ADVANCEMENT OF MINE VENTILATION NETWORK ANALYSIS FROM ART TO SCIENCE

VOLUME IV SENSITIVITY OF LEAKAGE AND FRICTION FACTORS

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by

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16. Abstract

In this report, the available information on friction factors and leakage characteristics are reviewed and summarized. The use of Penn State Mine Ventilation Simulator (Volumes II and III of the Final Report) to analyze the sensitivity of the ventilation system to the changes in leakage and friction factors is discussed with examples. In simple ventilation systems, the results of changes in leakage and friction factors can be easily solved and explained by the laws of air-flow. In complex systems, with fixed quantity branches and multiple sections, the effect of changes in leakage and friction factors cannot be generalized. This study points out the importance of good input data for obtaining reliable information from computer analyses and demonstrates the application of simulators to analyze the effectiveness of ventilation systems. An extensive bibliography on friction factors, leakage characteristics and ventilation economics is included.

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FOREWORD

This report was prepared by The Pennsylvania State University under USBM Contract No. H0133040. The contract was initiated under the technical direction of PMSRC, with Mr. Edward D. Thimons acting as the technical project officer. Mr. Frank Naughton was the contract administrator for the Bureau of Mines.

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I. INTRODUCTION

1.1 General

The single most critical factor directly affecting the health and safety of workers engaged in the process of mining coal underground is the mine environment. This is broadly defined as the space in which man works when underground, and includes both the physical and chemical conditions of the surrounding enclosure and the nearby mining equipment. Mine ventilation, which essentially deals with mine atmospheric environment, is a subclassification of this broad field. The basic objectives of mine ventilation are to supply adequate fresh air to the working places, to dilute and carry away the dust and toxic gases, and to maintain the humidity and temperature in the mine within desirable limits. To achieve these objectives, it is sometimes necessary to course intake air through several miles of underground airways.

In the last three decades, coal mining in this country has witnessed an unprecedented deployment of large and powerful machines resulting in mines with highly concentrated workings and large production from relatively few mines. The combination of all these factors has resulted in increased gas and dust liberation at the working faces. At the same time, machine bulk and the need to maintain necessary auxiliary functions make ventilation planning and implementation very difficult. In compliance with the Federal Coal Mine Health and Safety Act of 1969, a minimum velocity of 60 feet per minute in working places is required and respirable dust levels (minus ten microns) cannot exceed two milligrams per cubic meter of air. Methane accumulations are not allowed to exceed one percent in the working areas. The regulations further stipulate that the primary ventilation system must deliver at least 9,000 cfm of fresh uncontaminated air to the last open cross-cut with 3,000 cfm required at the face.

Good ventilation, therefore, is essential for efficient mine operation. On one hand, available ventilation facilities impose certain restrictions on production level. On the other hand, ventilation possibilities and requirements cannot be defined other than in relation to a production plan. Consequently, ventilation and production planning are interdependent; and experience has shown that the development of larger machines for higher production rates calls for increased sophistication in ventilation planning. Moreover, over the last two decades, ventilation standards have risen steadily and stringent regulations have been enacted. In view of these factors, it is evident that the primary ventilation system must be carefully designed and implemented.

1.2 Scope of the Report

Volume II and III of the final report have discussed the mathematics of mine ventilation network analysis, and have presented a computer program for analyzing mine ventilation problems. The program serves as an experimental mine wherein analysis of proposed alternatives for ventilation planning can be evaluated. The accuracy and usefulness of the results from this program, however, depend on the input parameters specified which, in turn, are essentially the ventilation system engineering factors. These embody parameters such as the friction factor of the roadways, leakages through stoppings, designated paths and interconnections through which the air must flow, and the operating characteristics of the fan in the system.

Of paramount significance is the fact that a mine ventilation system is dynamically changing with mining operations; new roadways and stoppings are constantly added to or deleted from the overall network. This causes changes in the fan operating point, and in the air distribution within the system. Roof and floor pressures have significant effects over time on the condition of the roadways, and on the ventilation control devices (stoppings, air-crossings, doors and regulators). These in turn may affect the surface characteristics and cross-sectional areas of the roadways and the leakage characteristics of the control devices.

In the report, the available information on friction factors and leakage characteristics is reviewed and summarized. The use of a network analysis computer program (Didyk, et al 1977) to analyze the sensitivity of the ventilation system to the changes in leakage and friction factors is discussed. A section on the economics of mine ventilation systems is also presented.

II. FRICTION FACTORS

2.1 General

The problem of coursing air through a mine divides naturally into two parts. The first concerns the fan and the second, the mine airways through which the fan must course the air current. The pressure head developed by a fan is expended in overcoming the resistance of the mine to airflow. The pressure losses in the system may be divided into two groups:

(1) Losses caused by the friction between the exposed airway surface and the air streams: These losses primarily depend upon the characteristics of the individual wall surfaces, and length, width and height of the roadways.

(2) Shock losses: These are cuased by changes in roadway cross-sectional area and/or airflow direction.

Friction losses in straight sections of approximately uniform area account for the major pressure losses in almost any mine ventilation system, accounting for 70 to 80% of the total pressure loss sustained. All formulas for friction pressure losses include an empirical factor determined by actual experiments. This factor is commonly called the "coefficient of friction" or "K-factor". The Atkinson formula for pressure drop, developed in 1854, is still the basic relationship between the quantity of air flowing in a roadway, its dimension and wall characteristics, and the head loss in the roadway (Atkinson, 1854):

$$H_f = (KLPQ^2)/(5.2A^3) \qquad (2.1)$$

where

H_f = Head loss due to friction in inches of water gage

K = Friction factor for the roadway in $lb\text{-}min^2/ft^4$

L = Length of roadway in ft

P = Perimeter of the roadway in ft

Q = Quantity of air flowing in ft^3/min, and

A = Area of the roadway in ft^2

Defining $R = \frac{KLP}{5.2A^3}$, the equation (2.1) can be stated as

$$H_f = RQ^2 \tag{2.2}$$

where R = Resistance of the roadway. For a rectangular roadway of height H, and width, W, both in ft, the formula (2.1) can be written as:

$$H_f = \frac{KL[2(H+W)]\ Q^2}{5.2\ (HxW)^3} \tag{2.3}$$

Substituting for the quantity Q, velocity V in ft/min, where V = (Q/A), equations (2.1) and (2.3) can be rewritten as:

$$H_f = \frac{KLPV^2}{5.2A} \tag{2.4}$$

$$H_f = \frac{KL[2(H+W)]\ V^2}{5.2\ [HxW]} \tag{2.5}$$

To determine the value for friction factor in practice, long straight sections of roadways with minimal dimension changes are selected, and the dimensions and length are accurately measured. The velocity of the air and head loss in the sections are then determined as precisely as possible. The value of the friction factor 'K', can then be calculated using any one of the above formulas.

2.2 Previous Studies

There is relatively little published information in the United

States in recent years on investigations of pressure losses due to friction in mine ventilation systems. The prioneering works of Greenwald and McElroy (1929), and McElroy and Richardson (1927) are still the standard references in ventilation system design and engineering.

During 1923 and 1924, McElroy directed a large number of carefully conducted experimental determinations of friction factors for mine airways - a few in coal mines (in Indiana) and a relatively large number in metal mines (at Butte in cooperation with the Anaconda Copper Mining Co.). At the same time experiments were also conducted in the Bureau of Mines' experimental coal mine at Bruceton, Pennsylvania under the direction of Greenwald (1924-1927).

Using the Butte results as a "frame work" and on a further amplification of all the available data, McElroy and Richardson (1927) developed a table of friction factors applicable to both coal and metal mines. Table 2.1 shows the Bureau of Mines schedule of friction factors for mine airways.

In the later 1920's Callen and Smith (1926-1929) published their measurements of air quantities and energy losses in mine airways. They claimed that the energy losses were proportional to the quantities flowing in the ventilation network. McElroy (1935) concluded that Callen and Smith's work did not call for changes in the friction factor table reported by the U. S. Bureau of Mines.

As early as 1916 (Sturrow, 1916-17), research was underway in England to determine conditions under which the square law of headloss ($H_f=RQ^2$) would hold and when it should be replaced by a direct relationship ($H_f = RQ$). The objective of the research was to study the behavior of

TABLE 2.1

Bureau of Mines Schedule of Friction Factors for Mine Airways[1]

Type of airway	Irregularities of surfaces, area, and alinement	Values of K[2]											
		Straight			Sinuous or Curved								
					Slightly			Moderately			High degree		
		Clean (basic values)	Slightly ob-struct-ed	Moder-ately ob-struct-ed	Clean	Slightly ob-struct-ed	Moder-ately ob-struct-ed	Clean	Slightly ob-struct-ed	Moder-ately ob-struct-ed	Clean	Slightly ob-struct-ed	Moder-ately ob-struct-ed
Smoothlined	Minimum--------	10	15	25	20	25	35	25	30	40	35	40	50
	Average--------	15	20	30	25	30	40	30	35	45	40	45	55
	Maximum--------	20	25	35	30	35	45	35	40	50	45	50	60
Sedimentary (or coal)	Minimum--------	30	35	45	40	45	55	45	50	60	55	60	70
	Average--------	55	60	70	65	70	80	70	75	85	80	85	95
	Maximum--------	70	75	85	80	85	95	100	95	100	95	100	110
Timbered (5-foot centers)	Minimum--------	80	85	95	90	95	105	95	100	110	105	110	120
	Average--------	95	100	110	105	110	120	110	115	125	120	125	135
	Maximum--------	105	110	120	115	120	130	120	125	135	130	135	140
Igneous rock	Minimum--------	90	95	105	100	105	115	105	110	120	115	120	130
	Average--------	145	150	160	165	160	165	160	165	175	170	175	195
	Maximum--------	195	200	210	205	210	220	210	215	225	220	225	235

1-McElroy, G. E., Engineering Factors in the Ventilation of Metal Mines: Bureau of Mines Bull., 385, 1935, p. 43.

2-All values of K are for air weighing 0.075 lb/ft^3. Values in Table 2.1 are expressed in whole numbers but must be multiplied by 10^{-10} to obtain the proper K value.

Note: Recent research have come up with less conservative estimates for K; however, additional work needs to be done in this area before these are recommended for use. (Kharkar, 1973).

air leakages through gob areas, and to minimize such leakage to reduce the possibility of spontaneous combustion. McElroy (1926-1935) in reviewing all published data and some experimental work on head losses in mine airways concluded that friction factors are, in fact, affected by the condition of flow (velocity) in mine airways and that the change in the resistance coefficient was greatest for smooth lined airways in the mine air flow range.

Other researchers have attempted to correlate the change in Reynolds number to a deviation from the square law of head loss.

The existence of departures from the square law has been demonstrated quantitatively in laboratory models by Sales and Hinsley (1951-52) and Brown and Hinsley (1957). They found that for Reynolds numbers below 5×10^4 the friction coefficient (and hence the quantity exponent) varied considerably while at Reynolds numbers greater than 10^5 the friction coefficient was substantially independent of the Reynolds number and the quantity exponent was constant with a value of 2. The departure from the square law was used by Briggs (1931-32), Low (1956) and Peascod and Keane (1955) who, in their treatments of leakage flows across a gob area, used a direct relationship ($H_f = RQ$) between headloss and quantity.

Hodkinson and Leach (1959) reported on measurements of pressure losses made in main airways over a wide range of airspeeds while investigating the mixing of respirable dust with air. Their studies, while confirming that there can be an appreciable deviation from the square law of air-flow resistance, also showed that the quantity exponent didn't change with airspeed and, therefore, is not a function of

Reynolds number. Teply (1970), reporting on similar underground measurements, found two regions around which mathematical expressions could be derived to approximate the friction coefficient. In the region with low Reynolds numbers, the friction factor against Reynolds number curve could be approximated by a second order polynominal. In the region of high Reynolds numbers, the oscillating curves that were observed were approximated by straight lines. His data unfortunately covered only a small number of points and a limited range of Reynolds numbers. From this brief review it can be seen that the relation existing between Reynolds number, airspeed, and the quantity exponent is not quite clear.

Where fully developed turbulent flow does not exist, the theoretical treatment is to retain the square law of head loss but to adjust the value of the friction factor according to known Reynolds number against friction factor curves. However, it is often more convenient in practice to allow for such variations by restating equation (2.2) as

$$H_f = RQ^n$$

where n = quantity exponent whose value lies in the range 1.8 to 2.2 for ventilated places underground (McPherson, 1974). In the majority of cases, though, the assumption of square law for ventilation planning is adequate. The PSU/MVS program (Didyk, et al, 1977) has provisions by which for every branch in the network, the value of the quantity exponent can be specified.

Though resistance to flow offered by mine airways supported with timber sets has been an object of study over the last half century, no accurate relation has yet been established from which prediction of the resistance coefficient of a projected timbered airway can be made.

McElroy (1935) observed that the coefficient of friction K was maximum when the timber support sets were 5 ft apart (center to center), its value falling for both decreasing and increasing spacing. The phenomenon is physically understandable if the timber sets are considered to offer shock resistance in addition to the frictional resistance of the bare airways.

In 1951-52, Sales and Hinsley carried out studies on the resistance of a model airway supported with three- and four-piece drift sets of round and square timber of different sizes. The model airway had an internal section (inside plywood lining) of 8 x 10 inches so that it represented a 1:12 reduction of a prototype airway. They found that for Reynolds numbers ranging between 50,000 and 300,000, the resistance coefficient for a given type of set with a given spacing was constant and independent of the Reynolds numbers of flow except with round timber where the resistance coefficient showed a fair degree of variation. Mishra (1971, 1974) has developed resistance relationships on the basis of model studies for airways supported by timber sets.

