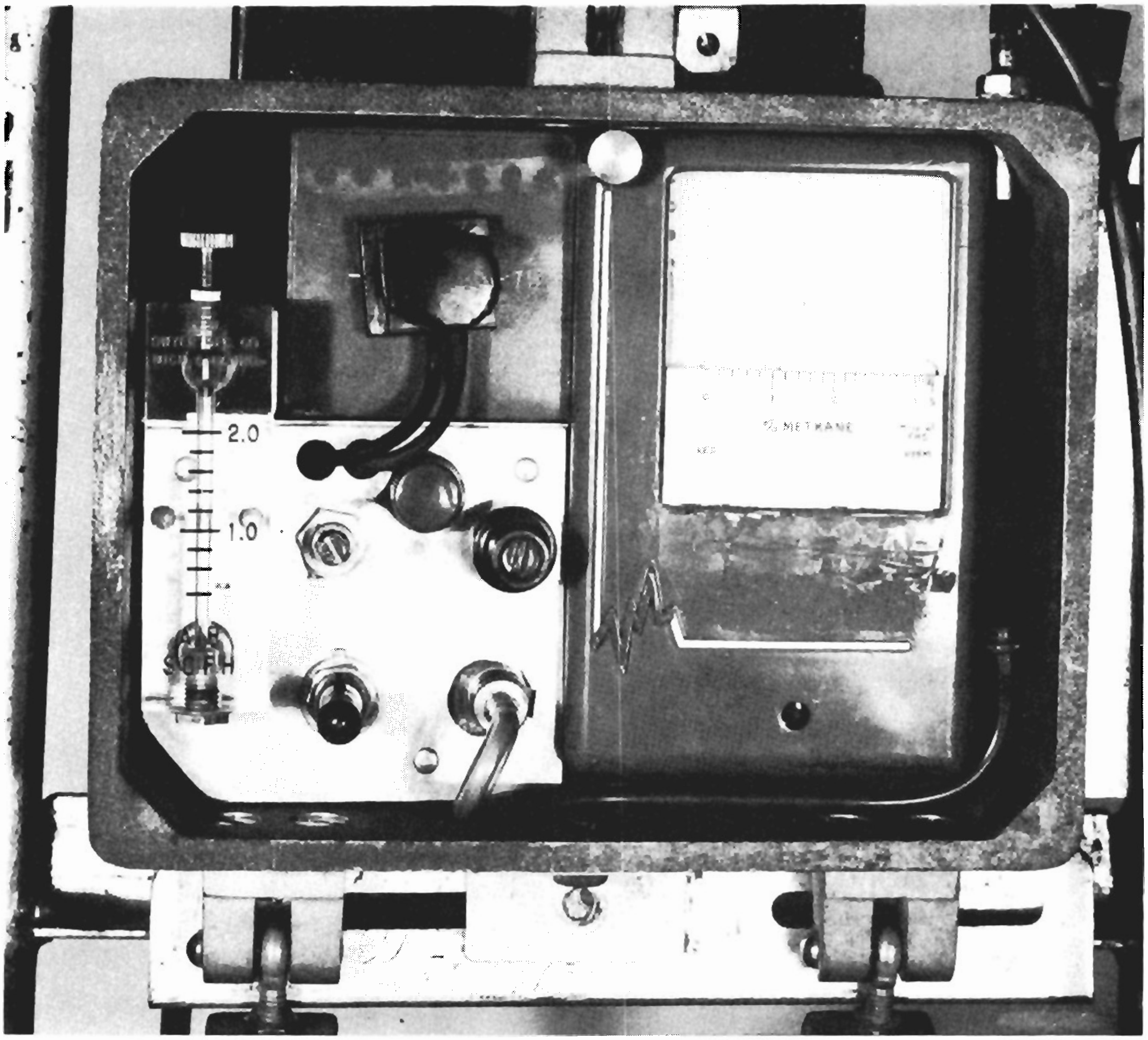


FIGURE 7. - Methane Analyzing and Recording Units.

which imbalances the bridge circuit. The meter of the recording unit responds directly to this change.[7] Methane percentage is recorded on a carbon-backed chart at approximately 3-second intervals. Figure 7 shows a closeup of the instrument.

The source of power for the bridge circuit is a 2-1/2-volt rechargeable nickel-cadmium battery. A voltmeter and a variable resistor control the voltage in the bridge circuit. The energy for powering the chart drive of the recording section of the methane monitoring device is provided by a 7-1/2-volt dry-cell battery.

[7] Zellers, D. H. Developments in Methane Monitoring. Paper pres. 49th Nat. Safety Cong., Chicago, Oct. 16 to 18, 1961, 9 pp.

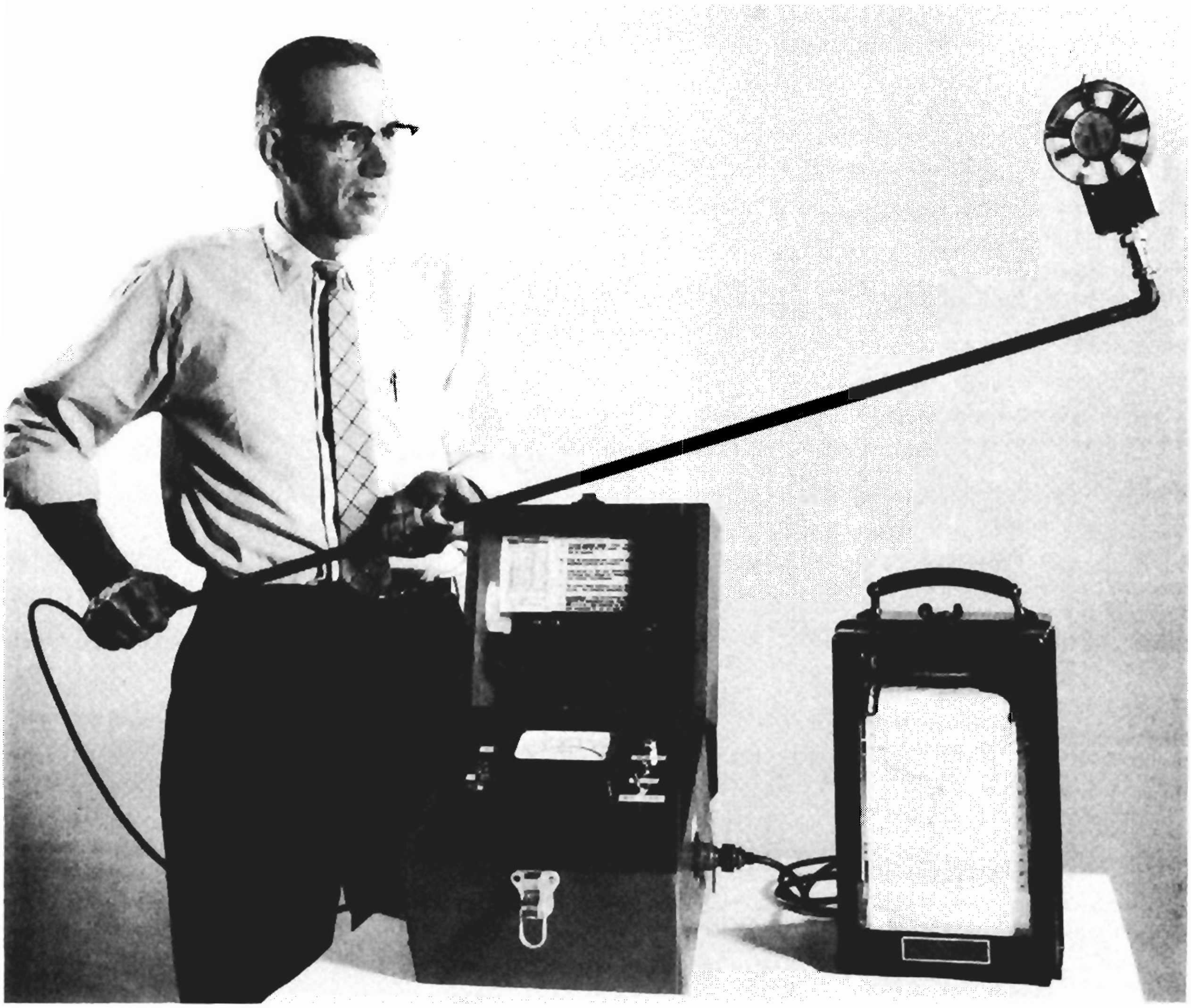


FIGURE 8. - Bureau of Mines Continuous Recording Anemometer.

Recording Anemometers

Bureau of Mines recording anemometers were used to record continuously the velocities of the air currents in the main returns of the 5 Left section. One of these velocity recording devices is shown in figure 8. The anemometer is mounted on the end of a 6-foot length of rigid copper tubing. Its vanes, when rotating in an air current, intersect a beam of light emitting from a light bulb set in the housing of the anemometer. The number of intersections, depending on the velocity of the air current around the anemometer are transmitted as electric impulses to a tachometer. Velocity of the air current in feet per minute can be read on the velocity meter and/or recorded on a chart for a permanent record. Two 6-volt dry cells and a 1-1/2-volt battery are used as the source of power for the tachometer and the bulb, respectively.

Miscellaneous

In addition to the units described, additional equipment such as portable M.S.A. W-8 and Riken[8] methane detectors, oxygen and carbon dioxide analyzers, permissible flame safety lamps, barometers, thermometers, psychrometers, pitot tubes, and volumeters were used.

Procedure

In previous Bureau degasification studies, more methane was liberated from the ribs and faces of outside entries than from other entries when mining in virgin coal. Therefore, initial tests consisted of drilling long bleeder holes into the solid rib of the face of each outside entry of the 5 Left Main entries under development and measuring the gas emissions from the holes. The initial test area was about 2,400 feet inby the upcast ventilation shaft.

All drilling was done on the intake air end of the line brattice so that any gas released would flow directly into the returns. Two-inch-diameter holes were drilled to depths ranging from 30 to 230 feet at various angles to the direction of the headings. The holes were positioned so that the gas would be emitted directly into the returns when the crosscuts connecting the entries were driven. In addition to rib holes, several holes were drilled in advance of the center entry.

Pressures, velocities, and methane concentration of the gas were measured to record the gas emissions from the holes. At selected places in the immediate return-air courses, air velocities and entry dimensions were taken to compute the volume of ventilating air. Methane percentages were read simultaneously on a Riken or on M.S.A. W-8 methane detector, and the quantity of methane in the air was computed.

The quantity and quality of the air coursing through the main return airways were determined from data recorded by continuous methane recorders and recording anemometers. The location of these instruments in the main return airways is indicated by figure 3. The anemometers were positioned in the vertical cross-sectional plane of the airways at the point of average velocity determined by anemometer traverses.

Results

At the end of the last operating shift and 2 days before the beginning of the degasification study, the amount of methane in the immediate return-air currents was 800 cfm and in the main return-air currents it was 2,000 cfm.

While conducting the degasification tests, 19 horizontal bleeder holes were drilled during the development of 9,000 feet of entries. All holes except two emitted gas during drilling operations and continued to do so for

[8]Reference to specific models of equipment mentioned in this publication is made to facilitate understanding and does not imply endorsement by the Bureau of Mines.

varied lengths of time. Shortly after completion of drilling a hole, the velocity and percentage of gas issuing from it was measured and the rate of flow determined; the shut-in pressure also was measured. As expected, variations occurred in the drilling time, the quantity of gas liberated during drilling, the shut-in pressures, and the total time gas issued from the holes. Generally, emission of gas increased with the depth of hole during drilling, but very little gas was detected in the first 10 feet.

Table 1 summarizes the actual drilling time, and the amount of methane released into the immediate return-air courses while drilling holes 1 and 2. Holes 1 and 2 were drilled when the mine was idle and 2 days after the last operating shift. Immediately before drilling, the quantity of methane in the immediate return-air currents was 480 cfm and in the main return-air currents was 1,270 cfm. When hole 1 was completed at the face of entry 5, the amount of gas emitted into the right immediate return air remained unchanged until drilling began at hole 2. One hour was required to transport the drilling equipment from hole 1 to the location of hole 2 and to prepare for drilling hole 2.

