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Methane Migration Characteristics of the Pocahontas No. 3 Coalbed

UNITED STATES DEPARTMENT OF THE INTERIOR

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Methane Migration Characteristics of the Pocahontas No. 3 Coalbed

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ILLUSTRATIONS

METHANE MIGRATION CHARACTERISTICS OF THE POCAHONTAS NO. 3 COALBED

by

Fred N. Kissell[1]

ABSTRACT

Methane-flow and pressure data taken from a mine in the Pocahontas No. 3 coalbed are compared with flow rates from lump coal obtained in laboratory experiments. From this, it is concluded that the main source of gas is the intact coalbed rather than a "crushed zone" near the working face. The permeability and sorption capacity of the intact coalbed are calculated and gas emission rates are theoretically accounted for.

INTRODUCTION

The Bureau of Mines has been concerned with the necessity of pinpointing the source of methane in coal mines, if preventive techniques are to be used to control methane emission. Various workers (10, 13),[2] particularly in Europe, have hypothesized that much of the methane emitted at the longwall working face originates in a high-permeability relaxed or fracture zone, which extends into the coal being worked and also into adjacent strata. Fracturing in the coal being worked takes place because increased stress levels resulting from the removal of adjacent coal have exceeded the compressive strength of coal in the zone, partially crushing it and increasing its permeability. Fracturing in the adjacent strata takes place because of caving. Although a large quantity of methane may be stored in coal, frequently under high pressure, the permeability of an undisturbed coalbed may be quite low and the gas does not flow freely to the working areas. However, crushing the coal greatly increases its permeability so the gas in the crushed zone flows easily into the mine.

On the other hand, Cervik (3) implies that a source of methane in U.S. mines is in the intact coal beyond the crushed zone. This seems to be a likely hypothesis because most U.S. coals are mined at shallower depths than European coals. Because U.S. mines are shallower, the ground pressure is less and the crushed zone may be less extensive. The permeability of the undisturbed coalbed may be considerably higher, given the enormous changes in

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[1] Physical research scientist.

[2] Underlined numbers in parentheses refer to items in the list of references preceding the appendixes.

permeability shown in laboratory permeability-stress experiments with coal. Also, the roof is not immediately caved.

That some basic difference in the source of methane may exist while mining, U.S. coalbeds and those in Europe have never been subjected to experimental test. It is the intent of this report to ascertain if a coalbed crushed zone really exists while mining by the advancing room-and-pillar method in the Pocahontas No. 3 coalbed, and whether the main source of methane is in this zone or in the unmined and intact coal adjacent to it. At the same time, an attempt will be made to explain theoretically the gas flow rates observed in a working mine in terms of what is known about the desorption and flow characteristics of Pocahontas No. 3 coal.

ACKNOWLEDGMENTS

Acknowledgment is due the Bureau of Mines people who have contributed data: Jack Perkins for the lump desorption results and John D. Kalasky and Joseph D. Hadden for the information from the Beatrice mine.

EVIDENCE FOR THE CRUSHED ZONE

The Bureau of Mines has been engaged in a program of drilling deep horizontal holes into coalbeds, packing them with inflatable packers, and measuring gas pressures. In contrast to results obtained by Gunther (4), the packers generally have succeeded in sealing the hole effectively; also, the seam permeability always has been high enough to insure that the pressure measured was a true equilibrium pressure--that is, it gives a true indication of the actual gas pressure in the fractures.

A curve of gas pressure versus depth in a horizontal hole in the Pocahontas No. 3 coalbed is shown in figure 1, along with the curve that might be expected if the coalbed had a constant permeability. It is clear from inspection of these curves that a zone of sharply increased permeability is in approximately the first 10 ft. This is the crushed zone.

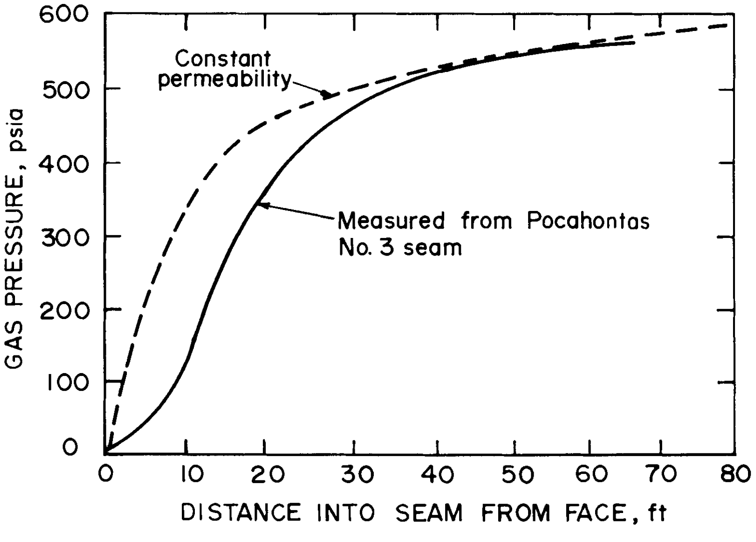

FIGURE 1. - A Pressure Curve.

THE SOURCE OF METHANE DURING IDLE PERIODS

The role of the crushed zone in contributing to gas flow is most easily assessed by comparing laboratory flow rates from lump coal with those measured at a working face in a mine when the mine is idle. It is assumed that the coal in the crushed zone simply consists of a large quantity of desorbing lumps and, because of the high permeability of the zone, gas flow to the face area is unimpeded. If the flow rate per ton of coal observed in the laboratory is equal to or exceeds that from a working face during an idle period when operation of the mining machine does not contribute to the gas flow, then this would be strong evidence that the crushed zone is in fact the main source of gas. On the other hand, if the laboratory flow rate falls short of the gas flow from the working face, then an additional source of gas in the mine is indicated.