In 1954, Taylor proposed a new method for the calculation of values for the friction factor for air flow in deep mines. Starting from thermodynamic equations, Atkinson's formula and gas laws, and assuming that temperature varies linearly with depth, a function was derived to find the friction head loss. The proposed formula was

$$K = \frac{0.3\ A^3 F}{BLM^2\ [V_1 + V_2]^2} \qquad (2.6)$$

where

K = Friction factor in $lb\text{-}min^2/ft^4$

A = Area of cross section of airway in ft^2

F = Friction head loss in ft (or ft-lb/lb)

B = Area of rubbing surface of air ft^2/ft length

L = Length of airway in ft

M = Weight of dry air flowing lb/min (mass flow)

V_1 = Apparent specific volume in ft^3/lb dry air at the beginning of the airway

V_2 = Apparent specific volume in ft^3/lb dry air at the end of the airway

The most common practice still is to calculate the value for "K" or the 'coefficient of friction' by using the Atkinson's relationship described earlier. Whenever corrections are to be made for 'shock losses' in calculating for friction factors, the following relation is commonly accepted (Hartmann, 1961):

$$R = \frac{H_L}{Q^2} = \frac{KP[L + L_e]\,Q^2}{5.2\,A^3.Q^2} = \frac{K[L + L_e]P}{5.2\,A^3} \qquad (2.7)$$

where L_e = equivalent length of a straight airway in ft which accounts for the shock losses and H_L = head loss in the roadway in inches of water gage. This method of determining shock loss expresses each loss in terms of equivalent length of a roadway of same cross-sectional area and shape, and permits a single calculation of the overall head loss for a given airway. This procedure is generally recommended for all routine ventilation calculations, selecting values of L_e from a table developed for this purpose (Hartmann, 1961). The values in this table (Table 2.2) are developed for an airway with a coefficient of friction, K, of 100 x 10^{-10} and a hydraulic radius, R_H, of 2 ft. Hartmann (1961) gives the following guidelines for use of the values in Table 2.2:

TABLE 2.2

Equivalent Lengths for Various Sources of Shock Loss, In Feet

(After Hartman, 1961)

Bend, acute, round	3
Bend, acute, sharp	150
Bend, right, round	1
Bend, right, shart	70
Bend, obtuse, round	0.5
Bend, obtuse, sharp	15
Doorway	70
Overcast	65
Entrance	3
Discharge	65
Contraction, gradual	1
Contraction, abrupt	10
Expansion, gradual	1
Expansion, abrupt	20
Splitting, straight branch	30
Splitting, deflected branch (90°)	200
Junction, straight branch	60
Junction, deflected branch (90°)	30
Mine car or skip (20% roadway)	100
Mine car or skip (40% roadway)	500

1. Values of L_e from Table 2.2 need not be corrected for K or R_H.

2. With a change in area (splitting not involved), include the shock loss in the airway section following the change. This also applies to a bend in conjunction with area change. Separate values are provided for shock losses at entrance and discharge.

3. At splits and junctions in airways, use only the portion of the total flow involved in a change of direction or area. Values from Table 2.2 assume an even division of flow and allow for bend and area change. Include loss at split or junction within the pressure drop for the particular branch.

4. Judgment must be exercised in making proper allowance for unusual sources of shock loss (e.g., obstructions).

5. Values from Table 2.2 are sufficiently accurate for all routine work. Calculate L_e by formula (2.8) for exact determinations.

For more precise calculations, such as would be required for a computer simulation of a proposed ventilation system, the following formulas should be used (Hartmann, 1961):

$$L_e = \frac{5.2\ wR_H X}{K(1098)^2} = \frac{3240\ R_H X}{10^{10} K}\ \text{ft} \qquad (2.8)$$

where w = air density, lb/ft^3

K = coefficient of friction in $lb\text{-}min^2/ft$

R_H = hydraulic radius in ft (area/perimeter)

X = Shock loss factor

The shock loss factor, X, may be calculated using the formulas given in Appendix Table A-3 in Hartman (1961).

Vujec (1970) defined two methods for the determination of the air-flow resistance. In the first method, the resistance can be determined accurately enough simply by measurement and calculation: the ventilating pressure (H) and flow rate (Q) are measured and the resistance calculated from the relationship $H = RQ^2$. The ventilating pressures can be measured by probe or a barometer. The flow rates must be measured very carefully because they have the greatest influence on the equation. Vujec, in the second method advocates that when a ventilation network is being planned the resistance coefficient must be determined from measurements under similar mine conditions.

In the U. S., Kharkar et al (1974) felt that it was necessary to develop new figures for friction factors or at least to test the validity of using the older values for new, fast moving, roof-bolted coal mines. On the basis of a survey of five mines in Pennsylvania, West Virginia, and Virginia and 108 roadway segments, they developed two friction factor tables. (Table 2.3 and 2.4). The new friction factor tables when compared with McElroy's table (Table 2.1) reveal the following:

(1) No change in basic values for smooth lined roadways.

(2) Decrease in 'K' values by 7 to 18 percent in the coal (sedimentary) group.

(3) Decrease in 'K' values by 5 to 20 percent in the timbered roadway group.

These lead to the conclusion that there is a need to do additional work on friction factors for present day coal mines.

The second classification, Table 2.4, shows that 'K' values for intake track roadways are slightly higher than that for bolted return

TABLE 2.3

Friction Factors for Coal Mine Airways Value of $K \times 10^{-10}$

TYPE OF AIRWAYS	STRAIGHT			CURVED		
	CLEAN	SLIGHTLY OBSTRUCTED	MODERATELY OBSTRUCTED	CLEAN	SLIGHTLY OBSTRUCTED	MODERATELY OBSTRUCTED
SMOOTH LINED	25 10, 15, 20	28 15, 20, 25	34 25, 30, 35	31 25, 30, 25	30 30, 35, 40	43 40, 45, 50
COAL SEDIMENTARY ROCK	43 30, 55, 70	49 35, 60, 75	61 45, 70, 85	62 45, 70, 85	68 50, 75, 95	74 60, 85, 110
TIMBERED	67 80, 95, 105	75 85, 110, 110	82 95, 110, 120	85 95, 110, 120	87 110, 115, 125	90 110, 125, 135

Note: Figures underlined are McElroy's min. ave. and max. values of friction factors (Table 2.1).

TABLE 2.4

Friction Factors For Coal Mine Airways Value of K x 10^{-10}

TYPE OF AIRWAYS	STRAIGHT			CURVED		
	CLEAN	SLIGHTLY OBSTRUCTED	MODERATELY OBSTRUCTED	CLEAN	SLIGHTLY OBSTRUCTED	MODERATELY OBSTRUCTED
INTAKE WITH TRACK, BOLTED	31	35	38	39	41	42
INTAKE (TIMBERED)	62	68	71	77	79	81
RETURN BOLTED	25	29	39	35	39	42
RETURN TIMBERED	67	75	82	85	87	90

airways. This may be due to moving trips along the haulage, the effect of additional support (concrete pillars, crossbars, etc.) required on haulage entries and/or an effect of an uneven roof due to more excessive weathering than found in the return air courses. There is a need for classifying the friction factors on this basis in addition to the basic type classification presented in Table 2.3 because all intake or return roadways in the same mine do not have the same 'K' value. Even with the serious limitations on the field studies and the various operating conditions in the mines studied, these results demonstrate the need for friction factor studies.

A recent paper by Rahim et al (1976) has summarized in a tabular form the results of almost all of the available studies in the area of friction factor determination. These tables (Tables 2.5, 2.6, and 2.7) are reproduced here for ready reference. On the basis of several experiments, they have proposed values for 'K' for different types of airways for Indian coal mines. (Table 2.8).

The attention directed to the determination of correct values for friction factors for various mine airway conditions is extremely well deserved since they not only have a direct bearing on mine resistance and air quantity flow but also affect the dispersion of gaseous contaminants (Stefanko et al, 1974, 1977).

2.3 Sensitivity Analysis

In the previous section, it was determined that there are changes in the values for the coefficient of friction. However, it is necessary to know how significant are these changes in values. Field evaluation

TABLE 2.5

Coefficient of Friction Values for Unlined Airways (After Rahim et al, 1976)

Condition of airway	Value of 'K' x 10^{10} lb-min^2/ft^4	Source
Unlined levels:		
Uniform section, no obstruction	20	Clive, Hay and Statham (1939-40)
Straight (1.8 x 2.4m) approximately	37	Greenwald (1929)
Even section driven in coal	28	Clive, Hay and Statham (1939-40)
Irregular section driven in coal	37	"
Regular section, very jagged sides	43	"
Very irregular driven in coal	56	"
Unlined Tunnel:		
Igneous rock	160	Jeppe (1946)
Sedimentary rock or coal	60	"
Unlined airway	40	"
Altofts airway (unlined)	33	Cooke & Statham (1928-29)
Unlined stone drift in coal measure strata	max: 56 min: 48	Mishra (1964)
Unlined galleries:		
Straight normal area	49	Murgue (1893-94)
Straight normal area	46	"
Straight large area	56	"
Straight small area	65	"

TABLE 2.5 (Continued)

Condition of airway	Value of 'K' x 10^{10} lb-min^2/ft^4	Source
Sedimentary rock or coal:		
Straight and clean	min: 30	McElroy
	Av.: 56	&
	max: 70	Richardson (1927)
Sedimentary rock or coal:		
(Sinuous or curved airways)	min: 40	"
	Av.: 65	
	max: 80	
Untimbered roadways with natural sides	70	Cooke & Statham (1926-27)
Unlined straight airways with fairly uniform sides	65	NCB Bulletin (53/73)
Unlined airways with rough or irregular conditions	85	NCB Bulletin (53/73)
Coal mines:		
Airway driven in rock, across the strike	53	Skochinsky and
Airway driven in rock, along the strike	42	Komarov (1969)
Levels driven in the coal of regular shape without ripping the roof or the floor	211 to 32	"
Level driven in the coal of regular shape with ripping	37 to 42	"
Airways in coal	42	"

TABLE 2.5 (Continued)

Condition of Airway	Value of 'K' x 10^{10} lb-min^2/ft^4	Source
Metal Mines:		
Airways (unsupported) driven along the strike: for angle of dip 60-75°	63	Skochinsky & Komarov (1969)
Airways (unsupported) driven along the strike: for angle of dip 75-90°	53	"
Airways (unsupported) driven across the strike:		
When air moving up the dip (angle of dip 60-75°)	90	"
When air moving up the dip (angle of dip 75-90°)	69	"
When air moving down the dip (angle of dip 60-75°)	116	"
When air moving down the dip (angle of dip 75-90°)	106	"
Unsupported airways in limestone beds	min: 37 max: 70 Av.: 52	Vujec (1970)

TABLE 2.6

Coefficient of Friction Values for Steel Arched Roadways (After Rahim et al 1976)

Condition of steel arch lining	Value of 'K' x 10^{10} lb-min^2/ft^4	Source
Arch girdered roadway	83	Hodkinson and Leach (1959)
Steel arch roadway	39	"
Steel arch girdered airway lined with brick work to spring	22	Clive (1937-38)
Steel arch girdered lined with timber over arches	59	"
Steel arch girdered return gate near face	65	"
Bricked between arches all round	30	Stochinsky and Romara (1969)
Bricked between arches to spring	40	"
Lagged with timber at the back fairly smooth	50	"
Average conditions (main airways)	54	"
Rather rough conditions (gate roads)	65	"
Arches poorly aligned, sagged	80	"
Smooth concrete all round	20	NCB Bulletin (53/73)
Concrete slabs or timber lagging between flanges all round	40	"
Rough conditions with irregular roofs, sides and floors	85	"

TABLE 2.6 (Continued)

Condition of steel arch lining	Value of 'K' x 10^{10} lb-min^2/ft^4	Source
Gangway supported with Alphine E-21	min: 82	Vujec (1970)
or molls supports (steel arch)	max: 106	"
at every 0.6-0.7 m.	Av.: 98	"
Perfectly straight arched roadway	32	Hodkinson and Leach (1959)
Main return arched roadway, with subsidence high air speed	100	"
Main intake, arched roadway	65	"
Nearly straight arched roadways, irregular section, major obstructions	81	"

TABLE 2.7

Coefficient of Friction Value for Timbered Airways (After Rahim et al 1976)

Condition of Airway	Value of 'K' x 10^{10} in $lb\text{-}min^2/ft^4$	Source
Straight, normal area	90	Murgue (1893-94)
Straight, normal area	78	"
Slightly sinuous, small area	127	"
Timbered airway 1.5 m centre:	min: 80	McElroy &
straight and clean	Av.: 95	Richardson (1927)
	max:101	"
	min: 85	
	Av.:101	"
	max:111	
moderately obstructed	min: 95	
	Av.:111	"
	max:122	"
Regular section in coal, heavily timbered	48	Clive, Hay and Statham (1939-40)
Irregular section in coal, heavily timbered	56	"
Rockingham airway, regular section	113	Hay and Cooke (1925-26)

TABLE 2.7 (Continued)

Condition of airway	Value of 'K' x 10^{10} in $lb\text{-}min^2/ft^4$	Source
Altofts airway, timber set 9.6 m apart	64	Cooke & Statham (1928-29)
Altofts airway, timber set 4.8 m apart	80	"
Timbered airway (1.5 m centre)	100	Jeppe (1966)
Airway with slight turns, tight timbering	53	"
Wood bars or steel girders on timber legs	100	NCB Bulletin (53/73)

TABLE 2.8

Coefficient of Friction for Different

Types of Airways

(Rahim et al, 1976)

Class of Roadway	No. of experiments conducted.	'K' Value x 10^{10} in $lb\text{-}min^2/ft^4$	
		Range	Average
1. Unlined/unsupported airways in coal	8	41-67	56
2. Airways supported with steel arches.	13	44-106	81
3. Airways supported with steel arches	7	59-84	74
4. Airways supported with steel beams on concrete walls	4	66-74	70
5. Airways supported with props and bars	6	110-169	137

of the effects of the changes in roadway surface characteristics on mine head and quantity are difficult. It is here that ventilation computer programs can be effectively used. Since air leakages play an important part in mine ventilation systems, and the effect of increasing or decreasing resistance coefficients on leakage also must be considered, the sensitivity analyses are described after a discussion of leakage in mine ventilation systems.