TABLE 1. - Drilling time and methane released into immediate returns while drilling holes 1 and 2

Hole	Depth of hole, feet	Elapsed drilling time, min	Methane released, cfm[1]		Remarks
			Left return	Right return	
1.........	-	-	264	216	Start drilling.
	-	5	-	216	-
	-	35	326	-	-
	155	45	-	-	Finish drilling.
	155	60	342	216	-
2.........	-	-	-	216	Start drilling.
	-	15	-	328	-
	230	40	-	[2]635	Finish drilling.
	230	70	-	376	-
	230	85	326	-	-

[1]Methane released includes emission from drill holes plus normal emission.
[2]Gush of methane from hole 2 when drill stems and auger were removed.

Table 2 summarizes the actual drilling time and the amount of methane released into the main return-air courses during the drilling of holes 13 through 17.

Table 3 is a summary of pertinent data for all the horizontal bleeder holes. On completion of holes 2 and 7, the velocity of gas was 6,000 fpm, and each of the two holes emitted 131 cfm of gas (188,600 cubic feet per day). Despite considerable leakage through cracks in the coal at the collar of the hole, hole 2 had a shut-in pressure of 63 psi. The shut-in pressure of hole 7 was only 5.8 psi. The data show that the greatest amount of methane was released from rib holes in the outside entries.

TABLE 2. - Drilling time and methane released into main returns while drilling holes 13 through 17

Hole	Depth of hole, feet	Drilling time, min	Methane released, cfm[1]		Remarks
			Left return	Right return	
13..........	-	-	326	167	Start drilling.
	40	25	[2]326	-	Finish drilling.
14..........	-	-	326	-	Start drilling.
	60	38	339	-	Finish drilling.
15..........	-	-	-	167	Start drilling.
	40	28	-	[3]167	Finish drilling.
16..........	-	-	-	167	Start drilling.
	70	34	-	190	Finish drilling.
17..........	-	-	-	190	Start drilling.
	82	41	339	226	Finish drilling.

[1]Methane released includes emission from drill holes plus normal emission.
[2]Methane emitted from hole 13 after drilling was only 0.4 cfm.
[3]No perceptible emission of gas from hole 15.

When a bleeder hole was sealed near its collar, and provided no mining was done in the immediate area, the time required for the gas to reach maximum shut-in pressure ranged from several minutes to several hours. After maximum pressure was reached, it was possible to open the seal, bleed off the gas, reseal the hole, and again reach the previous shut-in pressure. A pressure gage connected to the seal in the hole fluctuated continuously until the maximum pressure was reached; these fluctuations were more pronounced where there was less leakage around the seal.

Generally, gas emissions from bleeder holes immediately after drilling increased the amount of methane in the return air above the amount before drilling, and the increase was as much as 200 cfm. However, in a few exceptional instances, methane was emitted in sporadic gushes up to 400 cfm immediately after drilling.

Methane concentrations in the immediate return-air currents were measured with portable Riken and M.S.A. W-8 methane detectors. The readings indicated that occasionally, after bleeder holes were drilled at the faces of the outside entries at a projected crosscut, a slight reduction in the amount of methane released was realized at the working faces during mining operations. This reduction in methane liberation at the face area, however, was only temporary and the methane content in the face ventilating air gradually increased as entries advanced inby the crosscut. After entries were driven about 40 feet beyond the crosscut or one-half the distance to the next crosscut, the amount of methane in the air again approached the amount recorded before the holes were drilled. Consequently, a sustained reduction of methane releases at the working faces was not realized. The methane concentration readings in the immediate return air reflected the changes in methane emissions at the working faces; the methane concentration readings in the main return air included not only the face gas emissions, but also gas emissions along the solid ribs between the sampling points in the immediate and main returns.

TABLE 3. - Summary of free-flow drainage data from horizontal bleeder holes

Hole	Entry	Depth of hole, feet	Angle of hole with entry, degrees	Drilling time, min	Location of face inby drill hole, feet	Time interval after drilling, days	Velocity of gas, fpm	Shut-in pressure, psi	Methane concentration, percent	Volume of methane, cfm	Remarks
1	[1] 5,5L	155	30	42	-	-	2,700	40.0	80.0	47.0	Terminated in rock.
					440	55	125	0.0	98.5	3.0	
					500	60	125	0.0	97.0	3.0	
2	1,5L	230	31	40	-	-	6,000	63.0	100.0	131.0	-
					440	55	275	0.1	100.0	6.0	
					480	60	250	0.0	98.0	5.0	
3	[2] 2-3,5L	50	0	20	-	-	500	0.1	100.0	11.0	Terminated in rock.
4	1,5L	230	30	90	-	-	4,380	5.0	100.0	96.0	-
					380	34	175	0.0	100.0	4.0	
					480	47	175	0.0	99.1	4.0	
5	5,5L	185	32	75	-	-	2,830	2.5	100.0	62.0	Terminated in rock.
					360	34	1,000	0.0	99.2	22.0	
					420	47	1,000	0.0	99.1	22.0	
6	5,5L	190	33	75	-	-	3,000	9.0	100.0	65.0	Do.
					280	29	250	0.0	99.0	5.0	
					390	42	175	0.0	98.7	4.0	
7	1,5L	210	34	70	-	-	6,000	5.8	100.0	131.0	Do.
					290	29	125	0.0	99.6	3.0	
					300	42	100	0.0	99.4	2.0	
8	1,5L	115	33	35	-	-	300	0.1	100.0	7.0	Do.
					210	27	0	0.0	26.0	0.0	
					320	40	0	0.0	28.9	0.0	

9	1,5L	63	90	15	- 210 320	- 27 40	0 0 0	0.0 0.0 0.0	0.0 88.8 44.1	0.0 0.0 0.0	Do.
10	5,5L	65	22	45	- 190 399	- 27 49	700 100 100	0.5 0.0 0.0	100.0 98.8 98.6	15.0 2.0 2.0	Do.
11	3,5L	60	45	20	-	-	1,000	5.0	70.0	15.0	-
12	3,5L	30	45	15	-	-	400	0.3	100.0	9.0	Terminated in rock.
13	[3]1,L.S.	40	75	25	-	-	100	0.0	20.0	0.4	Do.
14	1,L.S.	60	85	38	-	-	600	0.1	98.0	13.0	Do.
15	5,L.S.	40	20	28	- 70	- 5	0 0	0.0 0.0	65.0 40.0	0.0 0.0	Do.
16	5,L.S.	70	20	34	- 70	- 5	1,500 1,000	2.0 0.4	98.0 100.0	33.0 22.0	Do.
17	5,L.S.	82	90	41	7 77	- 5	1,200 800	4.3 2.0	98.0 100.0	26.0 17.0	Do.
18	[4]1,L.S.	50	90	30	-	-	50	0.0	9.0	0.2	Do.
19	5,5L	145	90	50	-	-	400	2.0	100.0	9.0	Do.

[1] Entry 5, 5 Left.
[2] In crosscut between projected entries 2 and 3, 5 Left.
[3] Entry 1, Left Side.
[4] Entry 1 between projected rooms 2 and 3, Left Side.

The long horizontal bleeder holes were drilled at various times while developing the 5 Left Main entries and the Left Side entries. Some of these bleeder holes were used also for water-infusion purposes before all holes were drilled.

Water Infusion of Horizontal Bleeder Holes

Water infusions were made in several of the bleeder holes. The injected water probably infiltrated the natural crevices or cleat planes within the coalbed and displaced the gas. Initial water infusion studies by the Bureau were made to control air-borne dust by increasing the surface moisture of the coal in advance of mining. These studies showed that in addition to allaying dust, water infusion also partially degasified the coal in place at the working faces.[9]

A later study on the removal of gas from the Pittsburgh coalbed before mining showed that infusions of gas bleeder holes proved effective in increasing the release of gas from the solid coal above that released by free flow. Because of these results, tests were made to determine the effect of water infusion on the release of gas from solid coal in the Pocahontas No. 4 coalbed.

Water used to infuse the coalbed for degasification tests was obtained from an underground stream encountered about 100 feet below the surface while drilling the ventilating shaft in the 5 Left section. The water was collected by a water ring in the shaft and piped to the working places through 2-inch and 3/4-inch pipelines.

At prearranged connection points along the 3/4-inch pipeline, 50-foot lengths of 3/4-inch high-pressure hose were connected to obtain water for water-infusion experiments. The static waterline pressure at the working area, because of the difference in elevation between the source of water and the face region, was about 220 psi. Normal waterline pressures ranged from 200 to 220 psi. Waterflow during infusions was measured with a flowmeter.

Equipment

A high-pressure pump was used to produce pressures above the normal waterline pressures. The pump was a triplex-plunger type capable of delivering up to 35 gpm of water at a pressure of 400 psi. Pressure adjustment could be made to obtain pressures up to 650 psi. Figure 9 shows the general arrangement of the water pipelines at the pump site. This arrangement permitted the water to flow through the pump to the location where it was to be used at normal waterline pressures when the pump was not in operation.

Figure 10 shows details of a water-infusion seal. When the seal was inserted in a hole and the inby nut tightened, the 1-inch pipe was moved toward the collar of the hole. This compressed the rubber washers, causing

[9] Jackson, E. O., and W. M. Merritts. Water Infusion of Coal Pillars Before Mining, Independent Coal & Coke Co., Kenilworth, Utah. BuMines Rept. of Inv. 4836, 1951, 25 pp.

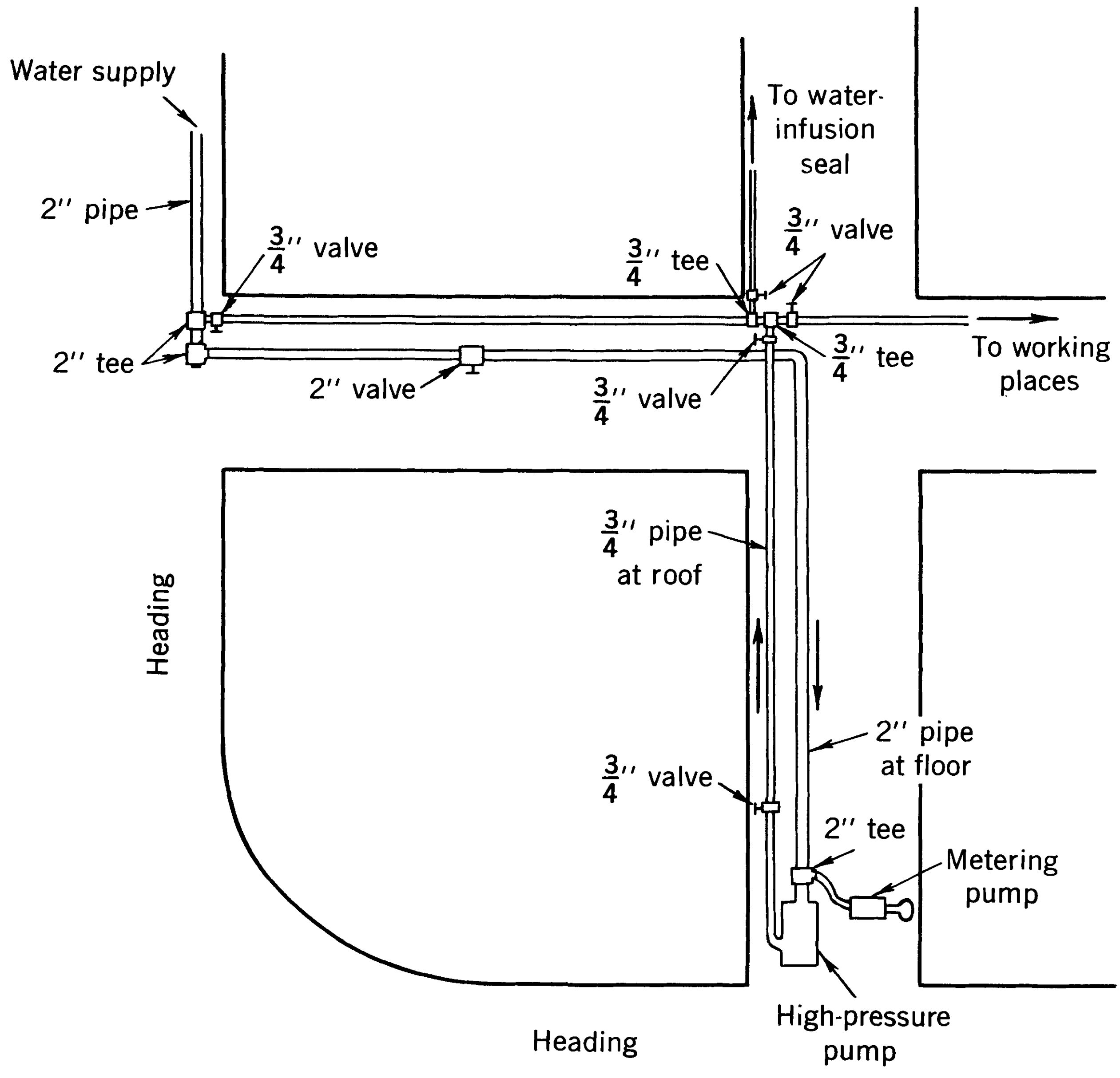


FIGURE 9. - Plan of Pipeline Installation at High-Pressure Pump.

them to expand and grip the wall of the hole firmly, making a watertight seal. The outby nuts acted as jam nuts to keep the 1/2-inch pipe from turning when the inby nut was turned to release the rubber washers for removal of the seal.