Results From the Laboratory-Desorption Rates From Lumps

Recent work on the desorption of methane from lump coal in the laboratory has been done by Airey (1). He found that the room temperature desorption of methane from lump coal follows the empirical formula

$$V(t) = A \left[1-e^{-\left(\frac{t}{t_o}\right)^{1/3}} \right],$$

where V(t) is the amount of methane desorbed in time t, and A and t_o are constants dependent on initial pressure and lump size, respectively. A theoretical estimation of flow rate based on unsteady-state Darcy flow from a thin slab was not a good fit to this equation, and a more accurate fit could be obtained by assuming that a distribution of slab thicknesses existed within a given slab. Airey also found that the parameter t_o varied directly with lump size for smaller lumps, but that for larger sizes the variation of t_o with size is considerably less. This was to be expected because it is known that coal has a basic structure of fine cracks and that larger lumps always contain numerous fractures. Because large lumps have smaller variation of t_o with size, the desorption curves of large lumps are much less affected by size. Figure 2 shows a desorption curve by Airey for ¼- to ½-in size range coal and also a desorption curve for 6- to 12-in size range coal based on an extrapolation of his parameter t_o. It may be seen that for the 75- to 325-hour period, the difference in the amount desorbed is about 25 ft^3 per ton; however, the rates of emission in cubic feet per ton-hour for the two size ranges are very similar.

The room temperature desorption rate of ¼- to ½-in size Pocahontas No. 3 coal has been measured by the Bureau of Mines (fig. 2). Despite the fact that the total amount of gas desorbed is greater than that for the coal used by Airey,[3] the emission rate, 0.3 ft^3 per ton-hr, between 75 and 325 hours is similar. Preliminary results on larger Pocahontas coal indicate a similar

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[3] To some extent the amount desorbed is greater because Perkins used dry coal, which is known to have greater capacity for absorbing methane than moist coal. The effect of moisture on the lump emission rate is not known, but more than likely it would be reduced.

emission rate. Because of this, it can be assumed that the lump emission rate may be compared directly with flow rates from a hypothetical crushed zone where the coal falls within a size range of ¼ to 12 in, provided the time period, initial pressures, and coal are all similar.

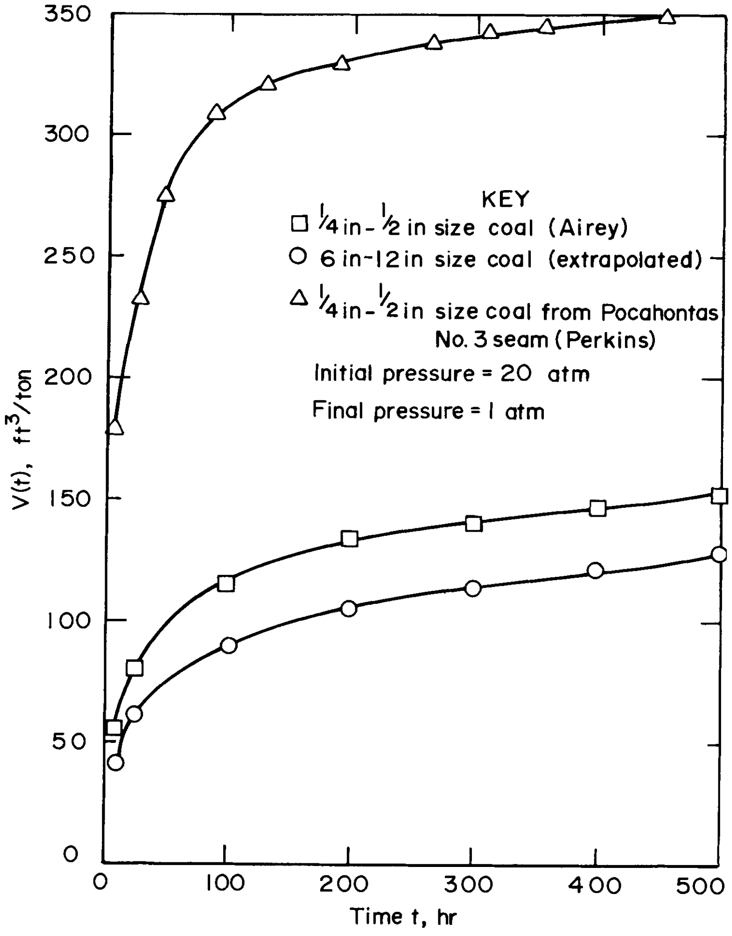


FIGURE 2. - Desorption Curves for Coal.

Results From the Mine

Some methane flow data were obtained recently in the Beatrice mine. This mine, operated in the Pocahontas No. 3 coalbed, is located near Keen Mountain, Va. Unit No. 7, section 3d North, where flow measurements were taken, is being worked by the advancing room-and-pillar method. The overburden here is approximately 2,000 ft thick. Anemometers and recording methanometers set up in the intake and return airways were used to obtain the methane flow from the face area.

Mining stopped on June 26, 1970, for 2 weeks. Figure 3 shows the flow rate during this idle period as well as the flow before and after idleness when mining was in progress. The 75- to 325-hour period selected above corresponds to the interval from June 29 to July 10. During this time the methane flow from the face area averaged about 74 cfm and the total variation was about ±13 percent.

The face area in this section was 2,400 ft^2, and a crushed zone 10 ft wide would contain 970 tons of coal. If the laboratory flow rate obtained by Perkins between 75 and 325 hr is applied to this 970 tons of coal, the theoretical flow rate is 5 cfm. This is much less than the 74 cfm observed in the mine during the corresponding June 29 to July 10 period. A wider crushed zone would produce more than 5 cfm, but this is difficult to visualize. Calculations given in appendix C yield a crushed zone thickness of 8.6 ft. A zone 20 ft thick would still only account for 10 cfm. Thicker zones would not permit the steep gradients shown in figures 1 and 4 to exist. A source of gas other than the crushed zone is indicated.