III. LEAKAGES IN MINE VENTILATION SYSTEMS

3.1 General

Considerable attention and importance has been attached in the past and at the present to the performance of the fan by the mine engineers. The relative merits of the selection of a fan of proper size based on its operation at high efficiencies, though important, can be of questionable economic importance and may sink into insignificance as compared to losses in the mine itself. Few indeed have given thought to the ventilation efficiency of a mine, a quantity which is difficult to define and determine because of the several tangible and intangible variables that may have to be included. The previous chapter dealt with friction factors, the technologic and economic importance of which can easily be studied. A second, and by far a more wasteful operating practice is to handle large volumes of air of which a very small percentage reaches the working fans.

Montgomery (1936) presented results from sixteen mines (Table 3.1) to illustrate the point that on the average only 19% of the air handled by the fan reached the last open cross-cuts. In fact, the question of air leakage continues to remain as one of the greatest problems in mine ventilation. The amount of air which escapes through stoppings, doors, air-crossings, and shafts is surprisingly large. In operating mines today, it is not uncommon to lose between 50 to 60 percent of the air between the fan and the last open cross-cut due to leakage directly into

TABLE 3.1

Leakage Characteristics of Mines

Mine No.	Quantity of Air at fan, ft^3	Quantity at last crosscut, ft^3	Pct. reaching last crossing
1	119,533	18,232	15.3
2	120,410	16,737	13.9
3	65,300	6,160	9.4
4	120,000	40,000	33.3
5	64,700	15,600	24.1
6	59,827	7,350	11.7
7	96,000	22,140	23.0
8	50,400	3,840	7.6
9	38,100	12,800	32.2
10	35,290	5,125	14.5
11	77,364	9,600	12.4
12	160,276	37,672	23.5
13	28,130	2,380	8.5
14	90,000	12,000	13.3
15	62,240	10,950	17.6
16	69,400	23,250	33.5

return. Figure 3.1 illustrates the serious effect of air losses on the efficiency of mine ventilation systems. In the working section where men need more fresh air and where the major job of diluting and carrying away gases and dusts is conducted, the effects of leakage can hardly be over emphasized. Comparatively, poorly maintained airways result in greater fugitive air losses causing greater quantities of air to be handled by the fan at higher pressures. This results in higher power costs in addition to even greater losses and dust problems because of the higher velocities wihin the mine ventilation system itself. All control devices are susceptible to leakage, and have a great effect on the intake-return pressure differential, therefore, the design, location and maintenance of these devices will influence the efficiency of the mine ventilation system, and hence should be given equal importance when evaluating any ventilation system. When the cost of fan installation, air shafts, overcasts, stoppings and other ventilation equipment are compared with the small percentage of air actually reaching the working face, a critical evaluation of the factors affecting air leakage becomes all the more important.

3.2 Ventilation Control Devices

Amount of leakage is a function of the pressure difference. Leakage between intakes and returns near the fan is usually the highest. In such situations even a small amount of leakage will result in large operating costs over the life of the mine. As the mine workings advance and panels are worked out, the number of leakage sources (stoppings, doors, overcasts, undercasts, etc.) increase along with the need for increased air quantities. It may be necessary to

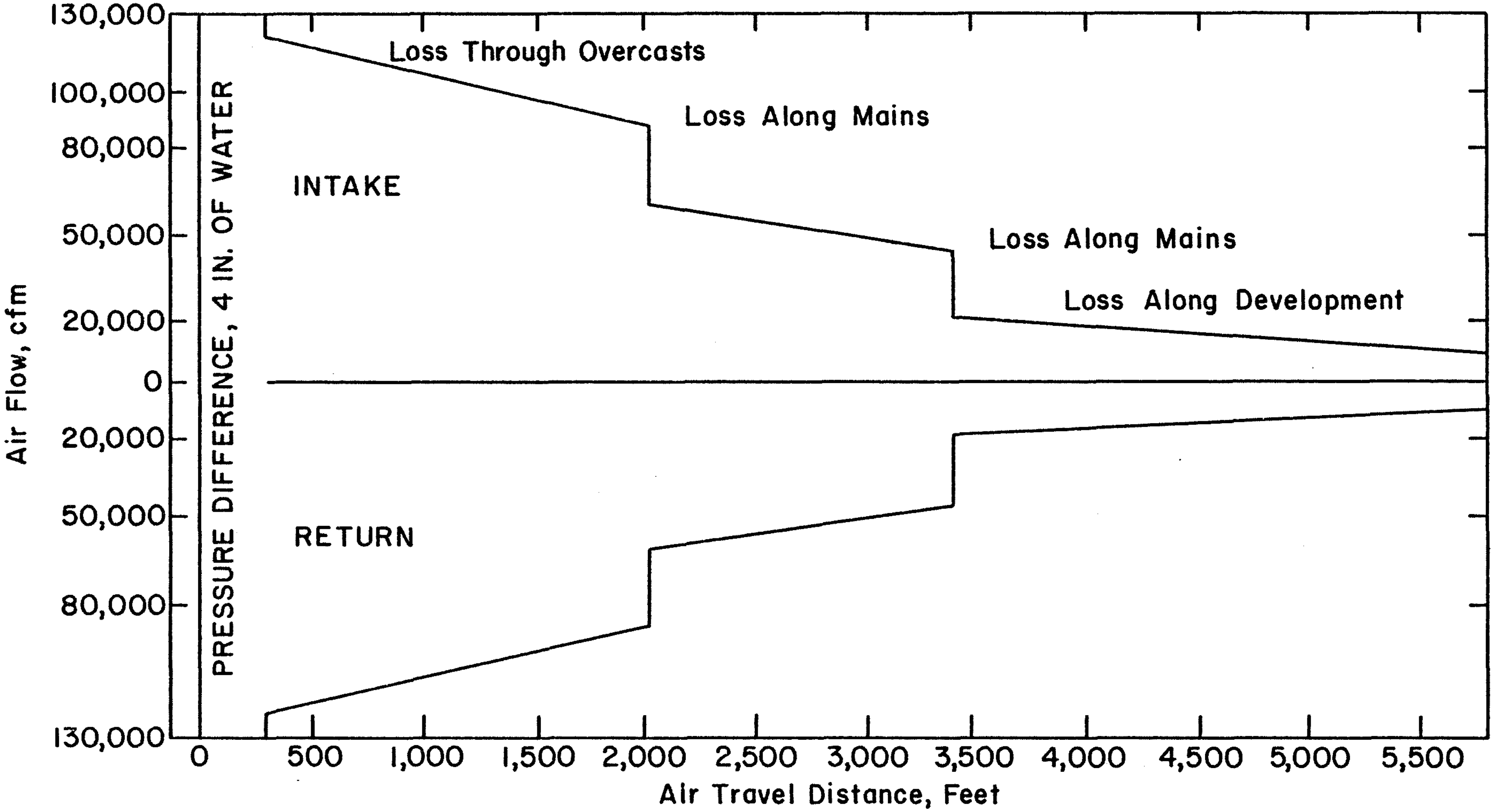

FIGURE 3.1

Illustration of Air Leakage Losses

(After Kingery, 1960)

increase the ventilating pressure continually to insure that the necessary volume of air is reaching the faces. Leakage and recirculation can be largely avoided by a properly planned system, substantial construction and close attention to details. Individual leakages are often insignificant and sometimes difficult to detect, but the cumulative effect of small leakages is by no means insignificant.

Temporary stoppings are often constructed of fire-resistant jute fabric, plastic, rough lumber covered with plastic or even telescoping sheet metal sections. These are extensively used in areas where frequent adjustments to air directions are necessary, such as in the working panels. Where these are constructed of concrete, gravel, cinder, or slag blocks, the blocks are usually wedged in place with dry joints. Such a construction permits fast erection and even allows for the recovery of the blocks. However, their ability to prevent leakage of air is also minimal.

Permanent stoppings are installed in places where a permanent or a long-term control of flow is needed, such as between the main intakes and returns. They should be substantially built so that they are airtight and, if they are functioning as seals or bulkheads, resist the disruptive forces of explosions. For the latter purpose, the stopping must be built according to specifications of the Coal Mine Health and Safety Act of 1969 as given in Title 30, Code of Federal Regulations, Section 75.329-2 entitled Construction of Seals or Bulkheads. Shown in Figures 3.2 and 3.3 are two such constructions, the one in Figure 3.3 being able to withstand greater pressures than the stopping in Figure 3.2. The greater the distance between the two masonary walls, the more effective will be the stopping.

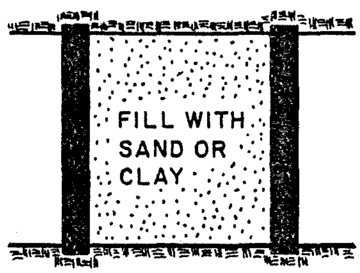

FIGURE 3.2

Well-Constructed Explosion Proof Stopping

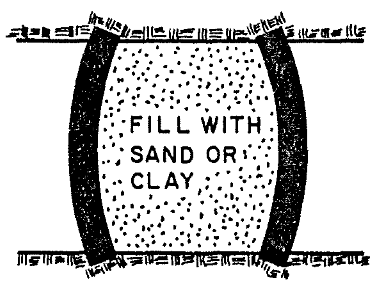

FIGURE 3.3

Excellent Construction for Explosion Proof Stoppings

Stoppings utilizing such construction are rarely seen in mines. Most permanent stoppings generally consist of 8" x 8" x 16" solid or hollow blocks laid with mortared joints. The entire block sides are then coated with cement or some type of block bond. The stopping is usually keyed-into the roof, floor and sides, and this area should be sealed with cement. It is common practice to provide a fire resistant stringer across the top to prevent cracking with the first onset of strata pressure. Since stoppings must be accessible for inspection, and repair, if necessary, it is advisable that they be built where the roof and sides are secured after having been cleared of loose or sloughing overhangs. Additionally, it is also necessary to avoid gobbing rock against the stopping (Figure 3.4) as it makes it difficult for inspection and maintenance, both against structural failure and leakage.

A door is simply a hinged or movable partition within stoppings designed to permit the passage of men and equipment. Mine doors may be constructed of metal (required for doors between the intake and return) or lumber covered with tarpaper, plastic or other sealent material. They serve the same function as stoppings, and are frequently used in haulageways. To avoid short circuiting between the return and intakes, doors should always be arranged in pairs to provide an air-lock so that one will always be closed while the other is open. Automatic self-opening doors are especially useful along haulageways and equipment travelways. However, doors along the haulageways do not have a sill, and therefore, conveyor belting or brattice cloth should be attached to the bottom of the door to minimize leakage.

Because the health and safety of underground employees may depend

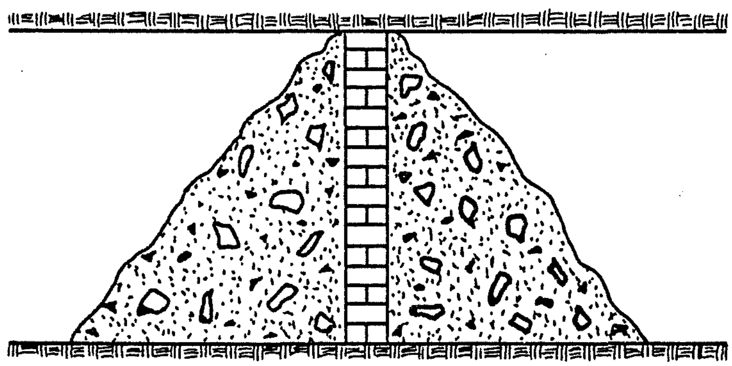

FIGURE 3.4

Gobbing of Rock Against a Stopping

upon the ventilation doors, personnel should be trained in the use of doors. Doors should also be marked in bold letters: KEEP CLOSED or KEEP OPEN, so that the ventilation aspects of doors are not overlooked.

Overcasts or undercasts are air bridges that allow the intake and return air to cross one another without mixing. Undercasts are less frequently used since these are below the general level of the mine and water tends to accumulate in them, causing a drainage problem. Shown in Figure 3.5 is a very poorly constructed overcast. From the discussions in the earlier chapter, it is obvious that across the overcast, abrupt changes in airflow directions (horizontal to vertical-up to horizontal to vertical-down, and then horizontal again) cause severe shock losses. Shown in Figure 3.6 is an ideal overcast, with long sweeping curves at the approach and discharge side. The general cross-sectional area of the overcast should be one-half of the area of the approaching airway, if the approach is gradually contracted, and the discharge is gradually expanded, as shown in Figure 3.6.