In some water-infusion tests a wetting agent was added to increase the penetration effect of the water solution. Wetting agents reduce the interfacial tension between the solid and the liquid. The liquid wetting agent was injected into the infusion water by means of a constant-volume gear pump at the rate of 144 cc per minute.

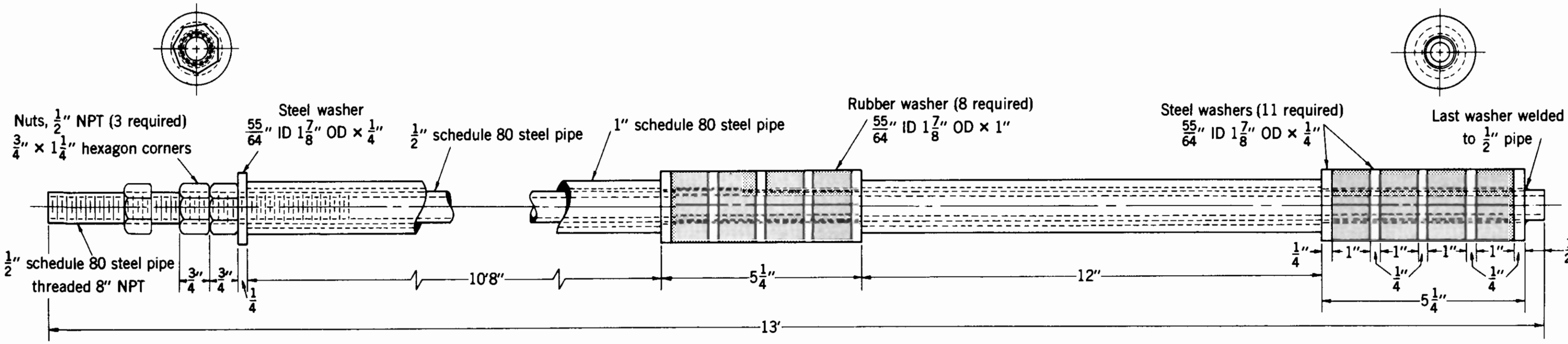


FIGURE 10. - Water-Infusion Seal.

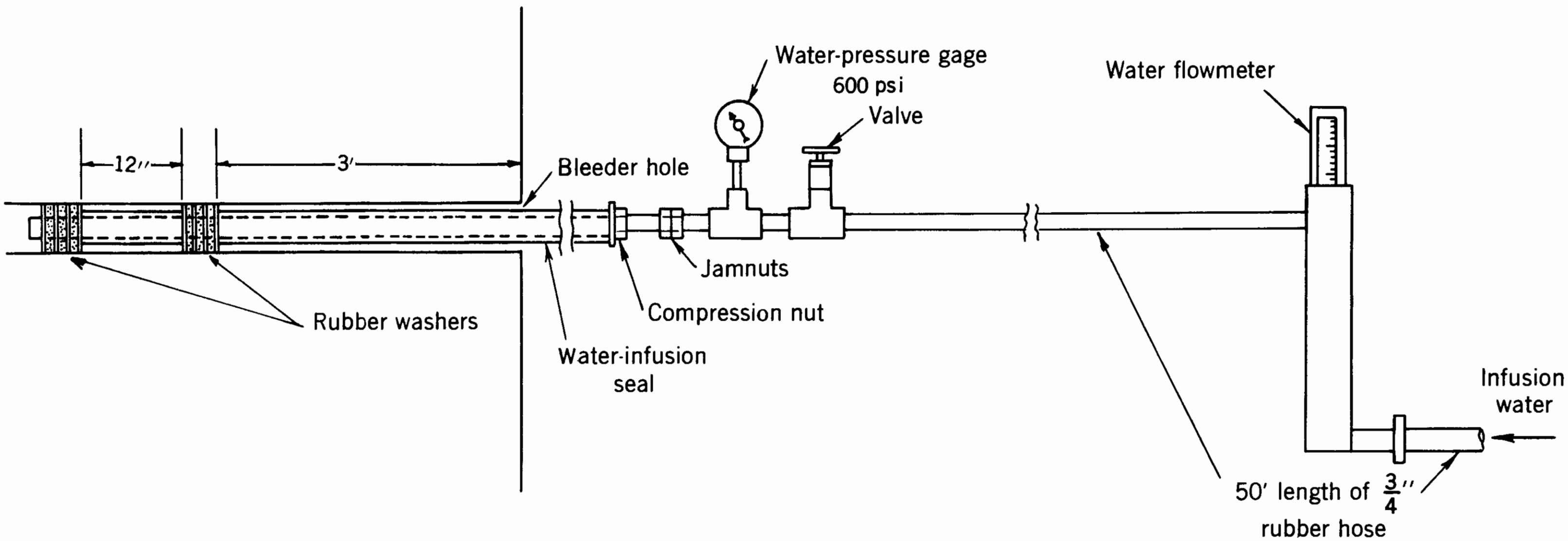


FIGURE 11. - Arrangement of Equipment Used to Infuse a Bleeder Hole.

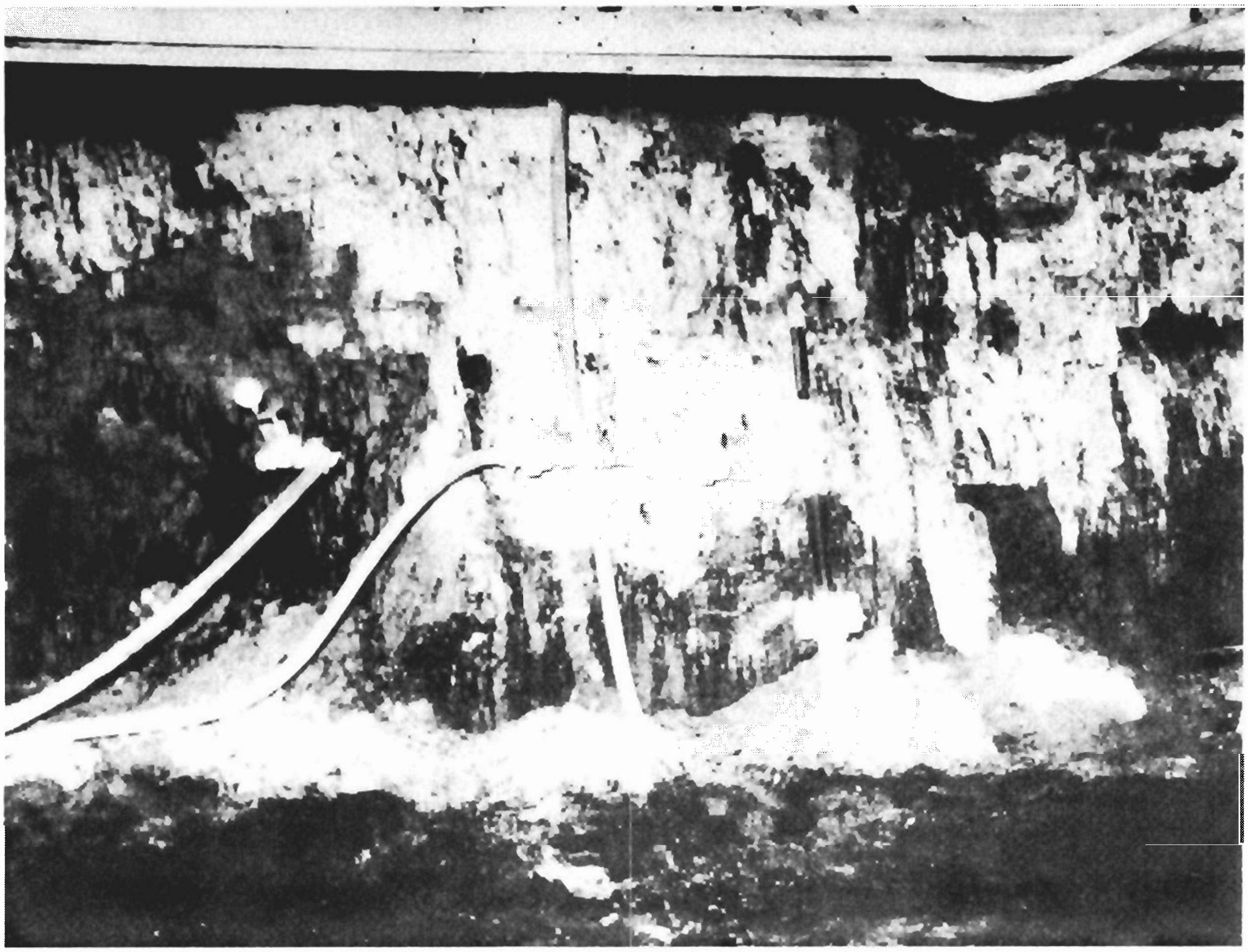


FIGURE 12. - Dark Areas Are Wet Coal Surfaces Caused by Water Oozing From Coal During Infusion.

Procedure

Some of the bleeder holes were infused with water under pressure to determine the effect of emission of gas from the face and ribs compared with free flow. Figure 11 shows a typical arrangement of the equipment used to infuse a bleeder hole.

The water infusion seal usually was set about 3 feet in the hole. If the coal at the collar of the hole was badly cracked permitting the water to leak past the seal, the seal was set deeper in the hole.

The path and distribution of the water injected into the hole were traced by noting small gas-water bubbles appearing on coal surfaces at the face and gradually spreading to adjacent faces and ribs, fig. 12. Occasionally, fluorescent dye with alkali added to make the dye more soluble in water was injected into the infusion water. The progress of the infused water was traced by means of an ultraviolet light approved for use in gassy mines.

The following data were recorded for each infusion: Rate of waterflow into the hole, pressure of water feed, pressure at infusion seal, and the quantity and methane content of air passing established sampling points in the face area. If water remained sealed in the hole after infusion, the sealed-in pressure also was measured.

Infusion of Bleeder Holes in 5 Left Main Entries

When developing the 5 Left Main entries, 12 horizontal bleeder holes were drilled in the coalbed ahead of mining operations to remove the gas. Holes 1, 2, 3, 11, and 12 were water infused to determine if additional quantities of gas could be removed by this method. A summary of the infusion data is given in table 4.

Holes 1, 2, and 3 were infused when the mine was idle because of vacations. The location of these holes relative to the faces is shown in figure 13. At the end of the infusion of hole 1, the water was sealed in the hole and the sealed-in pressure reached 90 psi. This pressure increased to 110 psi in the next 20 hours and decreased to 60 psi at the end of 50 additional hours. Figure 14 is a graph of pressure versus time after infusion of hole 1. Near the end of the infusion period, water seepage was noted across the face and 14 feet outby the face along the left rib of entry 5. Hole 1 was infused again for 29 minutes and sealed for 47 hours, and within this time the sealed-in pressure decreased from 90 to 50 psi.