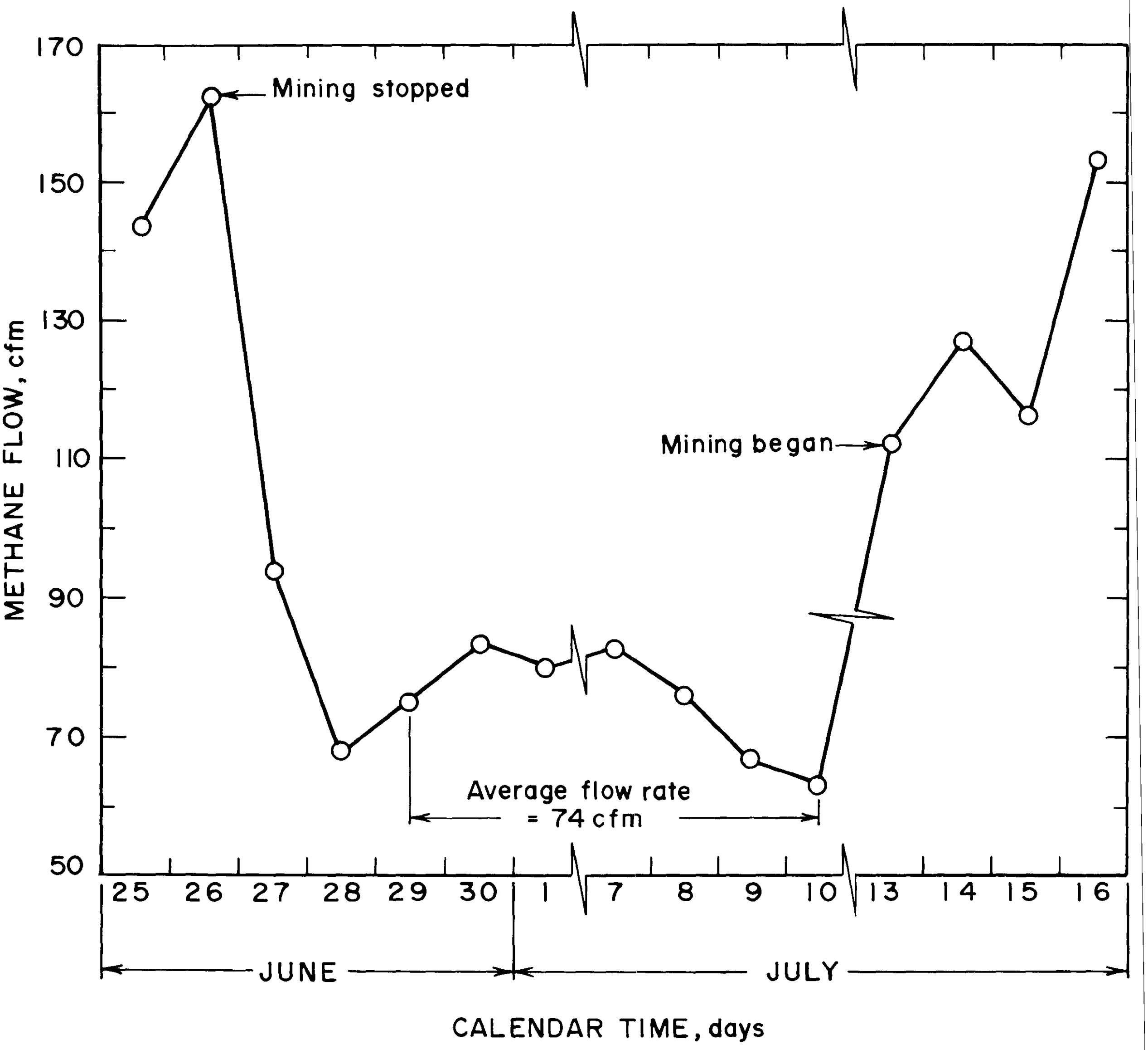


FIGURE 3. - Methane Flow From Face Area.

During mining, the roof and floor in the working face area were both intact; very little gas was issuing from adjacent strata. It follows that the primary source of gas during the idle period must be behind the crushed zone in the undisturbed coalbed.

ESTIMATION OF COALBED PERMEABILITY AND SORPTION CAPACITY

Assuming the primary source of methane in this idle mine is in the coalbed behind the crushed zone, the 74-cfm flow rate from figure 3 can be used to obtain the coalbed permeability and sorption capacity if a pressure curve such as figure 1 is known. In fact, the Bureau of Mines has obtained pressure curves from measurements in the same section. One taken 360 hr