The overcast should be properly built with high quality material. In the area where the overcast is required, the roof is heightened to the desired height. Common practice, where continuous miners are used, is to ramp up at the spot where the overcast is located, before returning to bench the coal. It is good to use solid blocks, and set them in mortar. The two wing walls must be anchored firmly in the ribs. Steel beams are then placed on the wing-walls over which the roof is built of blocks set in mortar. Two walls are then built from this roof to the heightened roof line. Figure 3.7 illustrates an inexpensive but good quality overcast. Here a solid concrete wall is built on either side of the airway, and a sufficient number of pipes of large diameter are

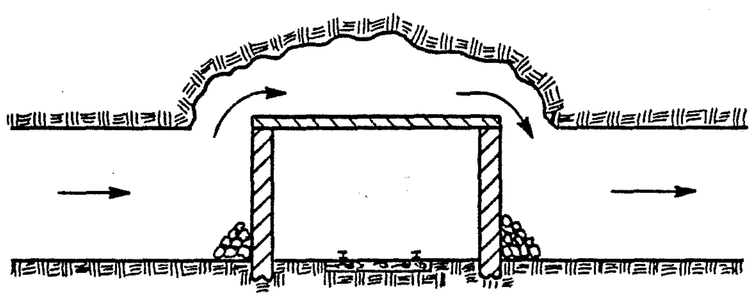

FIGURE 3.5

Poorly Constructed Overcast

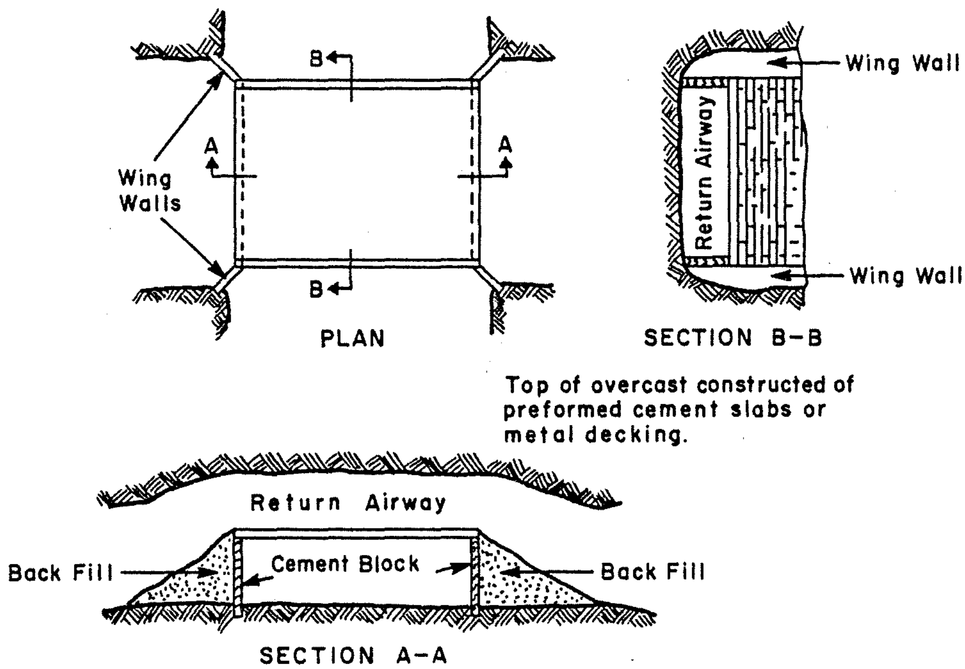

FIGURE 3.6

Excellent Construction for an Overcast

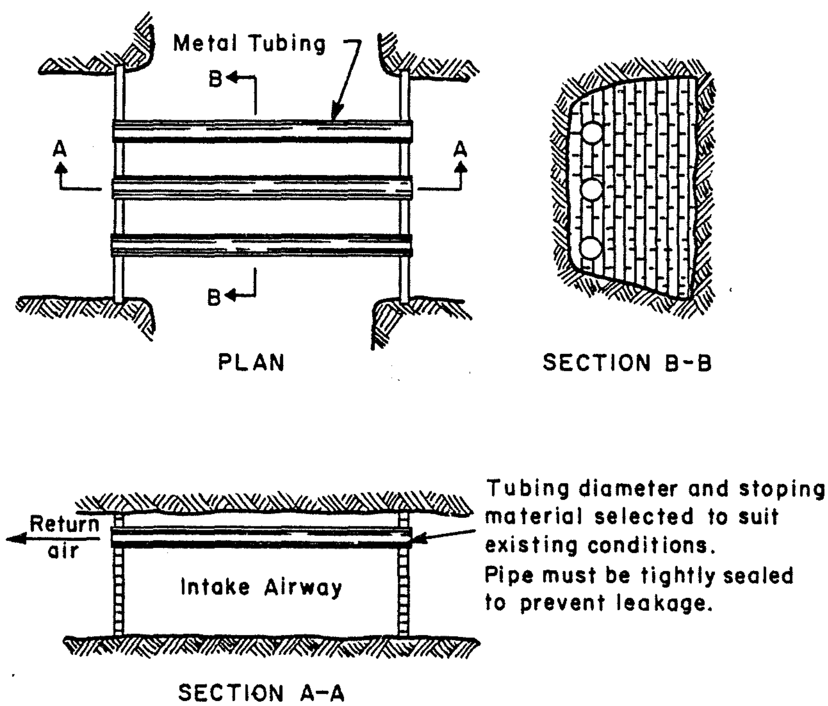

FIGURE 3.7

Inexpensive but Effective Overcast

laid on the top of the walls to carry the air across the main airway. It is possible to carry a split of 25,000 ft^3/min with two 36" or three 30" pipes. It is important to ensure that at the ends the pipes are properly bedded in the concrete and made air-tight. Overcasts, if poorly constructed, result not only in high pressure losses, but also in severe leakage.

In ventilation planning, leakages have been handled by a system of rough allowances since leakages often cannot be measured accurately. Table 3.2 lists some recommended figures for use in mine ventilation planning. Where pressure and volume measurements can be made on either side of the leakage source these measurements are not only more reliable than visual inspection but may also serve as a base for estimating conditions. Leakage through stoppings, doors and regulators not only depends upon the pressure across the control device but also on the condition of the device itself. Ground pressures destroy control devices and before long fugitive air losses become so great that it is almost impossible to provide adequate air at the face. Some empirical relationship which will relate ventilating air pressure, condition and area of stopping and the leakage quantity through the stopping is more useful for ventilation planning than a guess at a constant leakage quantity.

3.3 Previous Studies

There are some well-known studies in the area of mine air leakage. Briggs (1931-32) tried to develop a method of expressing leakage conductance through the use of a 'porosity coefficient' - a factor which stands in relation to leakage in much the same position as does the

TABLE 3.2

LEAKAGE ESTIMATE IN COAL MINES
(Roberts, 1960)

---

1. Leakage across newly formed gobs (longwall faces)

Distance between Intake & Return Gobs (ft)	Leakage Across Gob as Percentage of the Air on the Face
150	20
300	10
600	5

No allowance made where solid stowing is done.

2. Separation doors: 3000 ft^3/min

3. Overcasts: 3000 ft^3/min

4. Surface Leakage: This is the air that leaks through the casing at the top of the upcast shaft. It is a function of fan W.G.

   5" -- 25,000 ft^3/min
   10" -- 35,000 ft^3/min
   15" -- 45,000 ft^3/min
   20" -- 50,000 ft^3/min

5. Leakage through stoppings

   a. Explosion proof -- None
   b. Temporary stoppings: 100 - 200 ft^3/min depending on the age, physical construction and maintenance

6. Fixed air quantity requirement

   a. Headings in coal: 3000 ft^3/min
   b. Hard headings: 25 $ft^3/min/ft^2$ or 5000 ft^3/min
   c. Motor, haulage and Pump House: 5000 ft^3/min
   d. Battery Charging Station: 10,000 ft^3/min
   e. Diesel Locomotive Garage: 100 ft^3/min/hp

7. Adiabatic Expansion in Shaft: allow 1% per 300 ft

8. Air Velocities:

   a. Faces 150 - 500 ft/min
   b. Main Airways 150-1000 ft/min
   c. Shaft - <2500 ft/min
   d. Fan Drift - < 2500 ft/min
   e. Diesel loco shed: > 150 ft/min

---

coefficient of friction to the resistance of airways. On the assumption that the difference of pressure across a stopping (P) is directly proportional to the quantity flowing through it, it is possible to write:

$$f = b\ (P_1 - P_2) = bP \qquad (3.1)$$

where f is the air flow through the stopping, P_1 is the pressure on one side of the stopping, P_2 is the pressure on the other side, and b is a constant, inversely dependent on the tightness of the stopping.

Mancha (1942) has proposed that the ratio of the head losses with and without leakage may be considered proportional to the ratio of the quantities at the two points in the ventilating circuits. While only an approximation, the validity of this assumption has been supported by Holdsworth et al (1951). Mancha (1942) further advocates that a lack of empirical data and knowledge of the condition of individual stoppings makes an exact analysis of underground stopping leakage impossible. Generally, leakage is most severe through old stoppings in the outby portion of the circuit. They are also subjected to higher pressure differences than the newer inby stoppings. Therefore, the circuit air volume diminishes at a decreasing rate progressing from outby to inby in the circuit. Some observers have suggested that the leakage through all of the stoppings be considered equal. This condition applies only in limited cases and is represented by curve B in Figure 3.8. The assumption of uniform stopping leakage is not justified under either theoretical or practical considerations. With stoppings of equal area and equal porosity, uniform stopping leakage presupposes the same pressure differential across each stopping, and this may not be true.

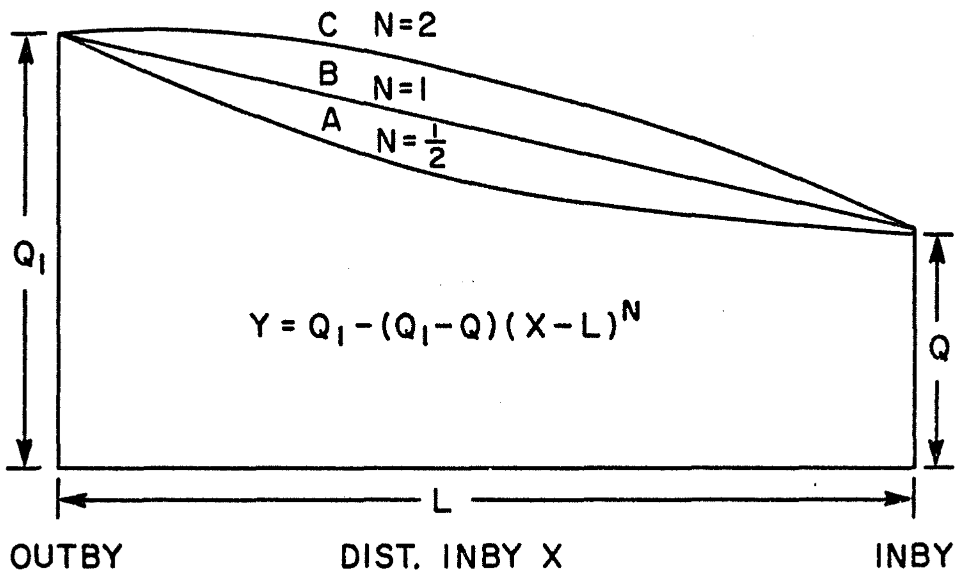

FIGURE 3.8

Leakage Characteristics
(After Mancha, 1942)

Furthermore, careful air-volume measurements taken at regular intervals along any underground circuit with leaky stoppings will always show a circuit volume decreasing at a diminishing rate. When plotted along the circuit the circuit volume will usually be found to lie between curves A and B.

Herbert (1953) presents results of ventilation surveys in 22 mines in the Central Coal Basin in the United States and gives figures for air losses due to leaky stoppings, trap doors, and overcasts. He concluded that the percentage of air loss varies between 8.9 and 70.7 percent. He further observed that in the worst case only 29.3 percent of the fan quantity reached the active workings.

Peascod and Keane (1955) developed a mathematical model for leakage across a gob separating the intake and return aircourses within an advancing longwall section and determined that the quantity losses are greatest in the outby portion of the intake, the first half of the roadway contributing to about 75 percent of the total loss. They further advocate the use of booster fans to reduce leakage in mine ventilation and show that for maximum leakage saving without recirculation the booster fan should produce the same pressure drop as the surface fan. Further, they show that the two similar fans, one used as a main fan and one as a booster at the mid point of the return, reduces the quantity loss due to leakage to one half of the value that will result when the total pressure is developed by one main fan. However, use of booster fans underground is fraught with serious safety hazards, particularly in the event of explosion, fire, etc. and is therefore not allowed in coal mines in this country.

Holland and Skewes (1962) investigated the materials used for stoppings in coal mines, and the methods for stopping construction. While the study did not come up with a mathematical relationship for leakage volume based on stopping material and method of construction, the following conclusions pertaining to leakage were drawn:

1. The porosity of stopping materials bears little relation to the leakage performance of the material in stoppings. Strength of the stopping material, however, may vary approximately inversely as its porosity.

2. The leakage of mortar-laid, unplastered, and unpainted stoppings varies approximately directly as the permeability of the material of which they are constructed. For this reason, therefore, permeability tests may be of value as an indication of suitable materials for stopping construction insofar as stopping leakage is concerned.

3. In every case, plastering the stopping greatly reduced leakage through it. Furthermore, if a stopping is plastered, the material of which it is constructed is secondary to the plaster in preventing leakage. Therefore, it is important that the plastering job be well done if leakage is to be reduced to a minimum. On the average, the leakage of a plastered stopping will be around 10 to 15 percent of a similar mortar-laid unplastered stopping.

4. In every case tested, painting the stopping greatly decreased leakage. As in the case of plastering, the effectiveness of the system is determined by how well the painting is done. On the average, the leakage of a painted mortar-laid stopping will be around 15 to 20 percent of a mortar-laid unpainted stopping. All painting was done with no

pressure differential across the stopping. In practice paint should be applied under the same conditions.

5. Painting and plastering a stopping will decrease its leakage on the average to around 2 or 3 percent of the leakage through a mortar-laid unplastered and unpainted stopping. The exact amount of leakage, however, will be determined by how well the plastering and painting is done.

6. A great variation insofar as permeability and leakage exists between the many different kinds of red dog and cinder concrete blocks. Careful testing is required in order to pick out the blocks having the best characteristics in this regard. Much ventilation leakage can probably be traced to the use of stopping materials that have a very high permeability and for this reason more care should be exercised in their selection and purchase.

7. The pressure differential on stoppings should be varied as little as possible. This is especially true of stoppings constructed of dry stacked and plastered materials.