Following the second infusion, hole 1 was infused simultaneously with hole 2. After these infusions, hole 3 was drilled and infused 10 times. The results of the first infusion are presented separately, while the other nine are presented collectively in table 4. Based on the quantity of methane removed, the increased release of methane per day ranged from 131,000 to 737,300 cubic feet in the immediate returns and from 744,500 to 1,890,700 cubic feet in the main returns.

Because the foregoing infusion tests were performed while the mine was idle, larger amounts of methane were removed than would have been advisable when the mine was operating. During operating shifts water infusion pressures were reduced to lower the amount of methane released by regulating the rate of water injection between 2 to 4 gpm to maintain a water-infusion seal pressure of only 50 psi.

At the close of the first operating shift following vacation, mine officials stated that no gas was detected at the active working faces in the infused area. In their opinion, airborne dust was decreased, visibility was increased, and the coal mined easier. Shuttle-car roadways were damp and compacted.

Hole 3 was infused three times at throttled pressures under operating shifts for 3 successive days. Infusions were stopped when shifts were not operating; water was sealed in the hole, and sealed-in pressures of about 20 psi were recorded. Data collected during these throttled infusions are given in table 4.

TABLE 4. - Pertinent water infusion data collected and methane released during infusions of some holes 1 to 12

Infused hole	Order of water infusion	Flow rate of water infusion, gpm	Infusion time, min	Quantity of water used, gal	Maximum pressure at water-infusion seal, psi	Sealed-in pressure after water infusion of hole, psi	Total methane increase for both immediate returns, cfm	Total methane increase for both main returns, cfm	Remarks
1	1st	14	62	868	160	90	720	1,542	Mine idle.
1	2d	14	29	406	160	90	138	206	Do.
1 2	3d 1st	6-8 6-8	105 105	735 735	160 160	- -	492	985	Mine idle; holes infused simultaneously.
3	1st	14	54	756	160	-	789	1,394	Mine idle.
3	2d to 10th inclusive.	13-15	591	8,300	205	-	91-512	517-1,313	Mine idle; range of increase in methane released for nine infusions.
3	11th to 13th inclusive.	2-4	1,260	3,780	50	20	290	904	Mine operated; three throttled water infusions.
11	1st	12	70	840	140	40	-	267	Mine idle.
12	1st	8-13	590	5,080	175	60	-	267	Mine idle; hole infused consecutively after hole 11.
11	2d	10-11	300	3,020	170	-	-	213	Mine idle.
12	2d	13-15	60	840	100	-	-	213	Mine idle; hole infused consecutively after hole 11.
11	3d	9-15	180	1,700	170	40	-	200	Mine idle.

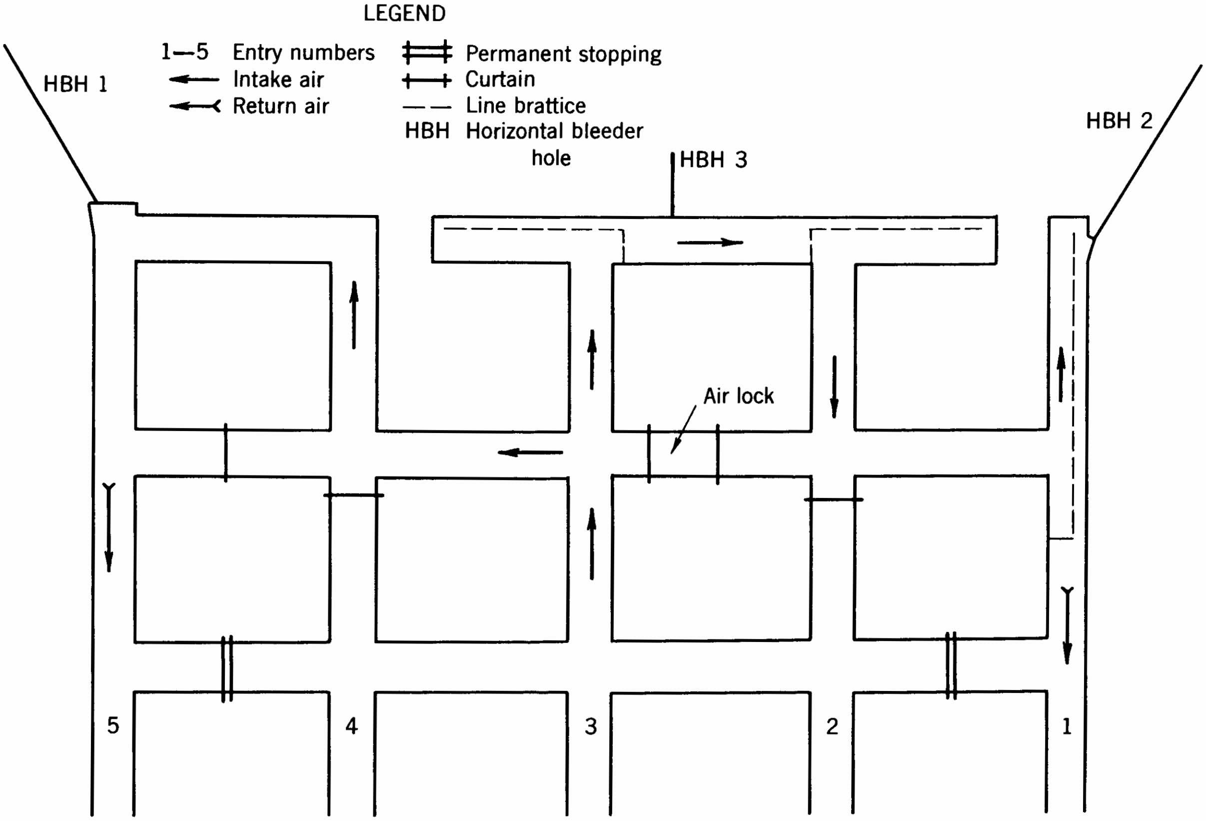


FIGURE 13. - Plan of 5 Left Entry Faces at Time of Drilling Holes 1, 2, and 3.

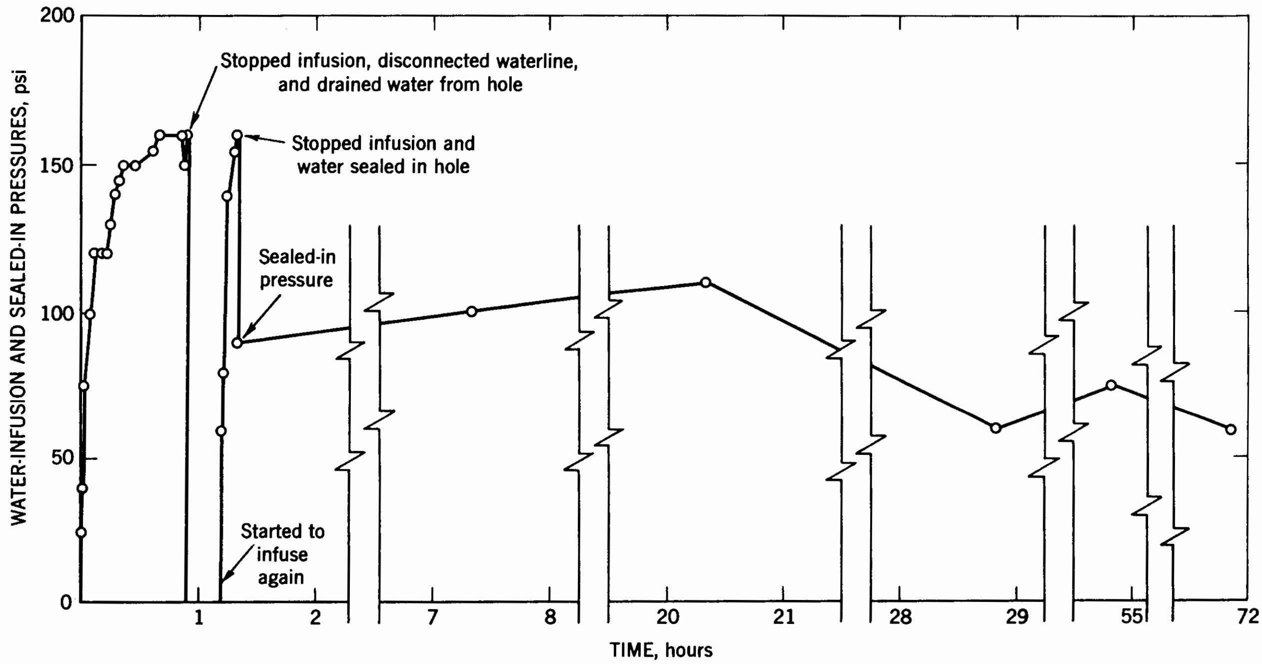


FIGURE 14. - Water-Infusion Seal Pressures During, and Sealed-in Pressures After Infusion at Hole 1 Versus Time.

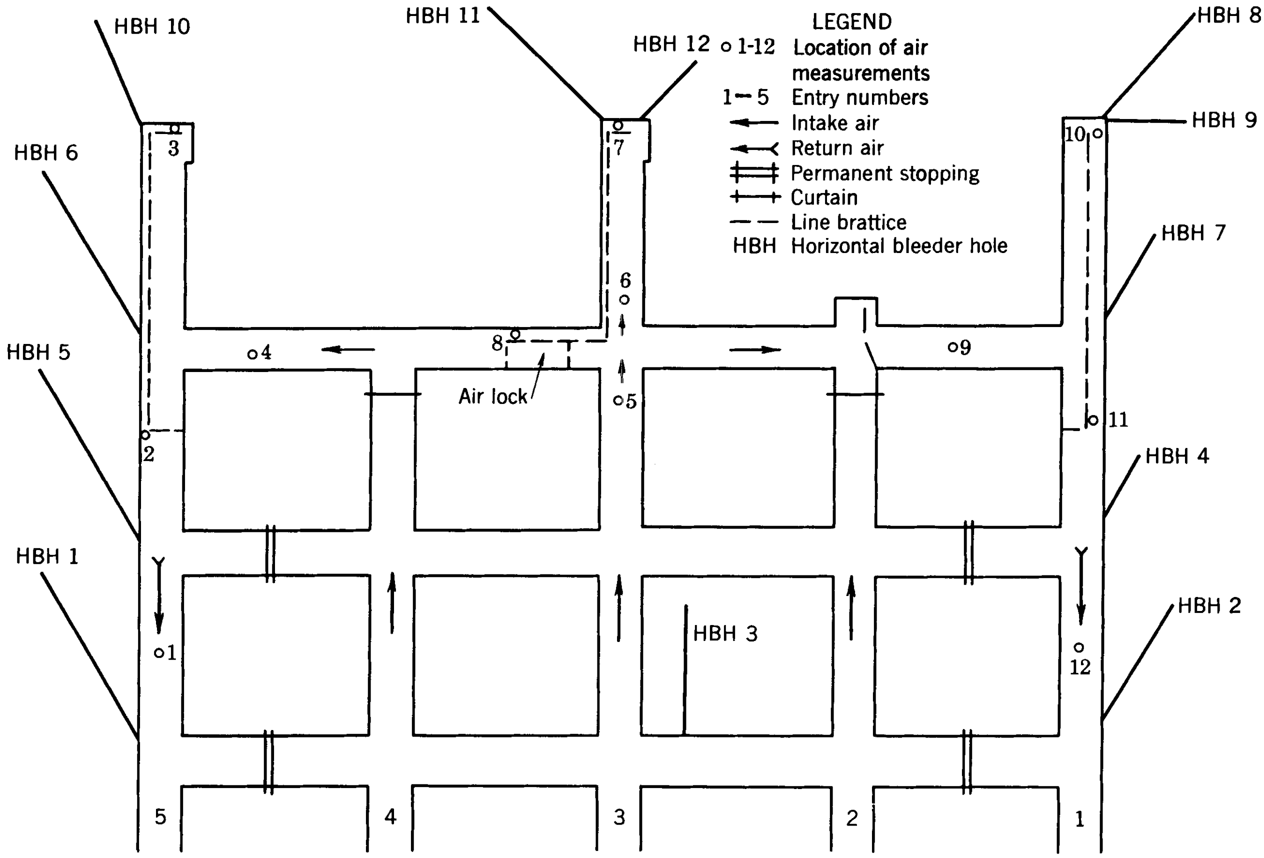


FIGURE 15. - Plan of 5 Left Entries at the Time Holes 11 and 12 Were Drilled. Shows locations of holes 1 through 12, and 12 places air measurements were taken.