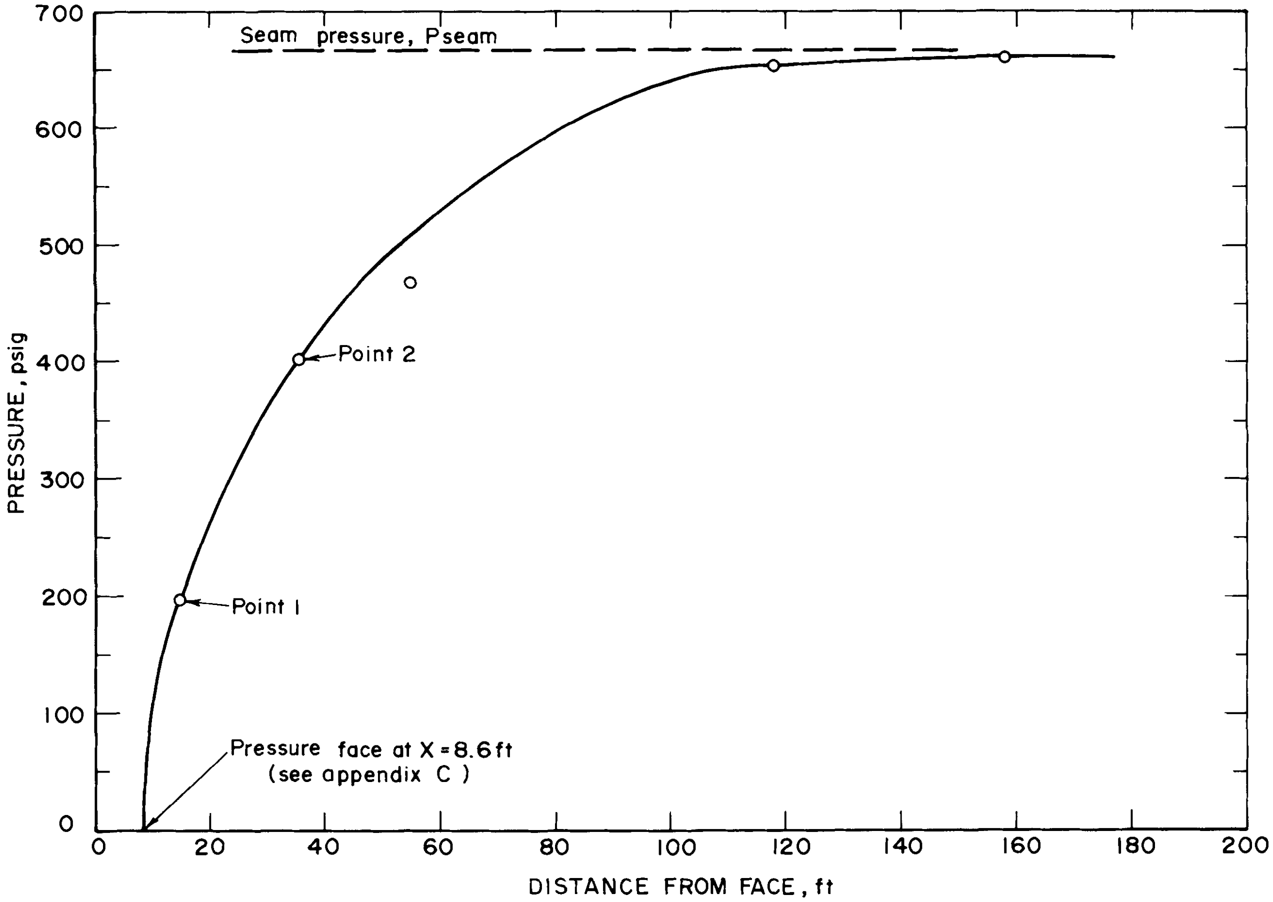


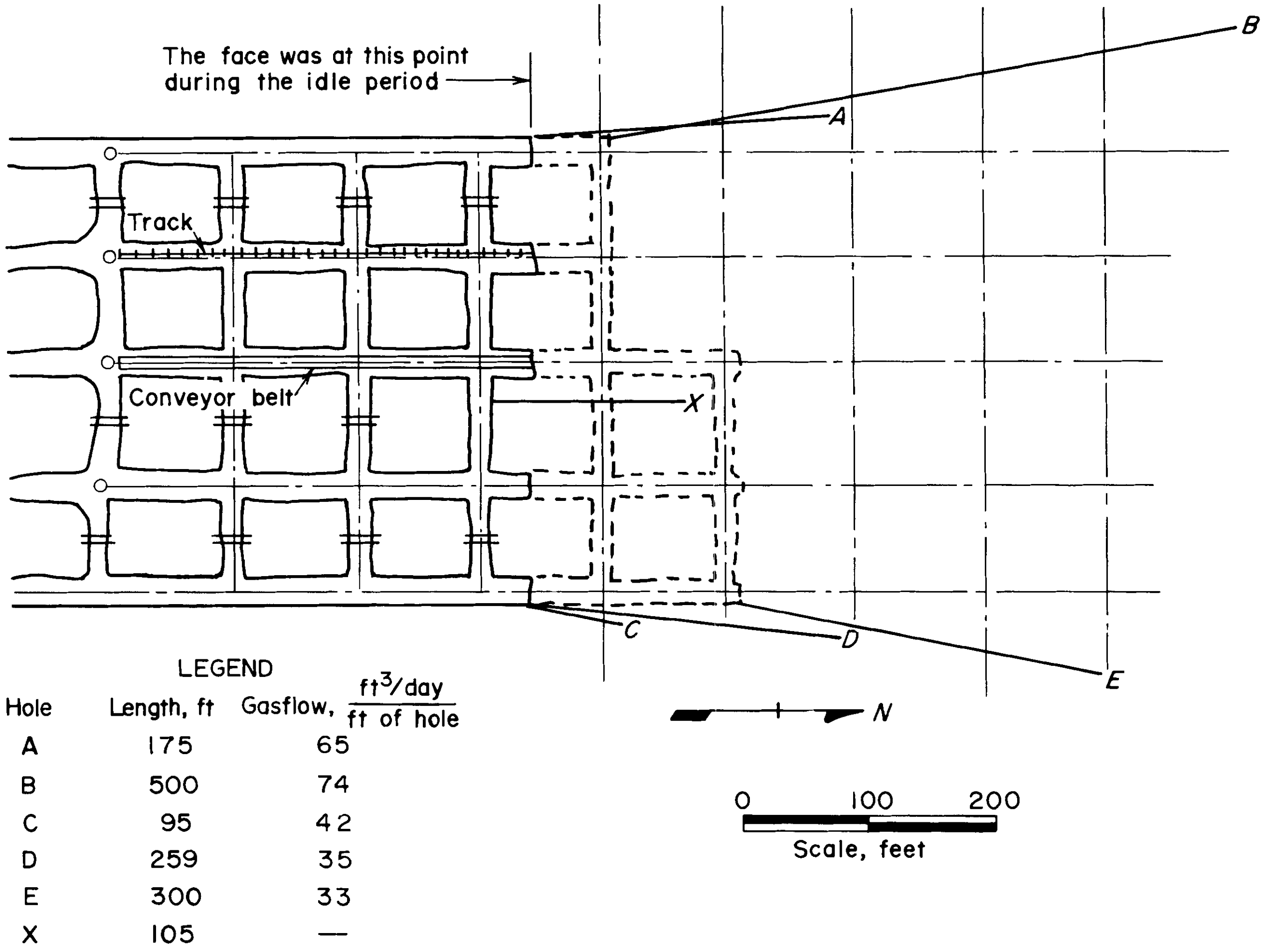
FIGURE 4. - Pressure Versus Distance Taken 360 Hr After Mining.

after mining is shown in figure 4. From figure 4 alone, the ratio $\frac{\text{permeability}}{\text{sorption capacity}}\ \frac{K}{S}$ may be estimated if it is assumed that flow from the coalbed follows the Darcy equation for unsteady-state flow of gases (1-2, 8)

$$\frac{\partial^2 P^2}{\partial x^2} = \frac{\eta S}{KP}\ \frac{\partial P^2}{\partial t}$$

The symbols are shown in appendix A. Details of the calculation are shown in appendixes B and C and $\frac{K}{S}$ = 0.57 millidarcy (md).