Kawenski and Mitchell (1963) reporting on their investigations at the Bruceton Experimental Coal Mine to evaluate materials and methods of constructing efficient mine stoppings, claimed that no single type of stopping is superior to others for all the conditions encountered in coal mining (Tables 3.3 and 3.4). In fact they furthered the work of Holland and Skewes (1962) and found that all stoppings leaked air in proportion to the pressure differential across the stopping and expressed the relation by the equation:

$$Q = aP^n \qquad (3.2)$$

TABLE 3.3

Air Leakage through Block Stoppings[1]

Block form	Aggregate	Air leakage, ft^3/min			
		Uncoated		Mortar-coated	
		Dry wall	Wet wall	Dry wall	Wet wall
Hollow core	Cinder	900	300	5	5
Do.	Slag	1,400	300	5	5
Do.	Concrete	900	50	5	5
Solid	Concrete	1,200	20	5	5

[1]Air leakage at one-in. water gage through 100 ft^2 area.

TABLE 3.4

Air leakage through Coated and Uncoated

Sheet Stoppings

	Air leakage,[1] ft^3/min
Stopping construction:	
Brattice cloth:	
Uncoated	35,500
Foam-coated	5
Latex-coated	10
Metal lath:	
Foam-coated	5
Plaster-coated	15
Polyvinyl chloride (uncoated)	5

[1] Air leakage through face of 100 ft^2 stopping subjected to one-in. water gage pressure differential.

where P is the pressure differential across the stoppings in inches of water; a and n are constants for a given stopping. Numerically, 'a' is the leakage quantity in ft^3/min at a pressure differential of one inch water gauge per 100 ft^2 area of the stopping. Values for 'a' ranged from 5 to 35,500 ft^3/min, being lowest for coated stoppings, moderate for dry wall and highest for brattice cloth. The exponent 'n' represents the logarithmic change of air leakage with pressure and ranges from 0.2 to 1.2 (Table 3.5). This research indicated that stopping leakage is a result of several factors:

1. Air leakage increases as the pressure differential increases.

2. For uncoated dry-wall construction, 65 to 95 percent of the air loss is through the joints; the block form or aggregate has little effect.

3. For uncoated wet-wall construction the block form and aggregate are significant. Air leakage through hollow core is 2.5 times greater than that through solid block; air leakage through cinder blocks is six times greater than that through concrete (hollow-core) blocks.

4. Air leakage through brattice cloth is high; leakage through plastic film is low.

5. Air leakage is significantly reduced by coating a stopping with a sealant. The efficiency of a sealant depends primarily on its ability to bridge and seal joints, holes, and the perimeter. A more detailed discussion of this work is reported in Bureau of Mines Report of Investigations 6710 (Kawenski et al, 1965).

Recent testing (Brainard, 1973) of the strength and leakage characteristics of cinder and fly-ash stopping blocks revealed that:

1. Fly-ash blocks have a density near 130 lb/ft^3, and a compressive

TABLE 3.5

Air Leakage Coefficients for Mine Stopping 100 ft^2 in Area

	Coefficients	
	a[1]	n
Stopping construction:		
Cinder block:		
Hollow core, dry-wall	900	0.4
Hollow core, dry-wall, mortar-coated	5	.9
Hollow core, dry-wall, foam-coated	5	.9
Hollow core, dry-wall, masonry paint	8	.8
Hollow core, dry-wall, latex-coated (brushed)	10	.9
Hollow core, wet-wall	300	.7
Hollow core, wet-wall, mortar-coated	5	.9
Hollow core, wet-wall, foam-coated	5	1.0
Slag Block:		
Hollow core, dry-wall	1,400	.3
Hollow core, dry-wall, mortar-coated	5	.9
Hollow core, dry-wall, foam-coated	10	.8
Hollow core, wet-wall	300	.7
Hollow core, wet-wall, mortar-coated	5	.8
Hollow core, wet-wall, foam-coated	5	1.0
Concrete block:		
Hollow core, dry wall	900	.3
Hollow core, dry-wall, mortar-coated	5	.9
Hollow core, dry-wall, foam-coated	10	.8
Hollow core, wet-wall	50	.8
Hollow core, wet-wall, mortar-coated	5	.9
Hollow core, wet-wall, foam-coated	15	.8

TABLE 3.5 (Continued)

	Coefficients	
	a[1]	n
Solid, dry-wall	1,200	.9
Solid, dry-wall, mortar-coated	5	.9
Solid, dry-wall, foam-coated	5	1.1
Solid, wet-wall	20	.9
Solid, wet-wall, mortar-coated	5	1.0
Solid, wet-wall, foam-coated	5	1.0
Sheet stoppings:		
Brattice cloth	35,500	.7
Brattice cloth, foam-coated	5	1.2
Brattice cloth, latex-coated (brushed)	10	.8
Metal lath, foam-coated	5	1.2
Metal lath, mortar-coated	15	.20
Polyvinyl chloride film	5	1.20

[1]Equivalent to air leakage in ft^3/min at one-in. water gage differential.

strength of approximately 1500 psi*. The respective figures for cinder blocks are 100 lb/ft^3 and 1250 psi.

2. Cinder blocks are approximately twice as permeable as fly-ash blocks over the 2" to 4 1/2" watergage differential. Air leakage through the cinder block stopping per 100 ft^2 area for watergage differentials of 2", 2 1/2", 3", 3 1/2", and 4" were 74, 85, 112, 129 and 156 ft^3/min respectively.

Kharkar et al (1974) also studied leakage in eight sections at five mines in West Virginia, Pennsylvania and Virginia. Leakage was determined by indirect measurement of quantities on the intake and return sides whereas the pressure drop across a stopping was determined by an altimeter survey. Stoppings were then classified according to their construction material, and the condition of the roof and floor. Block stoppings were generally made from cinder, slag, and gravel aggregates with or without mortared joints and coatings and were included within the study.

These studies of leakage, though carried out under different conditions of stoppings, when analyzed for leakage characteristics did show significant results. While experimental data were too limited to draw definite conclusions for the eight sections studied, graphs were plotted for cumulative leakage against the number of stoppings. An example of one such graph is presented in Figure 3.9. This graph explains that the rate of air losses is variable over the entire section of the airway, largest values being furthest from the working face and that three quarters of the total loss occurs in the first half of its intake.

For a comparison between the mines and the sections, the results were combined and presented in a graph (Figure 3.10). From this figure it is

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*Psi = pounds per square inch

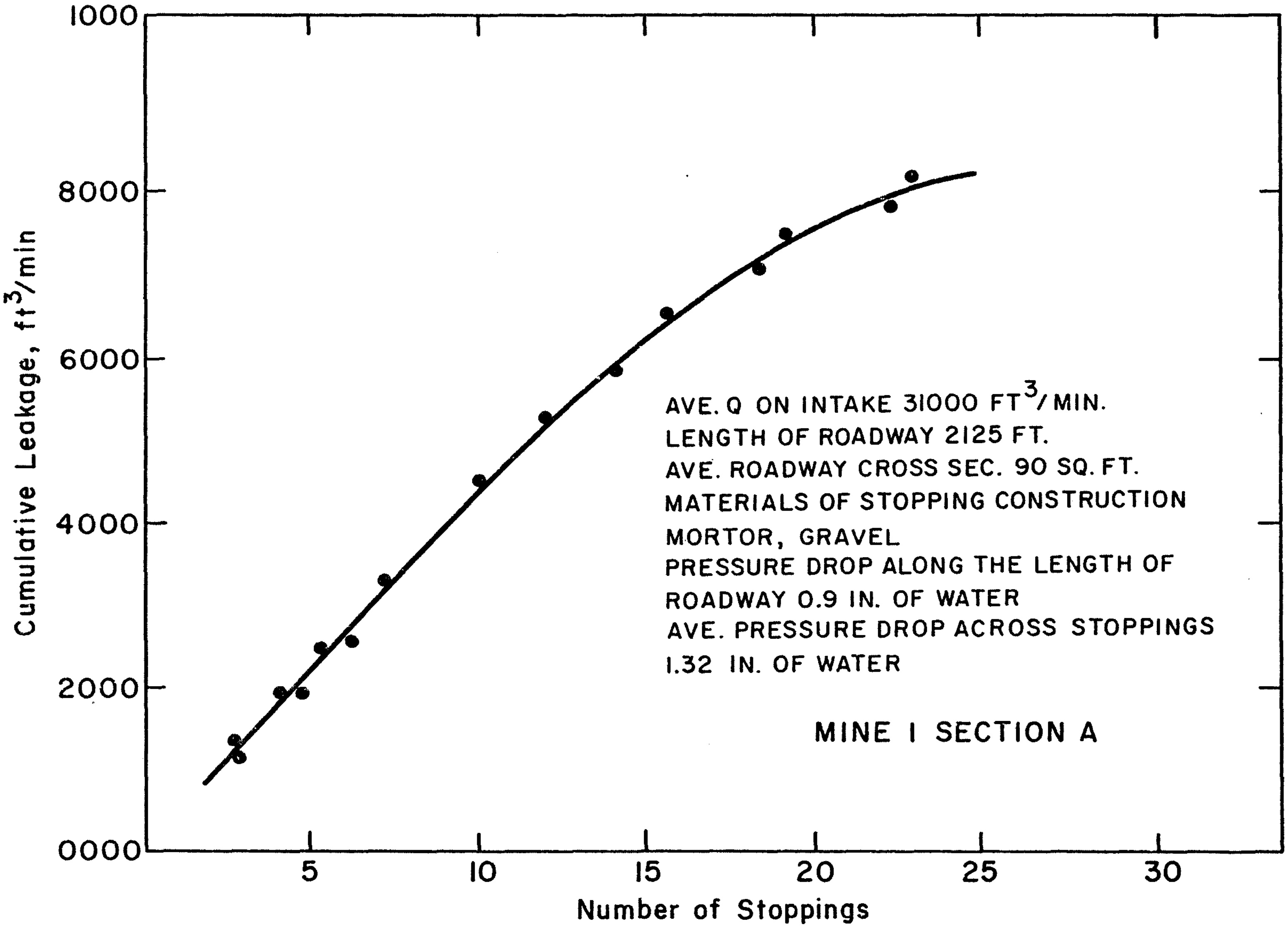

FIGURE 3.9

Graph Showing Cumulative Leakage Versus Number of Stoppings

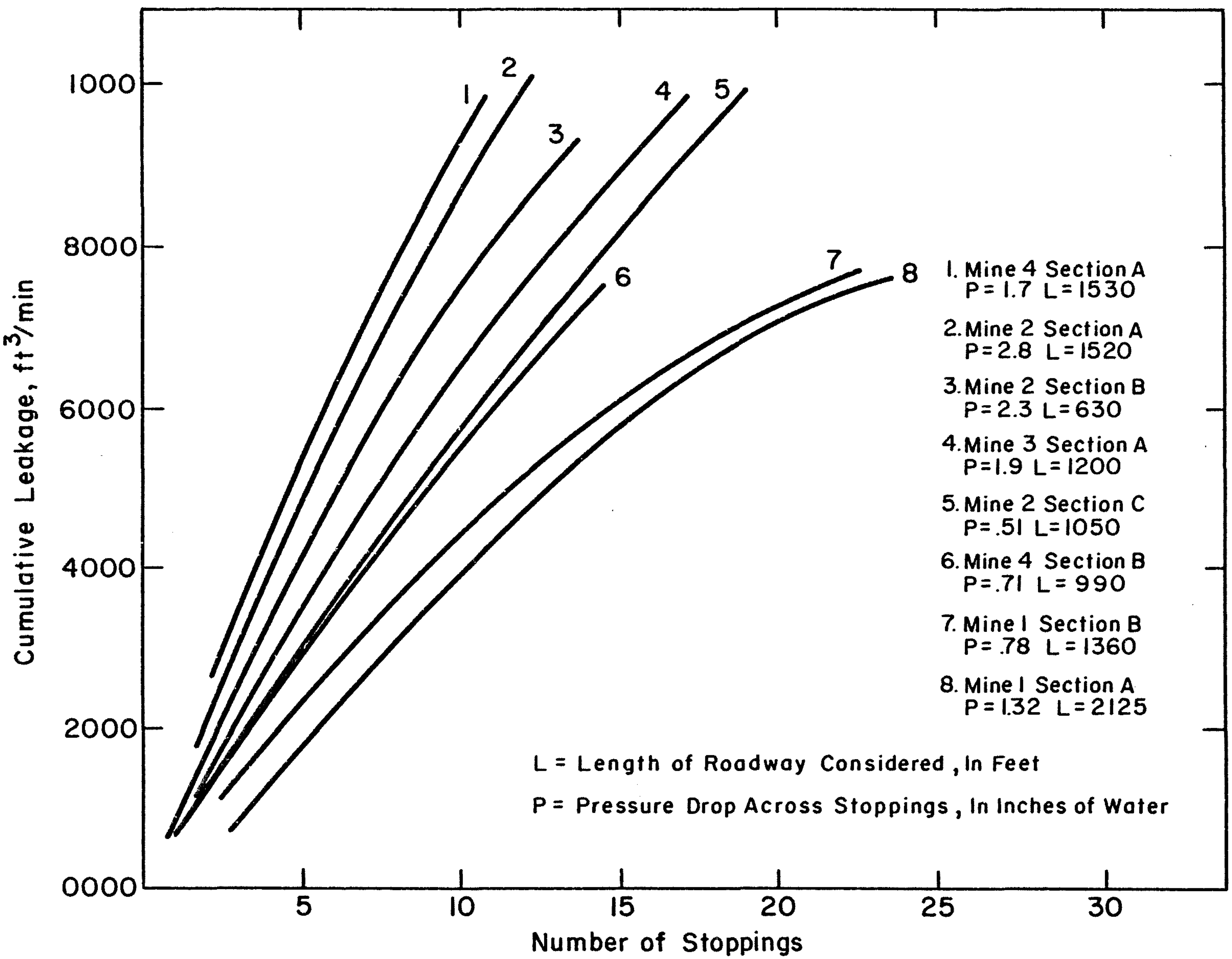

FIGURE 3.10

Graphs Showing Cumulative Leakage versus Number of Stoppings for Different Mines clear that for the same mine different sections have variable leakage characteristics. For example, Mine 4 Section A and Mine 4 Section B are distinctly different with regard to the amount of leakage.