As the 5 Left Main entries were advanced, bleeder holes 4 through 12 were drilled at three successive crosscuts inby holes 1, 2, and 3 as shown in figure 15. Holes 11 and 12 were infused alternately during idle shifts until hole 11 had been infused three times, and hole 12 infused twice.

Before hole 11 was infused, 32,500 cfm of air containing 1.8 percent methane (585 cfm) was measured in the left main return and 44,500 cfm containing 0.7 percent methane (312 cfm) was measured in the right main return. Following 30 minutes of infusion at hole 11, the methane content in the left main return increased to 2.35 percent (764 cfm). Also, after 60 minutes of infusion at hole 11, the methane content in the right main return increased to 0.90 percent (400 cfm). Portions of the recording charts showing these methane increases are reproduced in figure 16.

After about 3 hours of infusion of hole 12, the auxiliary exhaust fan at the top of the upcast shaft was stopped for 150 minutes and again for 30 minutes for routine idle-day maintenance when all personnel were out of the mine. Although the volume of methane in the main returns remained the same during fan stoppages, the percentage of methane increased materially because

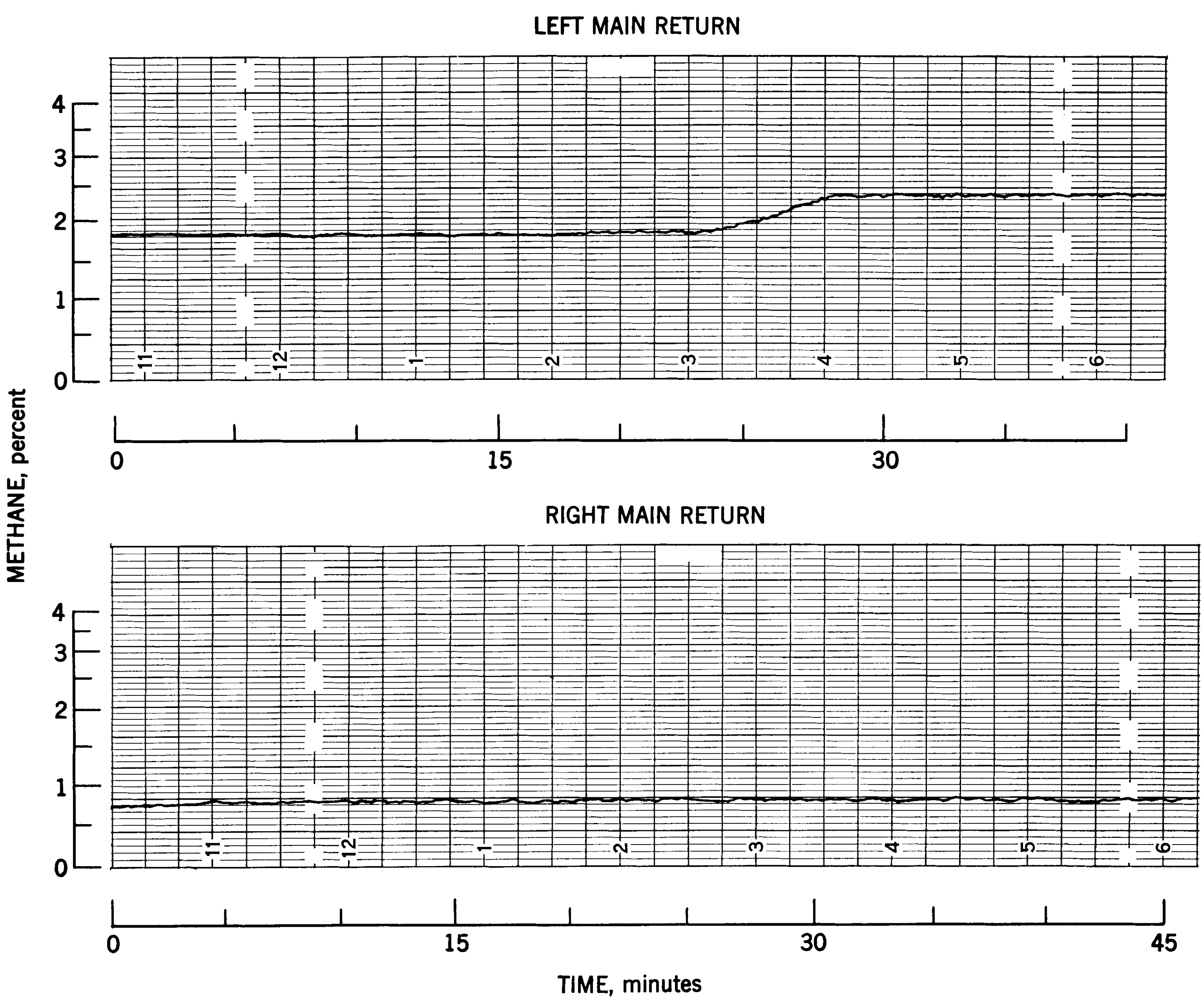


FIGURE 16. - Portions of Recording Charts Showing Percentages of Methane in the Return Air During Infusion.

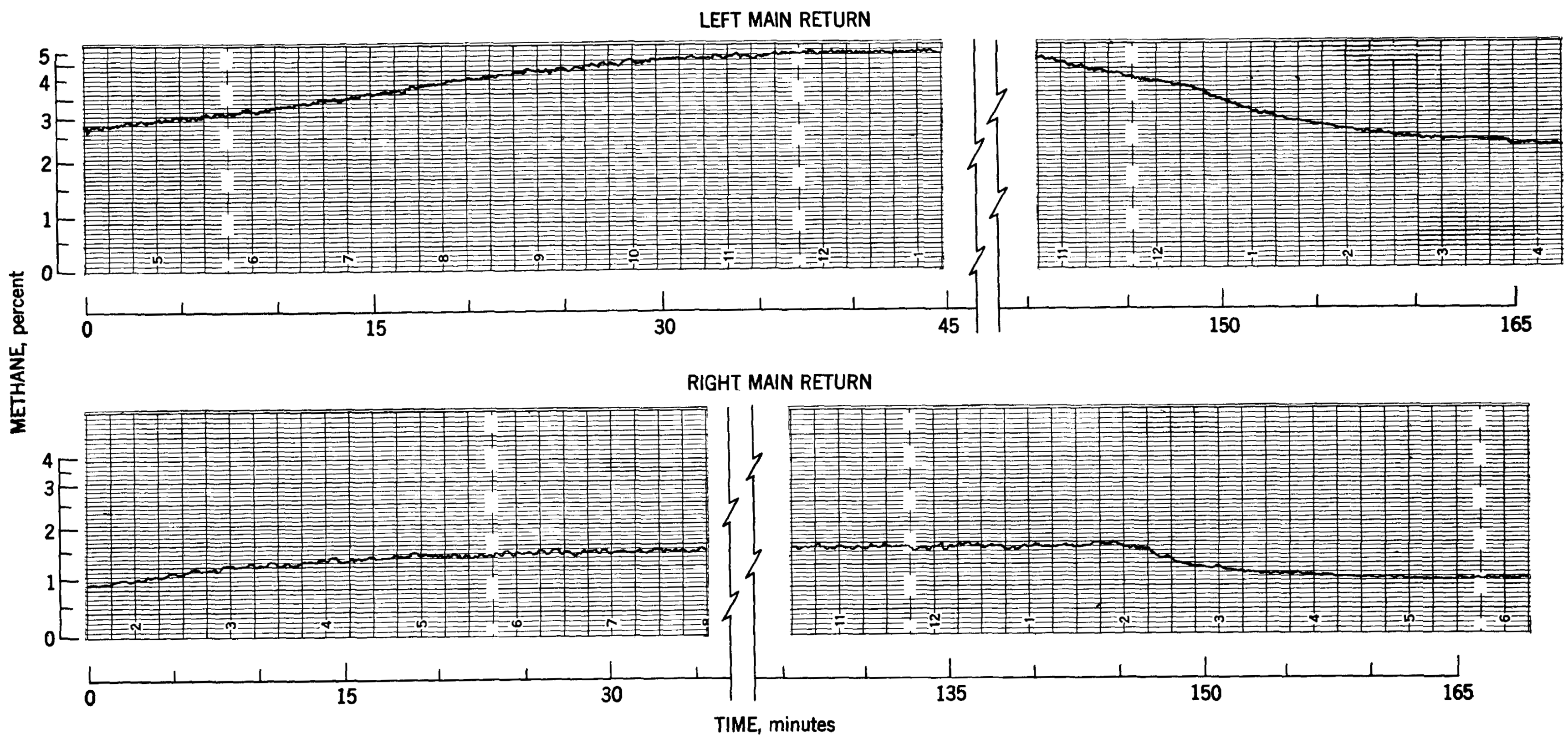


FIGURE 17. - Portions of Recording Charts Showing Percentages of Methane in the Return Air During Infusion When Fan Was Stopped.

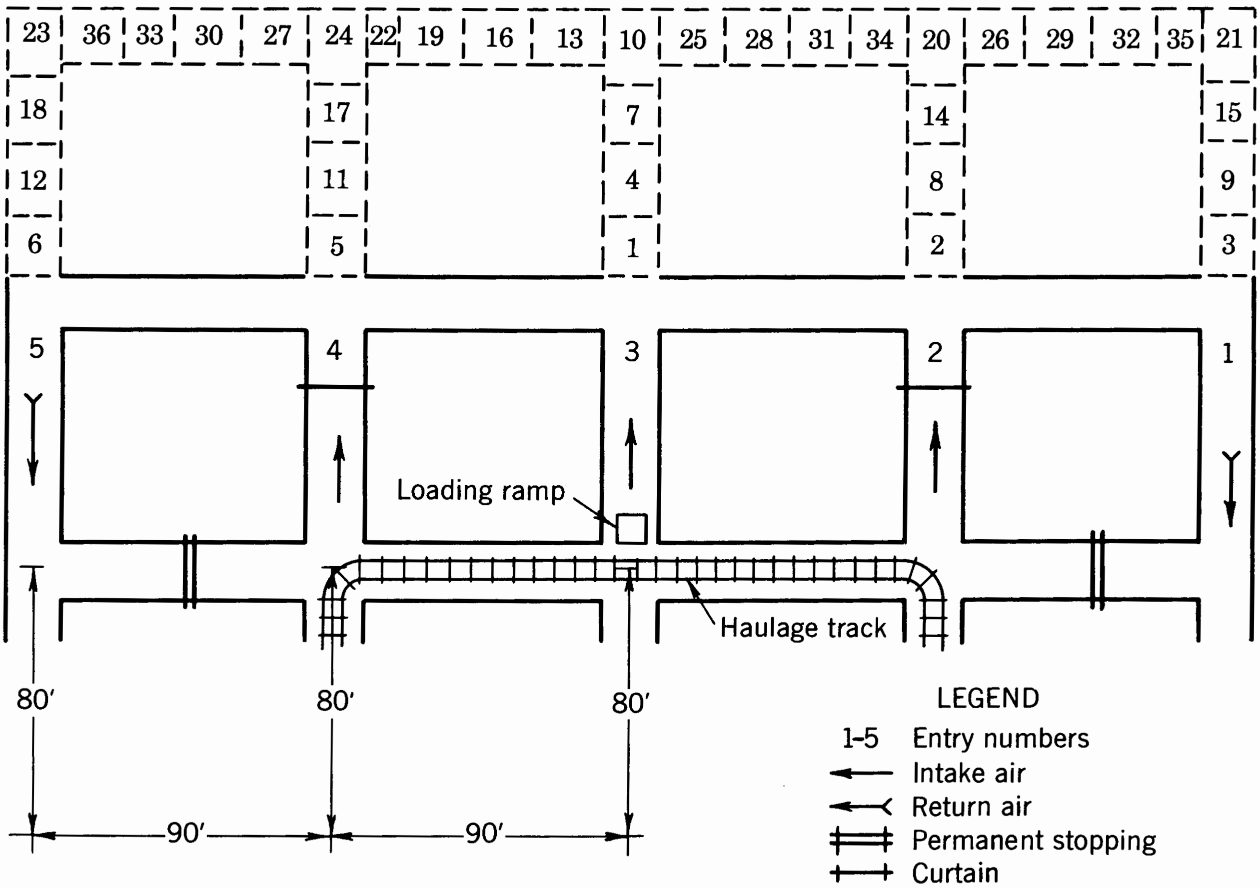


FIGURE 18. - Sequence of Cuts to Advance a Set of Five Entries in Consecutive Order.

approximately 50 percent less air was being circulated. Sections of the recording charts, showing the variations in the percentage of methane in the main return-air currents when the first fan was stopped, are reproduced in figure 17.