Next, the permeability K may be estimated independently using the slope of figure 4 at 0 psig and the observed emission of 74 cfm. This calculation is shown in appendix C. The permeability K obtained is 0.32 md.



LEGEND

Hole	Length, ft	Gasflow, $\frac{ft^3/day}{ft\ of\ hole}$
A	175	65
B	500	74
C	95	42
D	259	35
E	300	33
X	105	—

FIGURE 5. - Location of Horizontal Boreholes.

Since both $\frac{K}{S}$ and K are known, a sorption capacity S of 0.56 may be calculated. This is much higher than coal porosity, which mercury intrusion studies (14) have shown to be around 5 or 10 pct. However, coal has a great capacity for adsorbing gas. At 650 psi, 1 g of finely powdered Pocahontas coal can absorb at least 20 cm^3 of gas. This corresponds to a sorption capacity of 0.59, and thus a measured value of 0.56 for solid coal is not unreasonable.

It is possible to obtain a rough check on these values of K and S by calculating the expected emission from a horizontal borehole drilled into the working face. During the idle period mentioned previously, several holes were drilled horizontally into the coal to drain out methane (fig. 5). For a 250-ft hole the measured gas flow rate ranged from 10,000 to 15,000 ft^3 per day. The calculated flow using $K = 0.32$ md and $S = 0.56$ is shown in appendix D.

It is approximately 32,000 ft^3 per day. This is reasonably close, considering how gross many of the original assumptions are.[4]

It is obvious that all of the numbers obtained are very approximate. Gas flow calculations for coal mines always will have some degree of error because of the difficulty of incorporating all the relevant variables. For example, in figure 5 it may be seen that boreholes A and B on the left side of the face gave about twice the methane emission rate as boreholes C, D, and E on the right. No quantitative method exists yet to evaluate the geological factors that cause this difference.

METHANE FLOWS FROM THE WORKING FACE WHEN THE MINE IS IN OPERATION

Although consideration of flows when the mine is idle allows calculation of coalbed properties, a working mine is of more interest. Figure 6 shows the methane emission from the face area during four typical mining cycles--two before and two after the idle period. The curves are irregular because the machine was not cutting continuously. During mining, peak methane flows of 200 to 250 cfm were recorded regularly and occasionally increased to 400 cfm.

Several distinct features of the curves in figure 6 are evident. First, the background flow increases from the average 74 cfm during the idle period to flows ranging from 90 to 150 cfm or more, depending on whether mining is being done at the beginning or end of the week. Second, the operation of the mining machine adds another 60 to 120 cfm on top of this increased background.

This considerably higher methane flow compared with that during the idle period may be attributed to three sources:

1. Degradation at the face and subsequent emission from the mined coal as it is carried away,

2. Increased emission from the high-permeability crushed zone as it is being advanced into regions of high gas pressure, and

3. Increased emission from the intact coalbed behind the crushed zone because mining has advanced the crushed zone and created a steeper pressure gradient.

The 60 to 120 cfm over the increased background flow may be attributed to sources 1 and 2; that is, degradation at the face and advance of the high-permeability crushed zone. On the other hand, the background increase itself may be attributed to source 3.

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[4] It is possible that $\frac{K}{S}$ is slightly less than the 0.57 md value given. One of the holes drilled into the face (fig. 5, hole X) at the beginning of the idle period was sealed with inflatable packers and the pressure measured was 649 psig. Twelve days later (still during the idle period) the pressure had fallen only 13 psi. Calculations similar to those given in appendix B would indicate $\frac{K}{S} = 0.4$ md. This also would make the calculated borehole flow rate somewhat lower, and thus in better agreement with the measured values.