Dalzell (1966) has reported on the performance characteristics, efficiency, and operation pressure of jute line brattice systems on the basis of studies conducted in varying underground systems. In contrast to primary ventilation systems, line brattice or other face ventilation systems (secondary ventilation) rarely exceed lengths over 200 feet. It is not uncommon for air loss per lineal foot of the secondary system to range from 100 to 200 times of the primary system. Saran (1975) has attempted to utilize Dalzell's results by developing a face ventilation simulator which is executed in conjunction with the PSU/MVS. McPherson (1971,1975) has also utilized computer simulation to investigate the intricate air flow patterns in underground metal mine slopes. The leakage of air directly into the returns from the last open cross-cut to the face is an area where little work has been reported.

While this section has dealt at length with stopping leakages and has reviewed studies in this area, there are very few studies in the area of leakage through overcasts, undercasts, doors and air locks. A promising experimental method for studying leakages is provided by Thimons and Kissel (1974). They report on the use of a tracer gas (SF_6) in their work to accurately measure recirculation of return air into intake air caused by leakage through old stoped areas, to trace air "lost" from the intakes, and to check for potential leakage from adjacent mines in areas where mines may be connected through old gobs or stoped-out areas.

3.4 Leakage Considerations in PSU/MVS

From the discussion heretofore it is obvious that proper consideration of leakage in ventilation planning is important. It is a common practice in ventilation planning to assign a leakage quantity to a split, and to use the average quantity in the split [(air at the beginning + air at the end)/2] for head loss calculations, the underlying assumption being one of uniform leakage throughout the split. In most computer programs, however, leakage paths are assumed as fixed quantity branches, and the leakage quantity is dumped from the intakes into the returns at some points along the airway. In effect, leakage sources are treated as point sources, and the amount leaking, as being constant, without any relation to the pressure difference across this source. Experimental results presented before clearly dispute the above methods of assigning leakage.

The PSU/MVS program has facilities by which a leakage branch which may consist of one or more stoppings can be assigned a value (R_L), and an exponent (c) as that the quantity leaking in the branch obeys the law:

$$P = R_L \, Q_L^c \qquad (3.3)$$

where

P = Head across the leakage branch in in. of water

Q_L = Quantity leaking in 1000 ft^3/min

Thus, the quantity leaking through the branch is calculated internally in the program on the basis of the absolute pressures at the two ends of the leakage branches which in themselves are calculated internally on the basis of the fan and the ventilation network resistances. The value for R_L is derived as below. According to the Bureau of Mines research already discussed (Kawenski and Mitchell, 1963), the empirical relationship

$$Q = aP^n \tag{3.4}$$

where

Q = quantity leaking in cfm per 100 ft^2 of the stopping area

a = quantity leaking under 1-in. water gage differential per 100 ft^2 stopping area

n = pressure exponent, and

P = Pressure across the stopping in in. of water

holds true for leakage across a stopping. This relationship can be written as:

$$P = (\frac{1}{a})^{-n} \cdot Q^{-n} \tag{3.5}$$

Defining $c = -n$, we have

$$P = (\frac{1}{a})^c \cdot Q^c \tag{3.6}$$

Considering a stopping of area A ft^2, the above equation for this stopping will be:

$$P = [\frac{1}{a \times (A/100)}]^c \cdot Q^c \tag{3.7}$$

Kharkar et al (1974), have shown that the pressure head across some 10 to 15 consecutive stoppings did not change very appreciably; therefore, making the assumption that the consecutive stoppings are under the same pressure differential, the leakage across each stopping of the same material and construction will be proportional to the area of the stopping. For the case where the stoppings are identical, the above equation can be modified as:

$$P = \left[\frac{1}{a \times \frac{(A)}{100} \times N}\right]^c \cdot Q^c \quad (3.8)$$

where N = Number of stoppings considered together as a leakage branch.

Defining,

$$\left[\frac{1}{a \times \frac{(A)}{100} \times N}\right]^c = R_L, \text{ and } Q = Q_L, \quad (3.9)$$

the equation for leakage can be written as:

$$P = R_L \cdot Q_L^c \quad (3.10)$$

The inputs to the PSU/MVS for a leakage branch are the values for a, A, N, and c. On the other hand, the values for R_L and c can be directly input. For direct input, the value for R_L should be calculated as below:

$$R_L = \left[\frac{10^5}{a \times A \times N}\right]^c \times 10^4 \quad (3.11)$$

If the stoppings were of identical construction but different areas, the formula for R_L can be modified as:

$$R_L = \left[\frac{10^5}{a \times \sum_{i=1}^{N} A_i}\right] \times 10^4 \quad (3.12)$$

where A_i = Area of the i^{th} stopping in ft^2

N = Number of stoppings considered together as a leakage branch.

In using the formulas (3.11) or (3.12) for leakage, the number of stoppings to be considered as one leakage branch should not be too large.

IV. LEAKAGE AND FRICTION FACTOR SENSITIVITY

4.1 General

In this chapter, the effect of leakage and friction factors on fan head, fan quantity, air horsepower and leakage quantity are studied with the use of PSU/MVS. For normal airways in mines, the Atkinson relationship [equation (2.1)] holds:

$$H = \frac{KLPQ^2}{5.2\ A^3}$$

In the preceding chapter, the following relationship was derived for a leakage branch [equation (3.7)]:

$$H = \left[\frac{1}{a \times \frac{A}{100} \times N}\right]^c \cdot Q^c$$

c = Quantity exponent

Values for 'a' and 'c' are derived from those given in Table 3.5 and are a function of stopping material and construction.

Air horsepower is a function of head and quantity:

$$\text{Air Horse Power} = \text{AHP} = \frac{H \times Q}{6350} \qquad (4.1)$$

where H = Head loss in inches of water

Q = Quantity flowing in ft^3/min

Since head and quantity depend on the friction factor and leakage, and since a mine ventilation system is a complex network with numerous parallel and series connections between normal and leakage branches, the behavior of such a system to varying leakage and friction factors cannot be easily identified.

4.2 Sensitivity of Leakage and Friction Factors

For a simple mine ventilation system, shown in Figure 4.1, it can be readily deduced that:

1. If the friction factors of roadways are increased and there is not appreciable leakage through BG and DF, then for the same pressure drop across the circuit, the mine air quantity (Q) is proportional to the inverse of the square root of resistance $\frac{1}{R}$, i.e., $Q \alpha \frac{1}{K}$, where K is the friction factor.

2. If the friction factor of the roadways remain the same, as the leakage increases, the quantity flowing through the mine will increase with an accompanying decrease in pressure drop. However, the magnitude of the increase in leakage will depend on the location of the leakage branches. This is brought out clearly in Figure 4.2 where for identical stopping conditions, the leakage quantity decreases with increasing distance from the intake shaft.

In situations where both factors and leakages change, as is often the case in operating mines, it is difficult to identify the exact nature of relationships. Therefore, the ventilation layout in Figure 4.1 was analyzed with a range of values assigned to leakage and friction factors.

The intake and return, connecting the 500-ft wide face are 3,000-ft long. The entries are 7.5-ft high and 15-ft wide. Two leakage branches are considered. Each leakage branch consists of 15 stoppings, each stopping having an area of 110 ft^2. The system utilizes an exhaust fan with the characteristics shown in Figure 4.3.

4.3 Friction and Leakage Factor Sensitivity Analysis

The range of values for friction factor considered was from 40 to

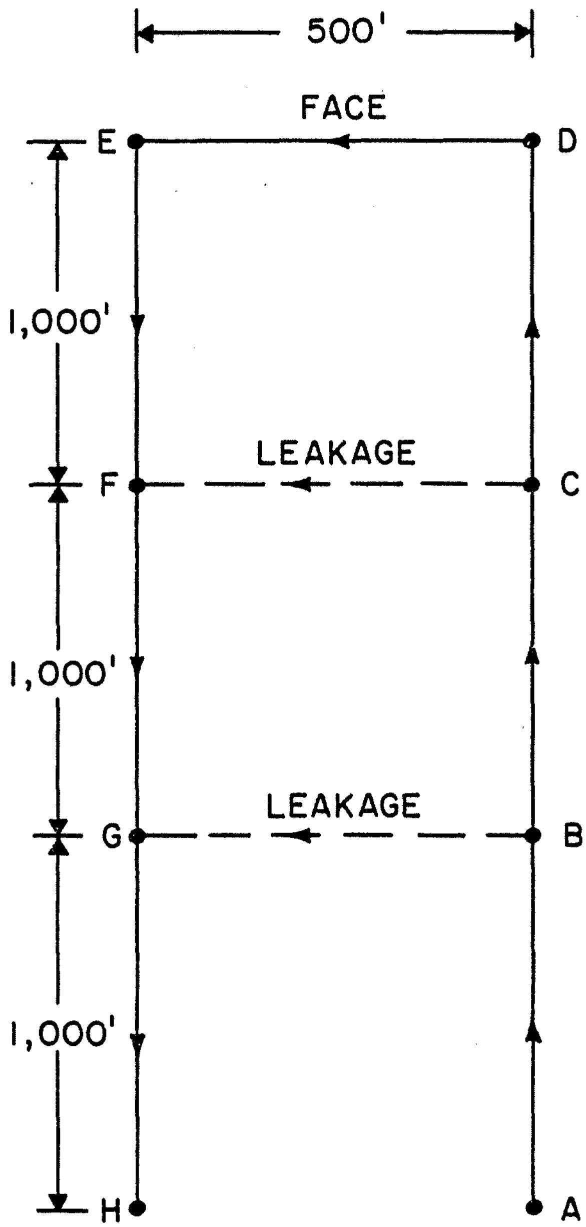

FIGURE 4.1

A Hypothetical Mine Ventilation Network

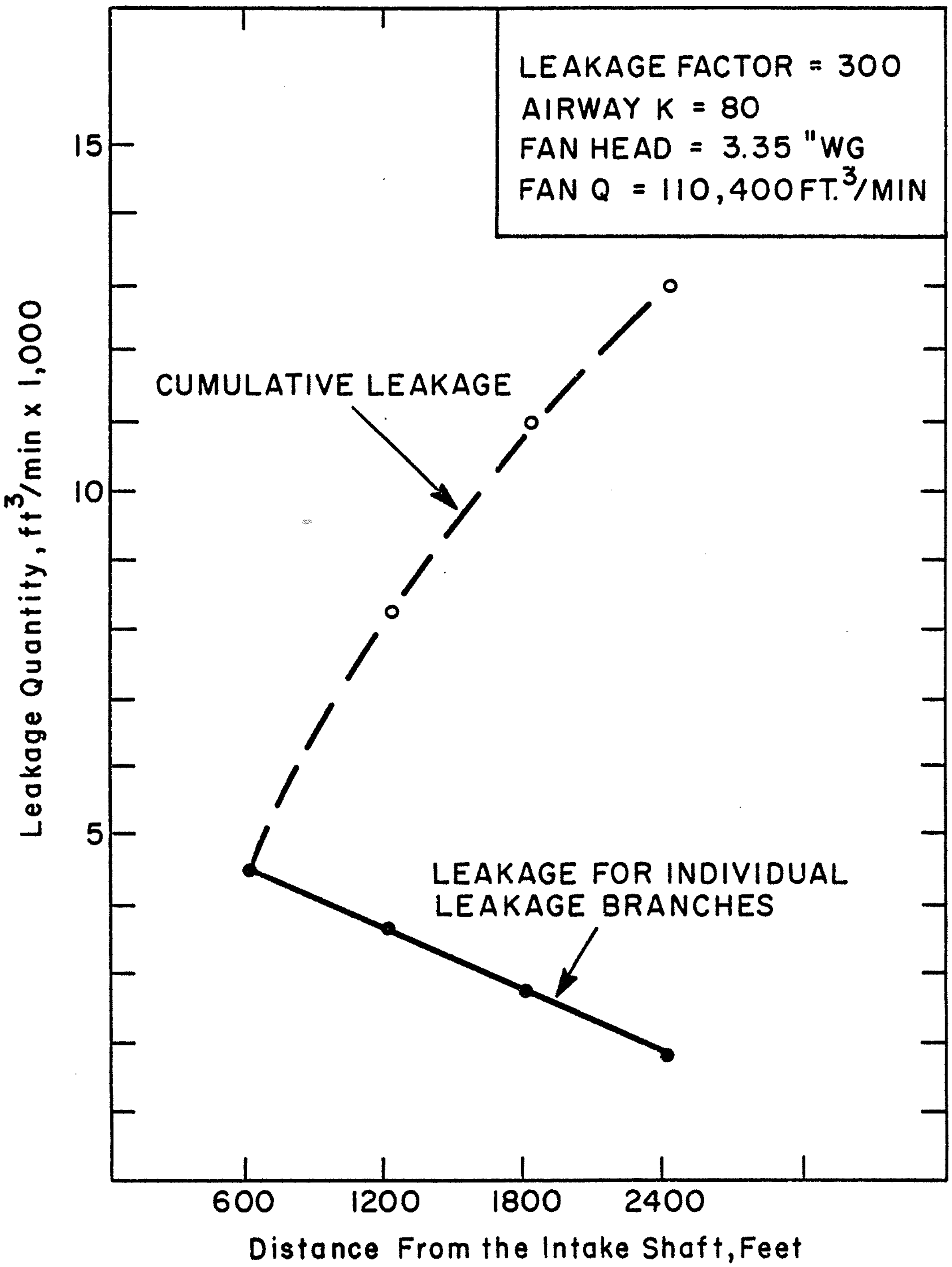

FIGURE 4.2

Leakage Quantities for Four Consecutive Leakage Branches in a Mine (1 Leakage Branch = 7 Stoppings)