Before the second infusion of holes 11 and 12, hole 12 was sealed and a pressure of 0.5 psi was measured. After 150 minutes of infusion, the pressure increased to 7 psi. The seal then was opened and gas containing 70 percent methane issued from the 1/2-inch opening at a velocity of 6,000 fpm. Hole 12 was sealed again and infusion of hole 11 continued for 150 additional minutes, and the sealed-in pressure in hole 12 rose to 9 psi. In the first hour of infusion, water seepage, as noted on the exposed coal surfaces surrounding the collar of hole 11, traveled 21 feet longitudinally and 0.25 feet vertically. Near the end of infusion of hole 11, the outside ribs of entries 1 and 5 were wet 160 feet outby the faces; all the faces on the section were wet, and water was dripping from holes 6 through 10.

Immediately after the second infusion of hole 11 was stopped, hole 12 was infused a second time. Twenty-four hours after this infusion, hole 11 was infused a third time. A summary of the infusion data is given in table 4.

After this test was completed, methane liberations at the face were reduced sufficiently to permit entry development by driving crosscuts from 4 to 5 rather than from 5 to 4, fig. 18. Operating time and production increased because less time was spent in tramming equipment.

Before, during, and after infusion of holes 11 and 12, the volume and methane concentrations of the ventilating air were measured at 12 sampling points to determine the distribution of methane emissions across the section. These sampling points are shown in figure 12 and the data collected is summarized in table 5.

TABLE 5. - Methane released at 12 locations in the working area before, during, and after infusions of holes 11 and 12

Location[1]	Before infusion			During infusion			After infusion		
	Air, cfm	Methane concentration		Air, cfm	Methane concentration		Air, cfm	Methane concentration	
		Percent	Cfm		Percent	Cfm		Percent	Cfm
1........	25,920	0.50	130	28,800	1.30	374	32,160	0.25	80
2........	21,660	0.41	89	22,080	1.08	238	22,800	0.18	41
3........	16,326	0.33	54	16,380	0.80	131	16,650	0.04	7
4........	25,250	0.16	40	26,260	0.45	118	27,775	0.02	6
5........	54,450	0.01	5	54,450	0.01	5	65,300	0.01	7
6........	14,640	0.02	3	15,200	0.10	15	16,400	0.01	2
7........	5,950	0.10	6	6,035	0.50	30	6,200	0.02	1
8........	14,760	0.19	28	15,030	0.53	80	15,300	0.02	3
9........	36,464	0.20	73	37,100	0.40	148	39,750	0.04	16
10........	18,160	0.31	56	18,400	0.60	110	18,560	0.06	11
11........	28,800	0.43	124	29,280	0.75	220	29,520	0.11	32
12........	38,130	0.50	191	39,525	1.00	395	41,850	0.16	67

[1] See figure 15.

Gas emissions from holes 1 through 10, except hole 3 which entry development had rendered useless as a bleeder hole, were measured before, during, and after infusion of holes 11 and 12 to determine the effects of infusion on the quantity and quality of the methane removed. This data is given in table 6. Before infusion, 100 percent methane was issuing from all holes except 9 and 11, but during infusion, the percentage of methane decreased to as low as 50 percent at one hole. The amount of methane emitted from the holes during infusion, however, was considerably greater than the amount emitted before infusion.

Discussion of Infusion Results, 5 Left Main Entries

After infusion tests at holes 1, 2, 3, 11, and 12 were completed, the methane liberations in the 5 Left section were the lowest ever measured in the section. Methane content of the ventilating air in the immediate and main returns was 120 cfm and 500 cfm, respectively, which represented a decrease of 680 cfm and 1,500 cfm, respectively, from amounts recorded before the degasification study was begun; this was equal to a methane reduction of 85 percent in the immediate return-air courses and 75 percent in the main return-air courses.

TABLE 6. - Methane released from holes 1 through 10 before, during, and after infusions of holes 11 and 12

Hole	Before infusion			During infusion			After infusion		
	Velocity of gas, fpm	Methane concentration		Velocity of gas, fpm	Methane concentration		Velocity of gas, fpm	Methane concentration	
		Percent	Cfm		Percent	Cfm		Percent	Cfm
1	500	100	11	1,000	90	20	300	100	7.0
2	1,400	100	31	2,000	97	43	800	99	17.0
4	1,000	100	22	1,600	75	26	500	95	10.0
5	800	100	17	2,300	60	30	500	100	11.0
6	500	100	11	3,000	50	33	355	98	8.0
7	1,500	100	33	2,500	80	44	800	95	17.0
8	300	100	7	1,200	70	18	250	97	5.0
9	0	0	0	1,000	70	15	22	98	0.5
10	700	100	15	1,500	65	21	500	97	11.0
[1]11	1,000	70	15	-	-	-	-	-	-
[1]12	400	100	9	-	-	-	-	-	-

[1]Hole infused.

After consultation with mine officials, it was decided to defer further degasification experiments until increased gas liberations occurred or were anticipated. Additional bleeder holes, therefore, were not drilled for the next 30 days, but methane readings were taken to determine if any changes occurred in the percentage of gas in the air currents.

Data collected 30 days after completion of tests showed that the amount of methane in the ventilating air was 80 cfm in the immediate returns and 470 cfm in the main returns. These amounts were less than those recorded at the completion of the tests in the 5 Left entries, and drilling of additional bleeder holes was again deferred.

Continuous methane readings were not taken in the next 40 days because the continuous methane recording units were removed from the mine for adjustment and recalibration, but analyses of the mine air for methane were made regularly by mine officials.

In this 40-day period, the 5 Left entries and the Right Side entries off 5 Left had been developed to the want area and mining in these areas was completed (fig. 3). Methane liberations sufficient to require operational downtime were not encountered during this period.

Infusions of Bleeder Holes in Left Side Entries

Methane monitoring units were again placed in the 5 Left Main returns when the Left Side entries were begun. The methane volume in the left and right main returns was 256 cfm and 123 cfm, respectively, and these amounts were used as a comparative basis for assessing the degasification results. As the Left Side entries approached the want area, methane volumes in the left and right main returns increased to 326 cfm and 167 cfm, respectively. In view of these increased emissions, further degasification in this area was deemed advisable.