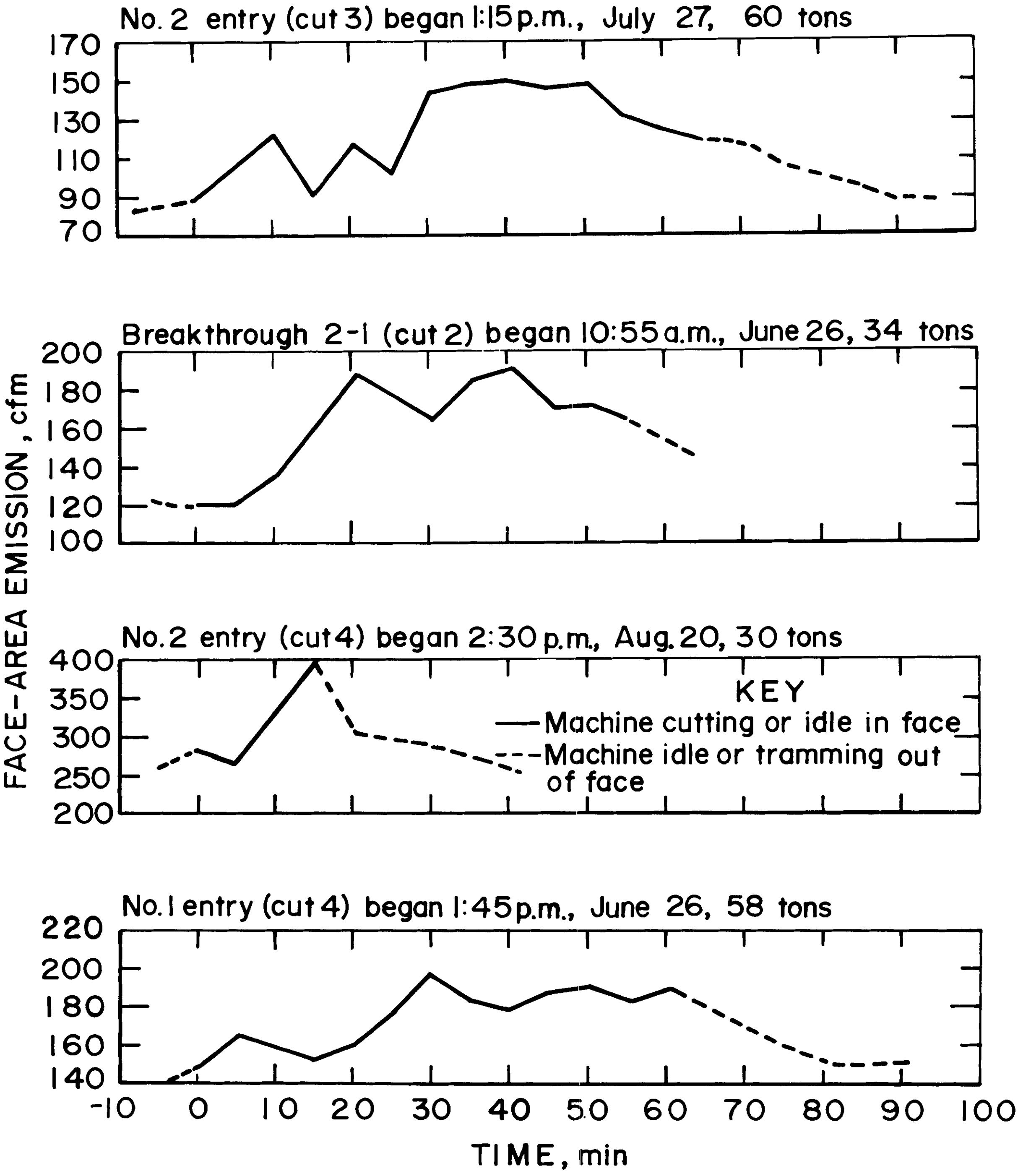

FIGURE 6. - Methane Flow From the Face Area During Cutting.

Using these three sources as a basis, it is possible to approximately account for the flow rates shown in figure 6 in terms of the various properties of the coalbed; that is, the pressure curve shown in figure 4, the desorption curve for Pocahontas No. 3 coal given in figure 2, and the calculated permeability and sorption capacity.

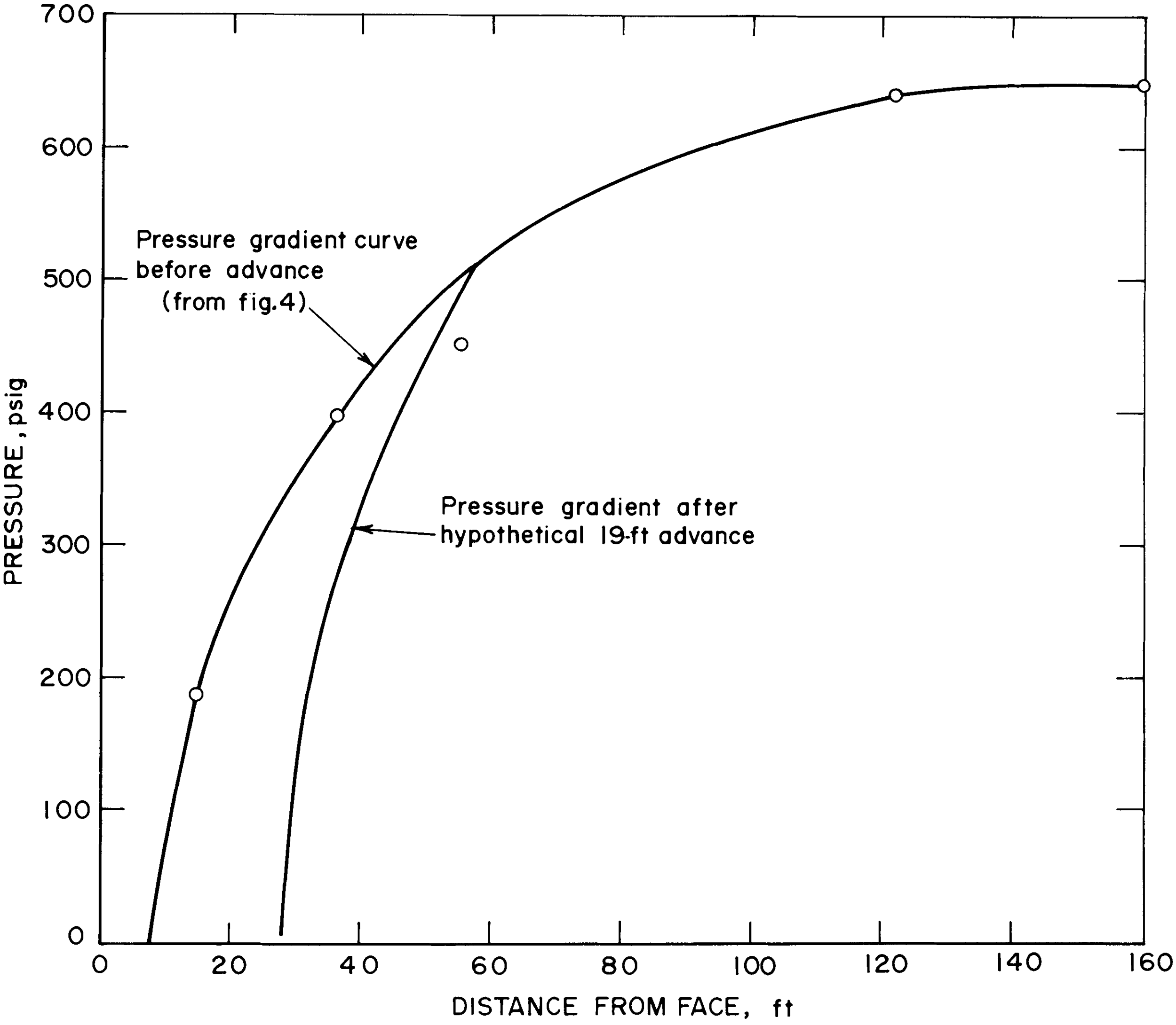

FIGURE 7. - Pressure Curves Before and After a Hypothetical 19-Ft Advance.

Source 3 may be estimated as follows: the average cut shown in figure 6 yielded about 50 tons of coal. This results from an advance of 19 ft in a 4- by 16-ft heading. Figure 7 is figure 4 redrawn to show the effect on the pressure curve of a hypothetical 19-ft advance. The slope at the "pressure face" increases by a factor of 1.6.[5] Given the same permeability, 0.32 md, the flow from the pressure face would be about 74 x 1.6 = 120 cfm. Thus the coalbed behind the crushed zone contributes 120 cfm.