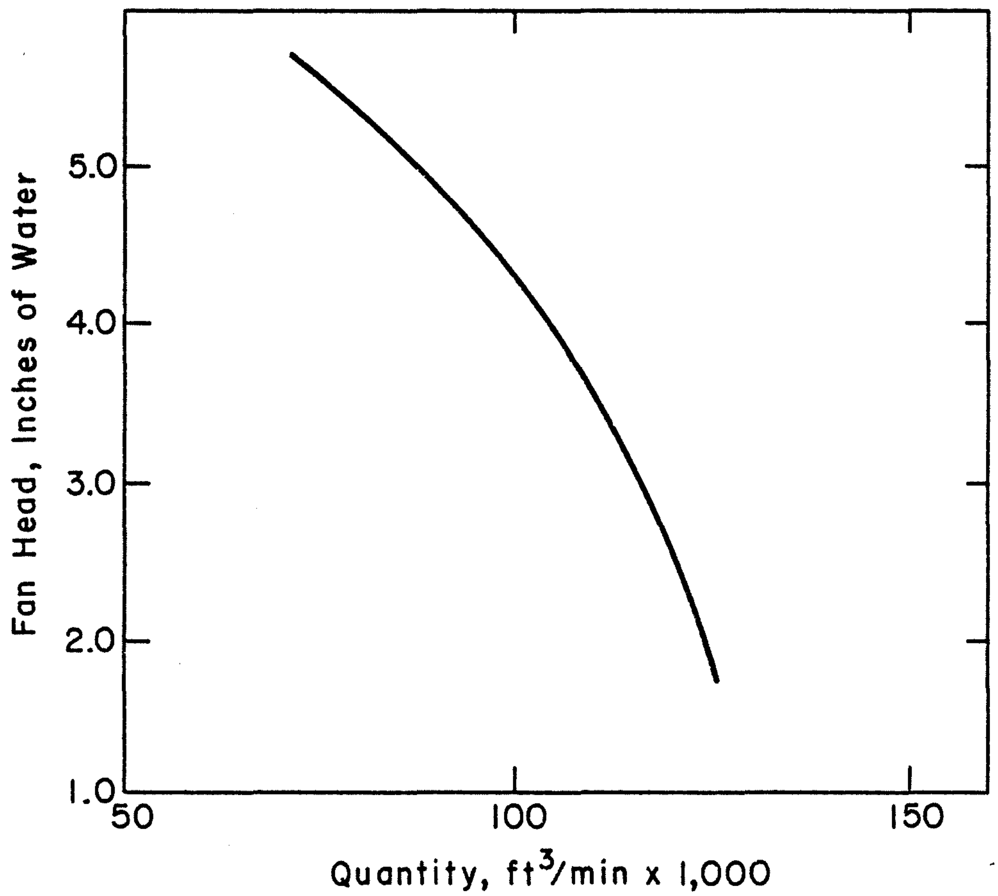

FIGURE 4.3

Characteristic Curve for the Fan Used in the Hypothetical Case

120, in increments of 20. The range of values for leakage factor considered was from 20 to 900 ft^3/min per 100 ft^2 area of stopping in increments of 20, 100, 300, 600 and 900 ft^3/min respectively. To isolate the effect of increasing friction factors on fan head and total leakage, all airways were assigned the same friction factor values. For a given friction leakage factor combination, values of total quantity, total leakage, total head, and air horsepower were noted. These are presented in Figure 4.4 and 4.5. As the friction factor increases (while holding stopping leakage factor constant), (a) the total air quantity of the mine and air available at the face decrease, and (b) fan head, fan air horsepower, and total mine leakage increase.

As the stopping leakage factor increases (holding the airway friction factor constant), (a) fan head, fan air horsepower, and air available at the face decrease, and (b) total mine air quantity and total mine leakage increase.

With the life of the mine increasing, both leakage and friction factor increase. For example, consider the case where the friction factor value increases from 60 to 80, and the leakage factor value from 100 to 300 ft^3/min. From Figures 4.4 and 4.5, the following values can be read off:

	Friction Factor 60 Leakage Factor 100	Friction Factor 80 Leakage Factor 300
Fan Head (in. of water)	2.0"	3.4"
Fan Quantity (ft^3/min)	108,000	111,000
Face Quantity (ft^3/min)	105,000	98,000
Leakage (ft^3/min)	3,000	13,000
Air Horsepower	50	59

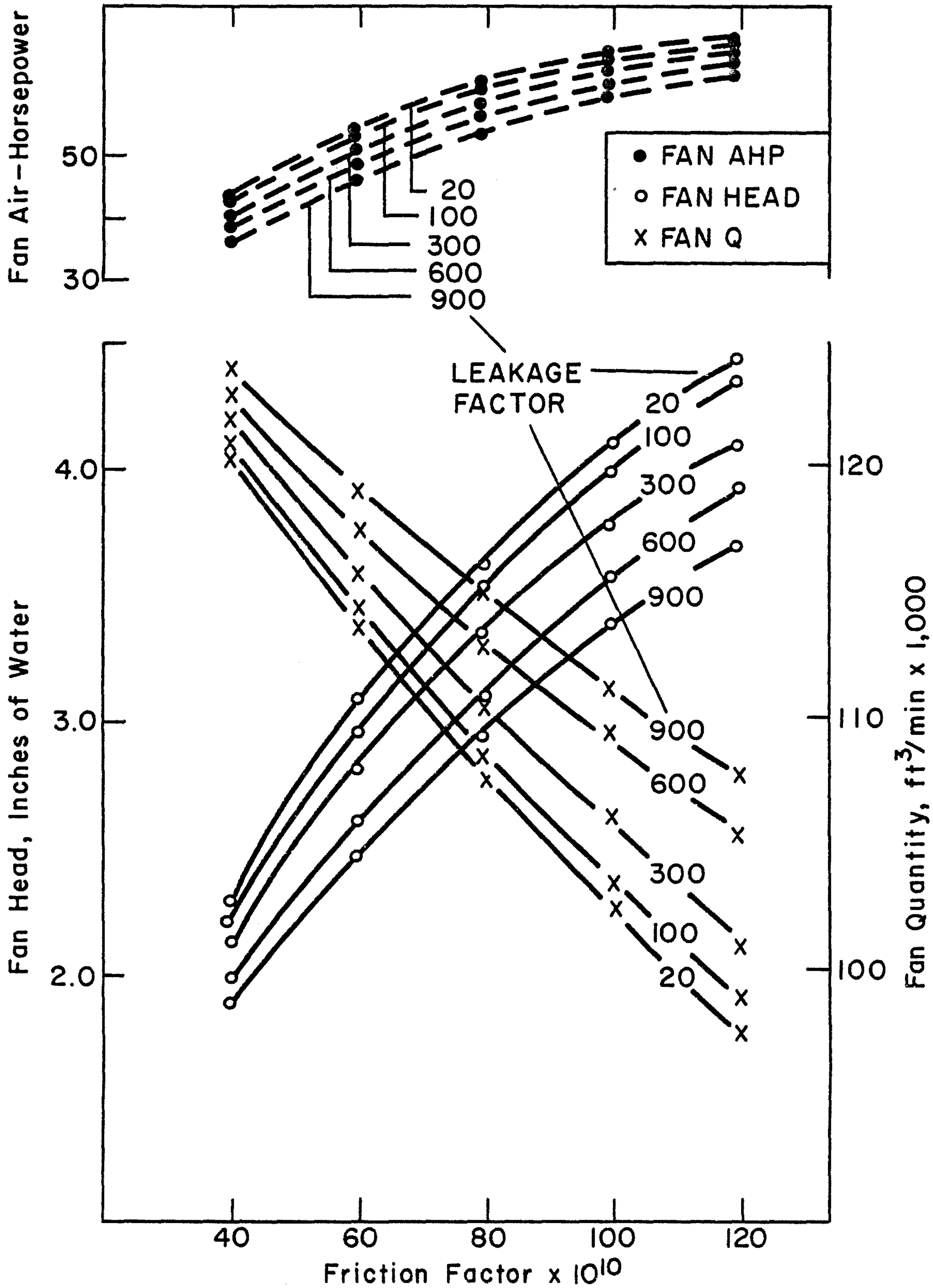

FIGURE 4.4

Effect of Friction and Leakage Factors on Fan Parameters in a Hypothetical Mine

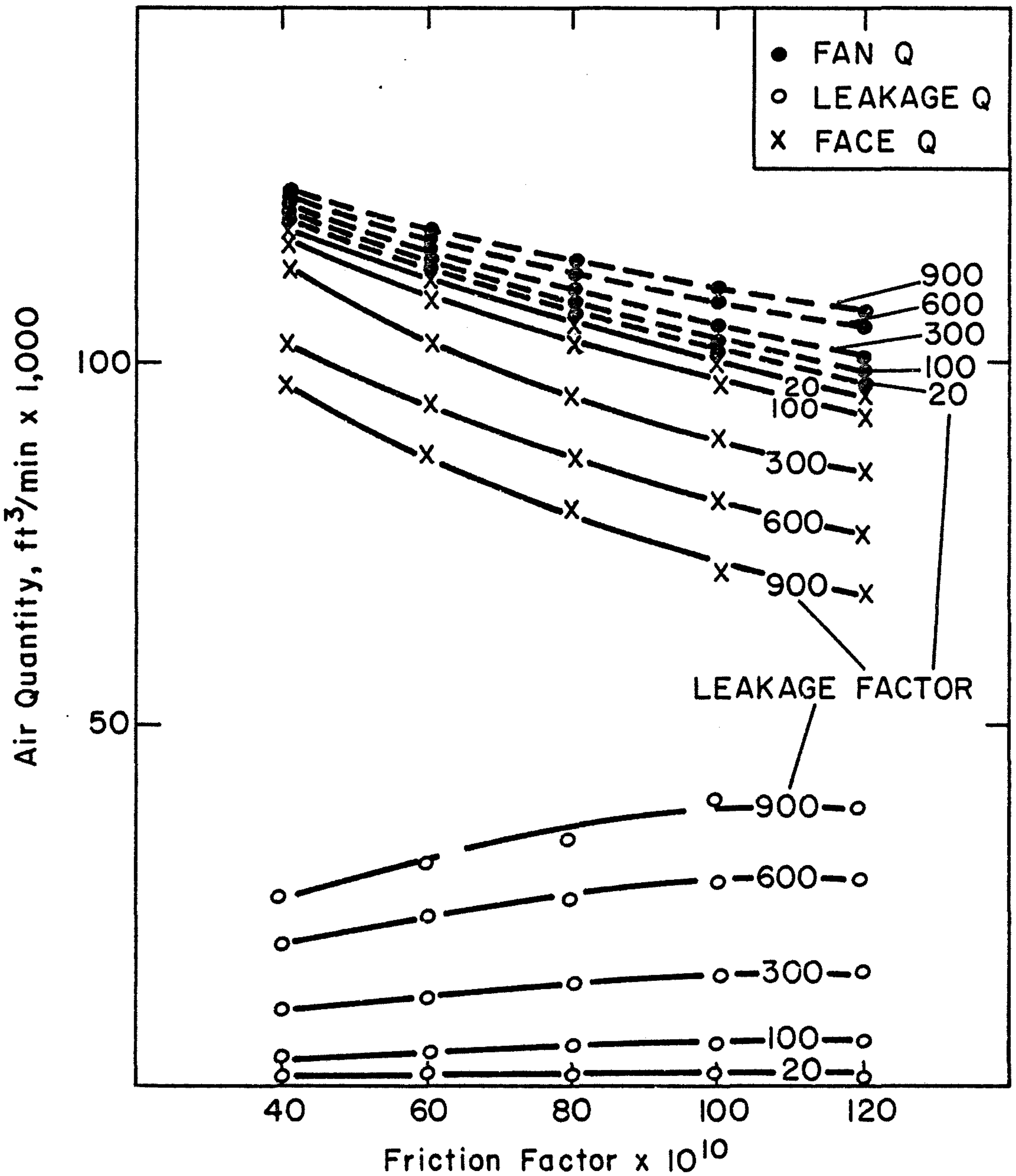

FIGURE 4.5

Effect of Friction and Leakage Factors on Mine Air Quantities in a Hypothetical Mine

The increase in friction factor and leakage result not only in an increased power consumption but also in a decreased quantity delivered to the face where the need for air is the greatest.

4.4 Leakage and Friction Factors in a Large Mine

The results obtained in the previous section were as expected from the general ventilation laws and could have been generally predicted. A mine ventilation network is much more complex than the hypothetical case described in the last section. The response of such a system to changing friction and leakage factors is not easy to hypothesize. To study how a mine would behave under varying leakage and friction factors, an operating mine whose ventilation layout is shown in Figure 4.6 was chosen.

The mine is located in Jefferson County, Illinois. Mining operations are at about 600-foot depth in a generally horizontal 7.5-foot thick coal seam. As shown in Figure 4.6, the main headings are in the East-West direction, the production entries being North and South, orthogonal to the twelve main entries.

Mining operations are of room and pillar type, openings being 7.5 feet high, 15 feet wide, with pillars at 80-foot centers. In a typical mining operation, a three-entry development panel is driven to about 400 feet off the main entry. An additional entry is then started to allow two (2) entries for intake, one (1) belt entry, and one (1) return entry. This development is driven a total of 3,000 feet, up to the boundary. From the boundary, a 500-foot panel is mined out while retreating to the main entries. For every six or seven mined-out panels, a 200-foot wide block of solid coal is left as abutment pillar. Figure 4.7 shows typical mining operations.