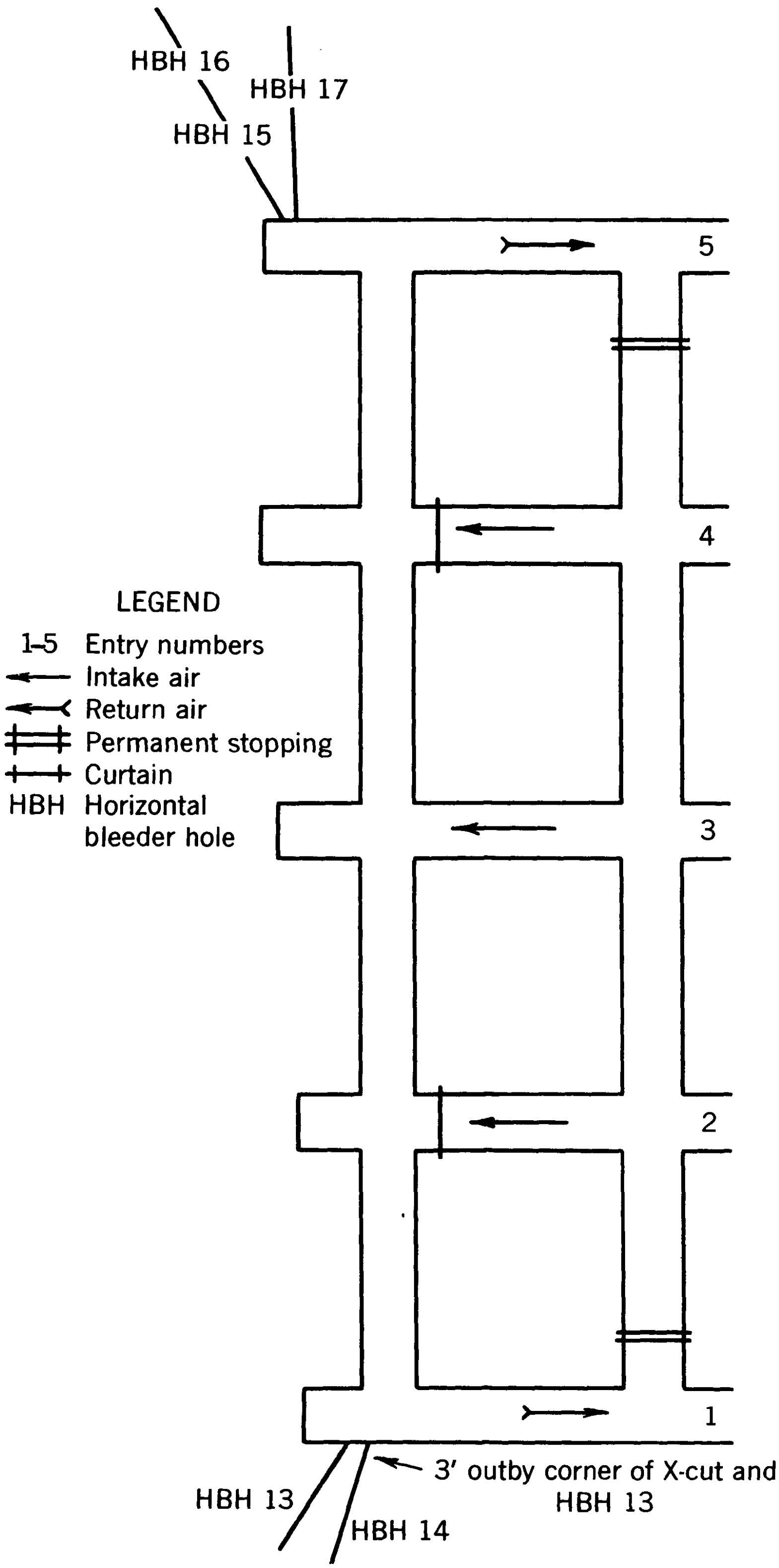


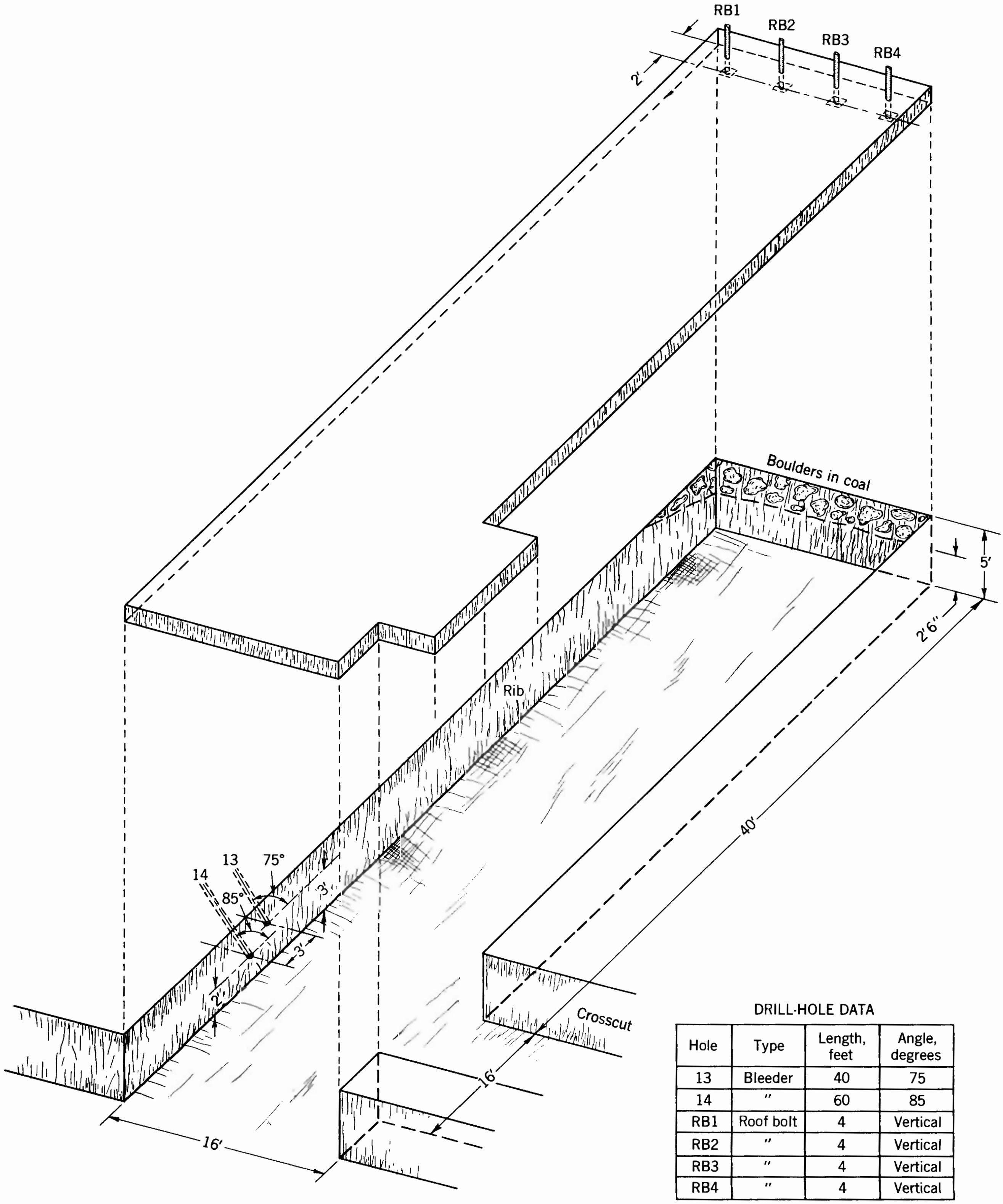
FIGURE 19. - Plan of Left Side Entry Faces at the Time Holes 13 Through 17 Were Drilled.

Five bleeder holes, 13 through 17, were drilled in the outside entries as shown in figure 19. Holes 13 and 14 were drilled in entry 1 and the other three were drilled in entry 5. Hole 16 was above and parallel with hole 15.

When infusion in the Left Side entries began, entry 1 had been driven about 40 feet inby hole 14, and entry 5 had been driven about 80 feet inby holes 15 and 16. The roof of entry 1 was irregular and contained coal streaks; the upper part of the coalbed contained sandstone boulders, fig. 20.

Hole 15 was infused using the high-pressure pump and normal waterline pressures a week after drilling. Pertinent data collected is summarized in table 7. Water, sealed in the hole in the 14 minutes between the first and second infusion reached a pressure of 100 psi.

Holes 16 and 17 were not sealed before commencing infusion of hole 15, and water flowed from hole 16 after 11 minutes and from hole 17 after 26 minutes. Both holes then were sealed and the pressure, 5 psi recorded at hole 17 after 161 minutes of infusion, increased to 10 psi while the pressure at hole 16 reached 5 psi near the end of the infusion period. Although no pressures were indicated at holes 13 and 14, gas containing 90 percent methane issued from hole 14 at a velocity of 125 fpm.



DRILL-HOLE DATA

Hole	Type	Length, feet	Angle, degrees
13	Bleeder	40	75
14	"	60	85
RB1	Roof bolt	4	Vertical
RB2	"	4	Vertical
RB3	"	4	Vertical
RB4	"	4	Vertical

FIGURE 20. - Face Area, Entry 1, Left Side, at the Time of Initial Infusion.

TABLE 7. - Pertinent infusion data collected and methane released during infusions of some holes 15 to 19

Infused hole	Order of water infusion	Flow rate of water infusion, gpm	Infusion time, min	Quantity of solution used, gal	Maximum pressure at water infusion seal, psi	Infusion pressure: Normal water line pressures, psi	Infusion pressure: Pump pressure, psi	Sealed-in pressure after infusion of hole, psi	Total methane increase in both immediate returns, cfm	Total methane increase in both main returns, cfm	Remarks
[1] 15	1st	8	3	24	140	200-220	-	100	138	389	Mine idle.
[1] 15	2d	8-12	338	3,280	140	-	400	-			Do.
[1] 15	3d	10	18	180	122	200-220	-	-	510	645	Do.
[1] 15	4th	10-12.5	307	3,360	160	-	400	-	-		Do.
[2] 18	1st	9-10	55	520	188	200-220	-	40	-	62	Do.
[2] 18	2d	20-28	120	2,720	250	-	650	-	-	282	Do.
[2] 18	3d to 12th inclusive.	9-10	1,330	12,000	195	200-220	-	-	-	18-70	Mine operated.
[2] 18	3d to 12th inclusive.	20-28	530	12,760	370	-	650	-	-	18-70	Do.
[2] 19	1st	8	160	1,280	180	200-220	-	-	-	44	Do.
[2] 19	2d	3-4	240	750	200	200-220	-	-	-	16	Do.

[1] Infused with water.
[2] Infused with water mixed with a wetting agent and fluorescein dye.

While hole 15 was being infused, water exuded freely from the coal surfaces surrounding the hole. This seepage extended 27 feet to the left after 1 hour and was noted at a maximum distance of 64 feet from the hole before the close of the infusion period.

Gas emissions accompanying the water seepage formed gas-water bubbles on the right rib of entry 5 on both sides of the hole being infused, fig. 21. After 2 hours, the velocity of the gas being emitted from the roof-bolt holes RB1 and RB2 at the face of entry 1 (fig. 20) increased from near zero before infusion to 3,000 fpm. Emissions were also detected from the floor and left rib of this entry. Minor emissions were noted from other roof-bolt holes near or in the want area, but not in other areas. Two hours after stopping the second infusion, the methane volume in the immediate and main returns was 300 cfm and 857 cfm, respectively.

About 12 hours after the completion of the second infusion, hole 15 was infused with water at normal waterline pressures for 18 minutes. Forty-five minutes after completion, a fourth infusion was made for 307 minutes using the high-pressure pump. (See table 7.) Two hours after completion of infusions of this hole, the quantities of methane detected in the immediate and main returns were 56 cfm and 118 cfm respectively, below the quantities detected before infusions.

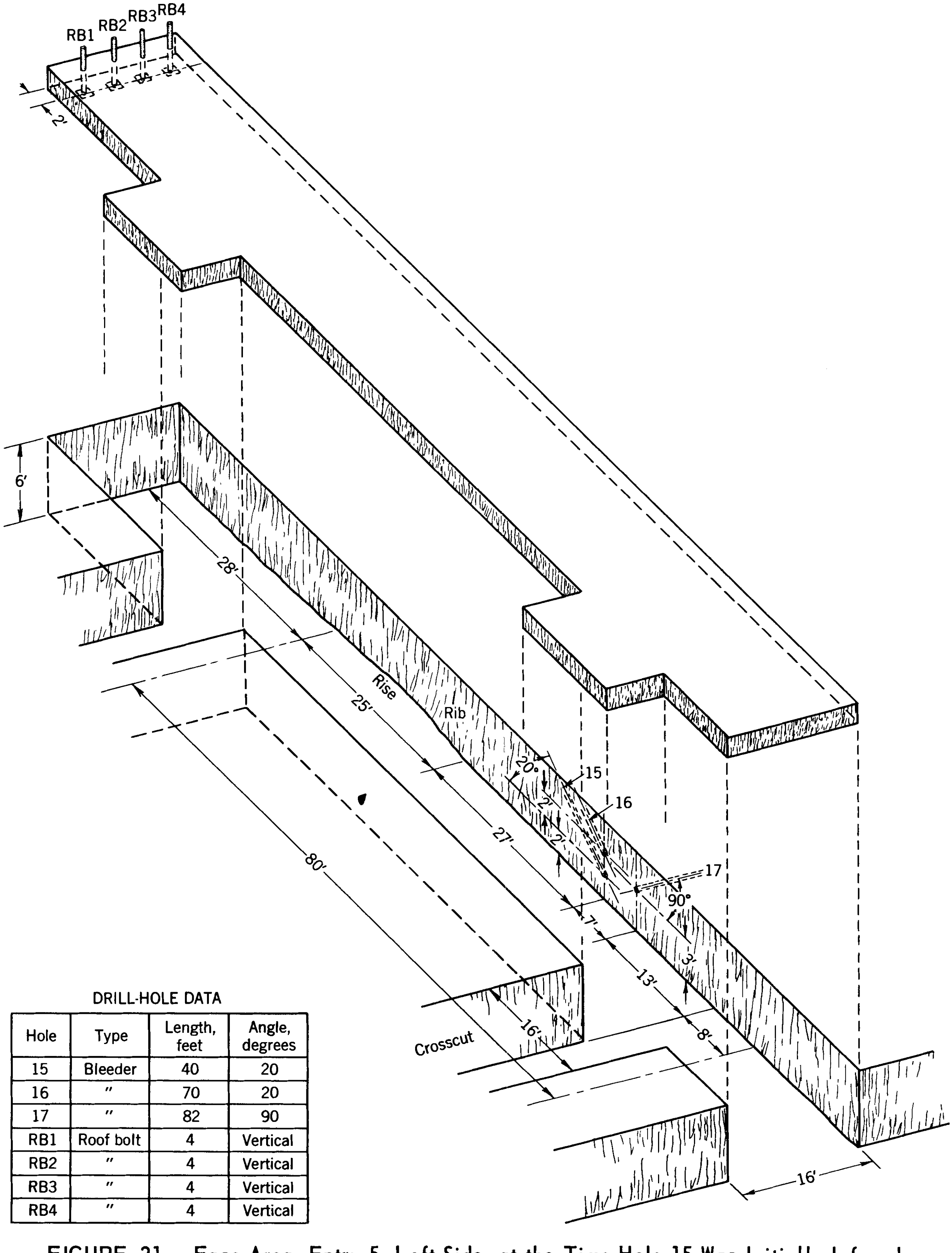
Discussion of Infusion Results, Left Side Entries

After infusion of the drill holes in the Left Side entries from 5 Left, the amount of methane in the main returns was reduced from 606 to 370 cfm. There was no indication of damage to the floor or roof as a result of water infusions.

Infusions of Bleeder Hole 18

Bleeder hole 18 was drilled in the left rib of Left Side entry 1 between projected rooms 2 and 3, fig. 22. A total of 12 infusions of hole 18 were made using normal waterline pressures and the high-pressure pump alternately. To aid in tracing the paths of water migration, fluorescein dye and a liquid wetting agent was added to the water used for holes 18 and 19. A summary of infusion data is given in table 7.

After 30 minutes of infusion at normal waterline pressures, the solution was noted to have traveled 22-1/2 feet longitudinally and 3 inches vertically along the surface of the left rib. After 10 minutes of infusion using the high-pressure pump, the solution was noted on the coal surface and had traveled 10 feet longitudinally and 1-1/2 inches vertically. After the second infusion, the solution was traced about 400 feet from the infused hole along the rib of entry 5 of 5 Left Main entries. When rooms were developed off Left Side entry 1 the solution had penetrated the solid coal the full 160-foot depth of these workings.