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[5] The 1.6 is obtained by fitting the theoretical curve to the hypothetical pressure curve after a 19-ft advance. The procedure is the same as that given in appendix B for the curve in figure 4, except that $\frac{K}{S} = 0.57$ is used and t_m is calculated. In this case, t_m is not the time since mining, but simply becomes a parameter. Next, the slope at the pressure face is calculated using the equations detailed in appendix C.

The contribution from sources 1 and 2 (degradation at the face and advance of the crushed zone) may be approximated as follows: The area between the curves in figure 7 was integrated graphically to obtain the amount of gas to be found in the coal. It is assumed that the advance was made in 1 hr, the sorption capacity of the coal was the value obtained previously, 0.56, and that complete desorption takes place. This gives an emission of 280 cfm. This volume is considerably higher than the 60 to 120 cfm emission rate actually measured in the mine. Most of the error probably is due to the assumption that complete desorption occurs in 1 hr (appendix B). Data given by Airey (1) and Perkins (fig. 2) would indicate that larger lumps are 25 pct desorbed in 1 hr, and if this is correct, an average emission of 70 cfm would result.

At present there is no way to separate the contributions of sources 1 and 2, or to obtain a more accurate value for the percentage desorbed without additional experimental information.

The results for a typical mining cycle may be summarized as follows:

Source	Measured emission (from mine), cfm	Estimated emission (theoretical), cfm
1. Degradation at face	60 to 120	70 (assuming 25-pct desorption).
2. Advance of crushed zone		
3. Emission from seam behind crushed zone	90 to 150	120

It can be seen that the observed emission rates can be theoretically justified to an approximate degree. It appears that during mining the crushed zone plays a more important role, than during the idle period, but still is not the main source of gas.

COMPARISON WITH OTHER COALBEDS

There is some evidence that the Pocahontas No. 3 coalbed may not be as permeable as other U.S. coalbeds. Horizontal boreholes in the Pittsburgh bed have given gas emission rates 10 times higher than the Pocahontas No. 3 (11), even with a lower gas pressure. Another study (9) has shown that the increase in methane emission resulting from mining a face in the Pittsburgh bed is typically about 10 cfm. This is considerably lower than the measured 60 to 120 cfm shown from sources 1 and 2. It is likely that the permeability of the Pittsburgh bed is considerably higher, perhaps by a factor of 50. Also the crushed zone is likely to be much less extensive.

CONCLUSIONS

The various sources of methane in the Pocahontas No. 3 coalbed have been estimated for an advancing room-and-pillar-mining operation. Consideration of gas flows during 2 weeks when the mine was idle allow the calculation of the coalbed permeability and sorption capacity. Also, it can be concluded that the crushed zone is not the main source of gas, whether during mining or not. Even though the results are rather crude, the analysis nevertheless provides a framework for more detailed and more accurate approximations in the future.

Even at the present state, the analysis indicates what sort of methane control scheme is likely to be successful in this coalbed. For instance, the low permeability obtained, 0.32 md, indicates that vertical wells may not provide a very efficient means of degasifying large areas. It indicates that drilling horizontal holes directly into the face will be much more effective than drilling into just the outside entries (5). Also, the analysis indicates that when the mining machine is operating in this coalbed, the major source of gas is within 50 ft of the machine.

It appears that Cervik's suggestion that most of the methane in U.S. coalbeds comes from within the intact coalbed is partly correct for the Pocahontas No. 3 seam. His suggestion may be more correct for other beds such as the Pittsburgh coalbed.

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[6] Titles enclosed in parentheses are translations from the language in which the item was originally published.

APPENDIX A.--NOTATION

P = pressure (atm).

Q = methane emitted (cm^3).

t = time (sec).

P_1 = pressure at which Q is measured (atm).

A = face area (cm^2).

η = viscosity of methane (cp).

x = distance into coal seam from working face (cm).

K = permeability (darcy).

$\left(\frac{dP}{dx}\right)_{xp=0}$ = face gradient $\left(\frac{atm}{cm}\right)$.

$\bar{P}$ = average pressure (atm).

$P\ (x,t)$ = pressure in the coalbed at point x and time t (atm).

P_{seam} = gas pressure in the undisturbed bed (atm).

x_o = pressure face.

x_p = distance from pressure face to point x in seam = $(x-x_o)$ (cm).

t_m = time since mining (sec).

ρ = gas density $\left(\frac{g}{cm^3}\right)$.

r_w = radius of borehole (ft).

r = drainage radius of borehole (ft).

h = length of borehole (ft).

S = sorption capacity $\left(\frac{\text{volume of gas adsorbed per atmosphere}}{\text{total volume of coal}}\right)$.

APPENDIX B.--CALCULATION OF PERMEABILITY AND SORPTION CAPACITY

It is assumed that methane flow in the coalbed follows the unsteady-state Darcy equation for gas flow, and it is assumed that the bed is a homogeneous semi-infinite slab; the end of the slab being the coal face. Before mining, the slab is at uniform pressure P_{seam}. At the time of mining, the end of the slab (face) is reduced to atmospheric pressure.

It also is assumed that the desorption from the pore structure of the coal is fast compared with the time required for the gas to flow to the face.

The equation is

$$\frac{\partial^2 P^2}{\partial x^2} = \frac{\eta S}{KP} \frac{\partial P^2}{\partial t},$$

and an approximate solution is (8)

$$\frac{P^2_{(x,t)} - P^2_{seam}}{P^2_{atm} - P^2_{seam}} = \operatorname{erfc} \frac{1}{2t_d^{1/2}},$$

where erfc signifies complementary error function and erfc = 1 minus the probability integral. Values of the probability integral may be obtained from standard tables.

Also above, t_d = dimensionless time = $\frac{t_m \bar{P} K}{\eta x_p^2 S}$.

$\frac{K}{S}$ is obtained by fitting the error function curve to figure 4 using the following values:

t_m = 360 hr.

$\bar{P}$ = 340 psia.

P_{seam} = 680 psia.

η = 0.012 cp.

From this, $\frac{K}{S}$ = 0.57 md.