DRILL-HOLE DATA

Hole	Type	Length, feet	Angle, degrees
15	Bleeder	40	20
16	"	70	20
17	"	82	90
RB1	Roof bolt	4	Vertical
RB2	"	4	Vertical
RB3	"	4	Vertical
RB4	"	4	Vertical

FIGURE 21. - Face Area, Entry 5, Left Side, at the Time Hole 15 Was Initially Infused.

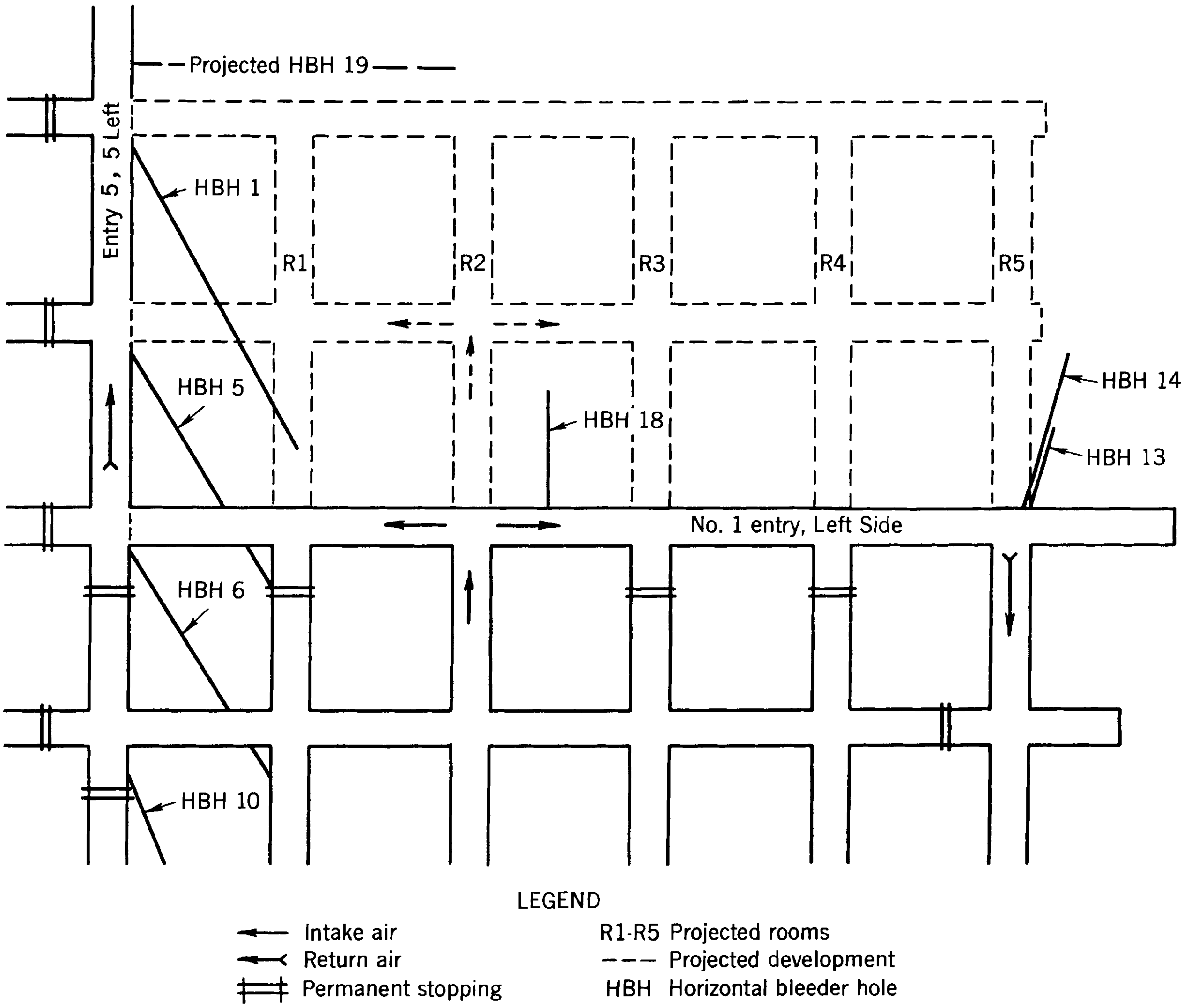


FIGURE 22. - Projected Plan of Development of Five Rooms off Entry 1, Left Side, at the Time Hole 18 Was Drilled. Also shows location of projected hole 19.

Discussion of Infusion Results, Hole 18

Although previous infusions had affected this area, the methane content of the ventilating air decreased from 378 cfm before, to 306 cfm after infusions of hole 18. The amounts of methane released varied, but were greater when using the high-pressure pump.

Water infusions appeared to be effective in reducing airborne dust in the working sections. From visual observations by Bureau personnel, infusions made with water containing wetting agents at pump pressures were more effective in allaying dust than those with water alone at normal waterline pressures. These observations were substantiated by a study made by management before, during, and after infusions of hole 18. A dust counter was used in this study to sample the airborne dust at the working faces and other strategic locations in the working section.

Infusions of Bleeder Hole 19

After rooms 1 through 5 were advanced to the last crosscut, hole 19 was drilled into the solid coal off Main entry 5, 5 Left, about 30 feet outby and paralled with the last projected crosscut connecting room 1 and Main entry 5. The projected location of hole 19 is shown in figure 22. Two infusions of hole 19 were made while developing the last crosscut connecting the rooms, and water with a wetting agent and fluorescein dye added was injected into the hole. A summary of infusion data is given in table 7.

Discussion of Infusion Results, Hole 19

At the end of the infusions in hole 19, the methane concentration of the ventilating air in the left and right immediate returns was 8 cfm and 39 cfm, respectively, and in the left and right main returns was 120 and 153 cfm, respectively.

At the time of the last infusion test at hole 19, extraction of pillars was begun, and the degasification study in virgin coal in this section was concluded.

MIGRATION TESTS--ANHYDROUS AMMONIA

Tests to determine whether gas migration is preferentially along or across bedding planes were conducted using anhydrous ammonia as a tracer. Five holes were drilled in a freshly exposed working face as shown in figure 23.

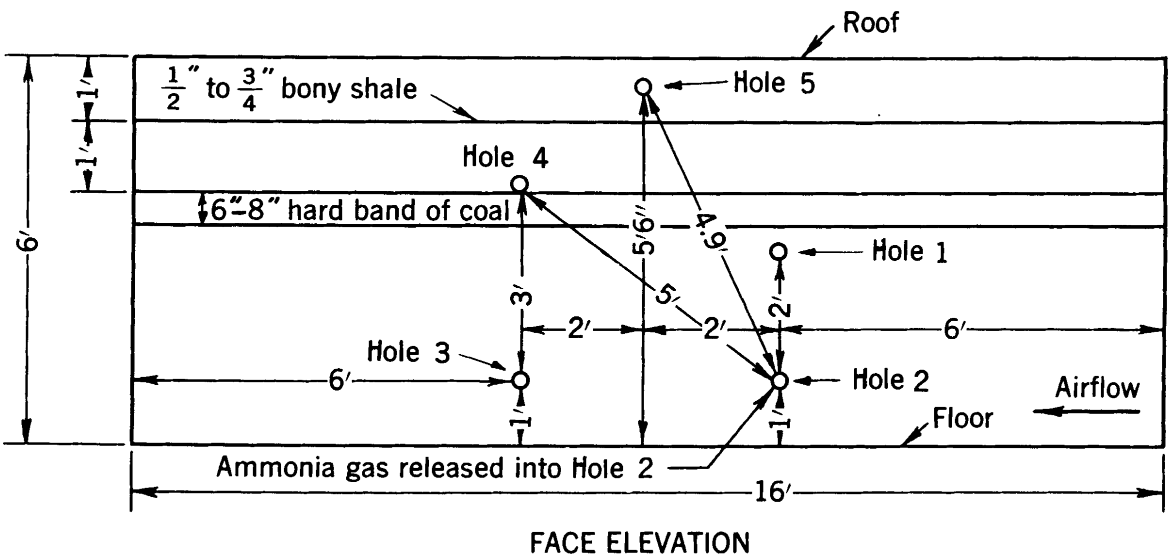


FIGURE 23. - Spacing of Holes in Coal Face for Migration Tests.

A 4-foot length of 1/8-inch-diameter rubber tubing was inserted 6 inches and sealed in each hole except hole 2.

A piece of red litmus paper and a piece of cloth saturated with a weak solution of hydrochloric acid were attached to the free end of the tubes. These tubes were held by an observer. While an observer sniffed the hand-held tubes, the ammonia was released under pressure of 80 psi from a cylinder through an infusion seal placed in hole 2. As the gaseous ammonia passed through the coalbed to the open tubes sealed in the coal, it was detected by its pungent odor, fuming of the acid-soaked cloth, and the change in color of the litmus paper.

It required 20 seconds for ammonia gas to pass from hole 2 to hole 1; 25 seconds from hole 2 to hole 3; 40 seconds from hole 2 to hole 4; and 55 seconds from hole 2 to hole 5, showing that the gas traveled faster along the bedding planes than across the planes.

CONCLUSIONS

From observations made and data collected, these methane drainage experiments proved that methane can be removed from the Pocahontas No. 4 coalbed before and during mining by drilling along horizontal bleeder holes in the coalbed, and infusing them with water.

During the degasification study, the amount of methane in the main return-air currents was reduced from 2,000 cfm to 270 cfm, a reduction of more than 86 percent.

Gas liberations by free flow from horizontal bleeder holes drilled 30 to 230 feet deep in the coalbed in advance of mining reached a maximum of 131 cfm immediately after drilling. The flow from the holes decreased steadily over periods ranging from several hours to several days, although some of the holes continued to emit gas at decreasing rates for several months. Shut-in pressures of the horizontal bleeder holes were as high as 63 psi.

Infusing some of the bleeder holes with water, or a solution of water mixed with a wetting agent, increased gas emissions from virgin coal areas. During one infusion when the mine was idle, gas liberations from the coalbed were increased 1,540 cfm.

Infusions of bleeder holes proved effective in releasing large amounts of gas from the coalbed before mining, and as a result, gas emissions while mining, were reduced materially. Operational downtime to dilute gas liberations to safe concentrations in the ventilating air that occurred frequently before degasification was eliminated.

Gas releases that were excessive while infusing at normal waterline pressures were reduced by decreasing the water-infusion seal pressures. This was done by regulating the rate of water injection.

In confirmation of results obtained in previous tests, maintaining a sealed-in pressure in infusion holes resulted in greater methane emissions than would have occurred if the hole had been drained immediately after infusion. Usually, the amount of gas released varied with the sealed-in pressures. Sealing water in the holes after infusion retards the escape of water from crevices in the coalbed and presumably prevents return of the gas into the infused area.

Because gas is emitted from long horizontal bleeder holes for relatively long periods of time and the release of gas from the coalbed is increased during infusion of holes, investigators believe that most of the gas was held in the coalbed rather than in the adjacent strata.

Infusion water did not penetrate the sandstone roof, nor the shale floor, and no damage from water was observed on either the roof or floor of the mine.

Observation and results of tests indicated that water infusions at normal waterline pressures reduced the amount of airborne dust at the face during mining, but were more effective in allaying dust when a wetting agent was added to the solution and infusions were made at pump pressures.

Experiments with tracers introduced into the infusion water and released under pressure into a horizontal bleeder hole proved that water and anhydrous ammonia travel through the coalbed. Infusion water mixed with a wetting agent and fluorescein dye traveled 400 feet through coal.

Various rates of travel of water or solutions used in infusions as detected on exposed coal surfaces were as follows:

1. Water at normal waterline pressure traveled 21 feet laterally and 3 inches vertically per hour.

2. Water and wetting agent with fluorescein dye at normal waterline pressure traveling 45 feet laterally and 6 inches vertically per hour.

3. Water and wetting agent with dye at pump pressure of 400 psi traveled 60 feet laterally and 9 inches vertically per hour.

Anhydrous ammonia under pressure released into a horizontal borehole traveled 12 fpm horizontally, 5 fpm vertically, and 6 fpm diagonally, through the coalbed.