In figure 3, x_o = 8.6 ft. The next step is the calculation of the gradient at the pressure face. If the error function curve is differentiated the result is:

$$\frac{dP^2}{dx} = \frac{P^2_{seam}}{\sqrt{\dfrac{\pi K \bar{P} t_m}{\eta S}}} \exp\left(- \frac{x_p^2 \eta S}{4K\bar{P}t_m}\right)$$

At the pressure face $x_p = 0$, so

$$2P\frac{dP}{dx} = \frac{P^2_{seam}}{\sqrt{\dfrac{\pi K \bar{P} t_m}{\eta S}}}$$

From this, $\frac{dP}{dx} = 0.5 \frac{atm}{cm}$. The Darcy equation is:

$$\frac{QP_1}{At} = \frac{K}{\eta} P \frac{dP}{dx} .$$

If $\frac{dP}{dx} = 0.5 \frac{atm}{cm}$, A = face area = 2.23×10^6 cm^2 and $\frac{Q}{t}$ = flow (at $t_m = 360$) = $63 \frac{ft^3}{min} = 3 \times 10^4 \frac{cm^3}{sec}$, then K = 0.32 md.

The Desorption Problem

Coal is known to have a system of fine pores called micropores, which are approximately 10 A in diameter, and also a system of larger pores (or cracks) called macropores, which range in size from micropore dimensions up to micron size. Most of the methane in coal is adsorbed on the surface of the micropores. When coal is mined, the coalbed is progressively exposed, and the gas pressure in the bed falls. Methane then migrates from the micropores into the larger pores and cracks that lead to the working area of the mine. This may be viewed as a two-step process. First, methane diffuses from the micropores into the cracks, and second, the methane flows through the cracks (by laminar or Darcy flow) to the working area.

If the diffusion step is assumed to be rapid, compared with the second step, then the gas adsorbed in the coal micropore structure will be in equilibrium with the gas pressure in the cracks, and the second step will be the rate-controlling step. However, if the micropore diffusion coefficient is very small the first step will be slower, and the pressure gradient curve and the face emission will be affected. Some evidence has been presented that the diffusion coefficient is quite small; that is,

$$\left(D = 10^{-10} \text{ to } 10^{-13} \frac{cm^2}{sec}\right)$$

notably by P. G. Sevenster (12), P. Zwietering (15), and F. S. Karn (7). However, the coefficients given by Sevenster and Zwietering are based on a BET surface area measurement of powders, not on an area calculated from the particle diameter. This leads to a low value for the diffusion coefficient. The disks used by Karn were tested only if they were sound; that is, the coefficient was low due to absence of microcracks. The point is that a typical piece of coal has microcracks even though these may not be immediately evident.

If the diffusion coefficient for powders is based on a surface area calculated from the particle radius (which is much smaller than the BET area), the coefficient is about 10^{-8} cm^2 per second (6). Bureau of Mines measurements on disks have given 5×10^{-7} cm^2 per second. The existence of a basic microcrack structure has been confirmed by Airey. Although even 10^{-7} is not large, the presence of a microcrack structure makes the area through which the gas may diffuse quite large. Thus it is assumed the desorption and diffusion from the pore structure into the microcracks is fast enough to maintain equilibrium, and the second step is the rate-controlling step. The Darcy equation is then written for this second step. It should be noted that the high value of 0.56 obtained for sorption capacity indicates that the assumption is reasonable in this case.

It appears that the diffusion is not fast enough to completely desorb lumps in an hour if the pressure is changed abruptly. Data from Airey (1) and Perkins indicate that lumps over 1/4-in size are approximately 25 pct desorbed in an hour, and so this 25 pct factor is used in estimating sources 1 and 2.

APPENDIX C.--CALCULATION OF PERMEABILITY

Because the high-permeability crushed zone is present, the gradient exactly at the working face will not give a correct value for coalbed permeability. Rather, a hypothetical "pressure face" is established. This is the point in the coal seam where, for the purpose of calculation, the pressure is assumed to be 1 atm. This point generally is about 10 ft back into the seam from the working face. It should be noted that the distance in appendix B is x_p--the distance to the pressure face, not the "working face."

The pressure face is obtained as follows:

Two measured pressures from the mine are selected, for example, points 1 and 2 in figure 4.

The value for the pressure face x_o is such that

$$\left(\frac{K}{S}\right)_{point\ 1} = \left(\frac{K}{S}\right)_{point\ 2},$$

since

$$\frac{K}{S} = \frac{t_d\ \eta x_p^2}{t_m\ \bar{P}},$$

then $(t_d\ x_p^2)_{point\ 1} = (t_d\ x_p^2)_{point\ 2}$. It then can be shown

$$x_o = \frac{x_2 - Bx_1}{1 - B},$$

where B

$$= \frac{\left(\frac{1}{2t_d{}^{1/2}}\right)_{point\ 2}}{\left(\frac{1}{2t_d{}^{1/2}}\right)_{point\ 1}}.$$

APPENDIX D.--EXPECTED FLOW FROM HORIZONTAL BOREHOLES

If the Darcy equation is solved for unsteady-state flow from a well or borehole, the result is (8)

$$Q_T = \frac{2\pi S r^2_w h}{1{,}000\ P}\left(P_{seam} - P_{atm}\right)Q_t ,$$

where Q_t is a dimensionless total production number given by

$$Q_t = \frac{\int_0^{t_d}\left[\frac{dP}{d(r/r_w)}\right]_{r=r_w} dt_d}{P_{seam} - P_{atm}} .$$

Here $t_d = \frac{2.634 \times 10^{-4}\ t\ \bar{P}}{r^2}\frac{K}{S}$. If t = 10 hr, r = 7 ft, r_w = 1.5 in, and $\frac{K}{S} = 0.574$, then $Q_T = 32{,}000$ ft^3 per day.

It should be noted that this equation applies for radial drainage only. If the coal seam is 4 ft thick, drainage is no longer radial when r exceeds 2 ft. Thus these results are only very approximate. The actual flow for r = 7 ft would be less than the calculated flow. This has been observed, as the actual flow rate was some 10,000 to 15,000 ft^3 per day.