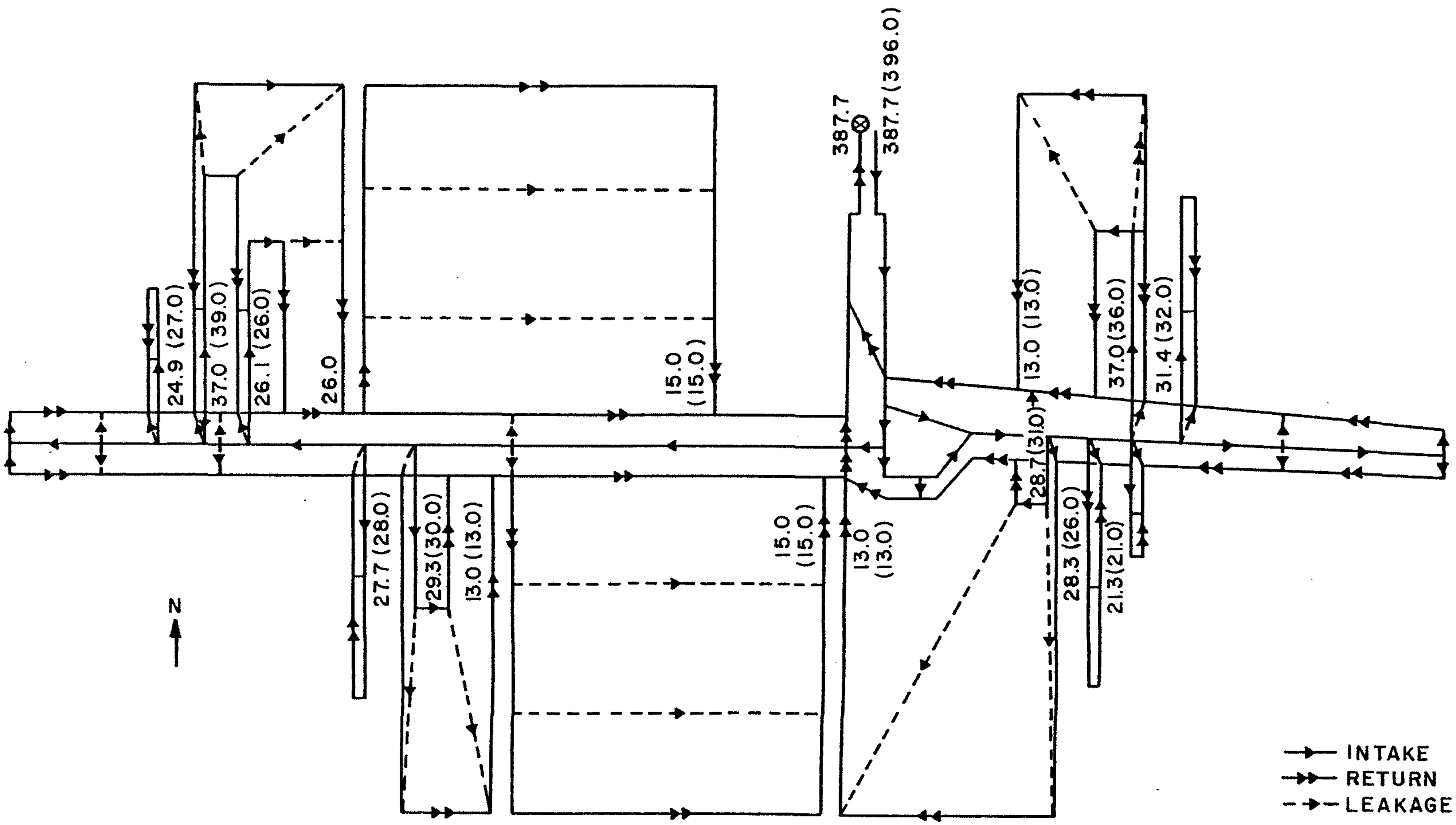

Figure 4.6. Mine Ventilation Layout Showing Some of the Computer Outputs for Air Quantities and Actual Air Quantities in Parenthesis

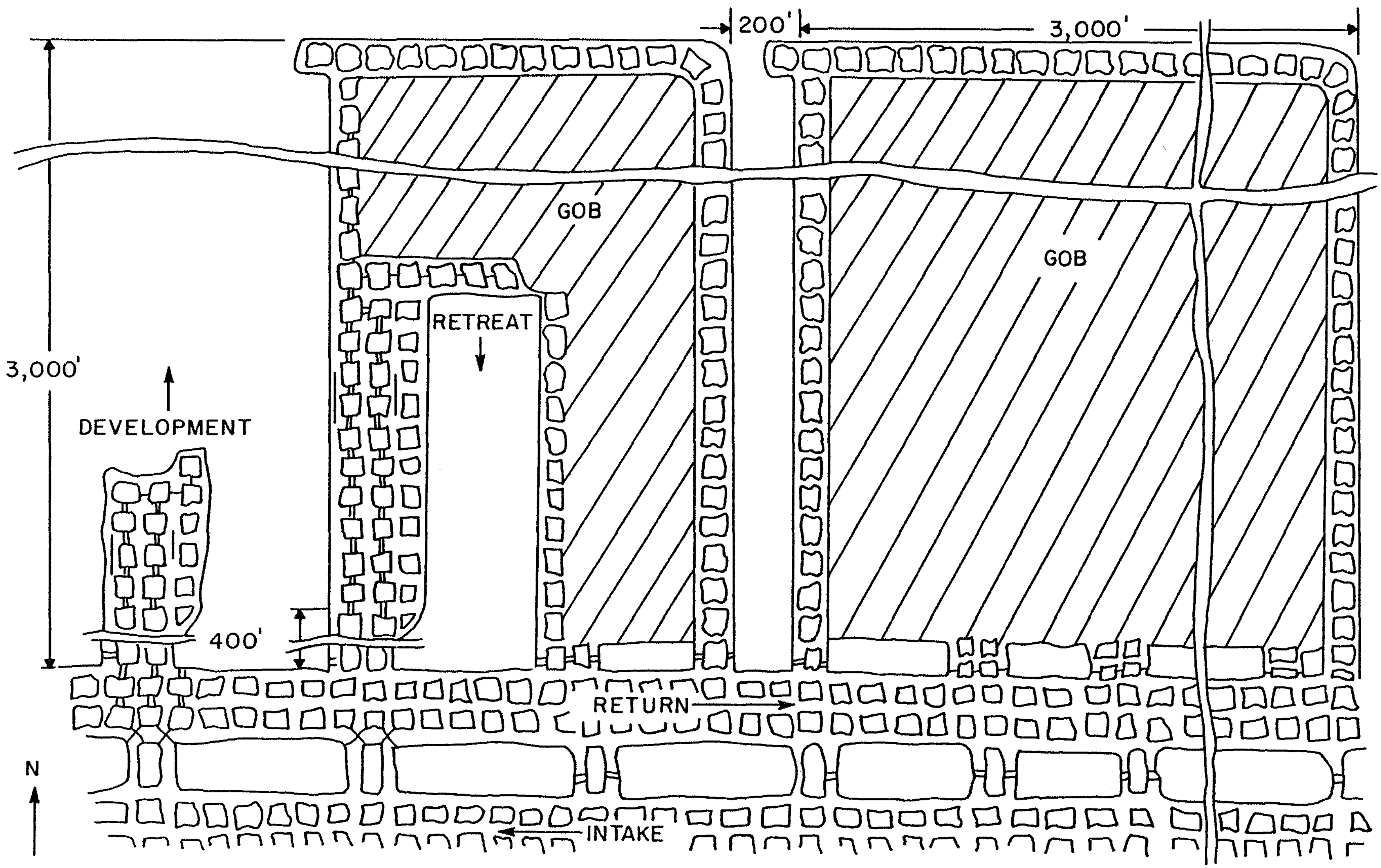

FIGURE 4.7. Typical Mining Operation

The mine employs two main fans for exhaust service in parallel operation. The fans are Jeffrey Aerodyne Fans, type 8HU-96, both operating at the blade setting 1B-1S. The characteristic curve for one fan for this blade setting is shown in Figure 4.8. The mine intake quantity is about 400,000 ft^3/min, and the fans operate at an average water gage of 5.8 inches, as shown by the dotted lines for the intersection of the mine characteristic and the combined parallel fan characteristic in Figure 4.8.

Twelve main entries in the East-West direction are standard, six central entries being the main intakes and three on either side, the main returns. Also, a six-entry intake and six-entry return airways will be employed from the main shaft to the main entries.

The two-compartment shaft is 600-feet deep, having an area of 347 square feet for intake and 146 square feet for return air. The side ratios for both intake and return shafts are 1.5 to 1, giving a shape factor of 4.08.

Two intakes and one return are standard for both development and retreat sections, retreat sections having an additional entry from the previously mined section for return. The gobs are ventilated with return air, using two entries around the perimeter of gobs. All entries are of the same size, 7.5-ft high, and 11-ft wide with a shape factor of 4.24, area of 113 ft^2 and perimeter of 45 feet.

The air quantities were fixed at 20,000 ft^3/min for West main headings, 18,000 ft^3/min for development headings, 15,000 ft^3/min for East main headings and retreat panels and 15,000 and 13,000 ft^3/min for the gob returns, to correspond with the actual mine quantities.

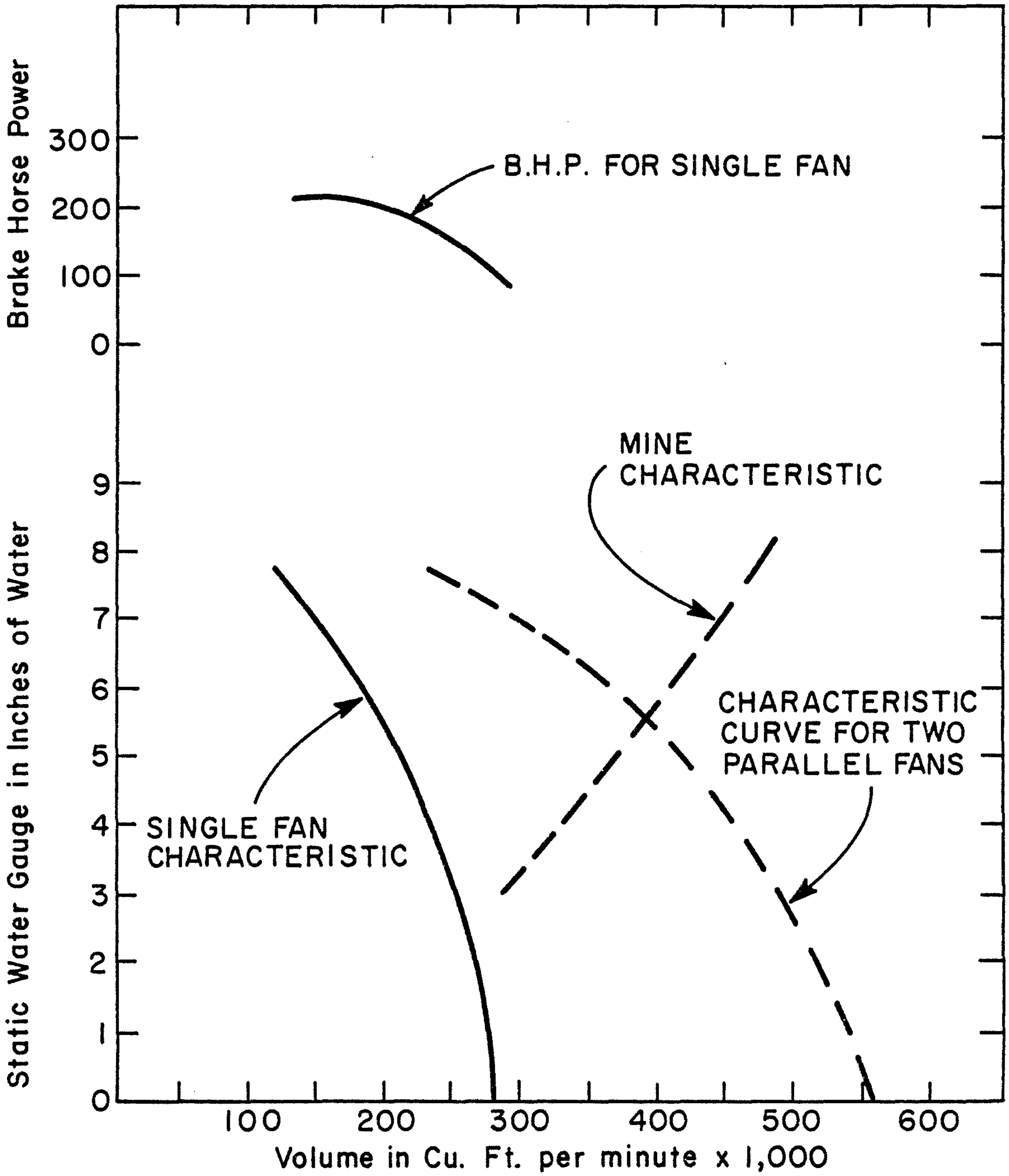

FIGURE 4.8. Constant Speed Curves for Jeffrey 8HU-96 Aerodyne Fan (880 RPM Blade Setting 1B-1S, Air Density = 0.075 lb/ft^3)

The mine intake airway friction factors were ranged from 40 to 100, in increments of 20, and return airway friction factors were ranged from 60 to 120, respectively. The stopping leakage factors had the values of 20, 100, 300, 600 and 900 ft^3/min per 100 ft^2 of stopping per inch water gage, while the quantity exponent was kept constant at 1.43 for all cases. Two different analyses were conducted:

1. The airflow was allowed to distribute itself obeying the natural splitting laws.

2. The face quantities were fixed at values indicated above (total face quantities = 235,000 ft^3/min).

4.4.1. Leakage and Friction Factor Analysis Under Free Splitting: The computer-simulated results show that the fan head, fan air quantity, fan air horsepower (Figures 4.9 and 4.10) are affected only by the friction factors, while total mine leakage and air available at the faces (Figure 4.11 and 4.12) are dependent on both leakage and friction factors. The observations are 1) As the friction factor increases, fan head, fan air horsepower and mine leakage increase, fan quantity and air available at the faces decrease; 2) As the leakage factor increases, air available at the faces decreases and total mine leakage increases. The leakage factor did not have any effect on mine resistance, fan head, fan air horsepower or fan air quantity. In general, the results are similar to those found for the hypothetical case, the only difference being that the effect of friction factors on mine parameters is greater than that of leakage factors.

4.4.2. Leakage and Friction Factor Analysis with Fixed Face

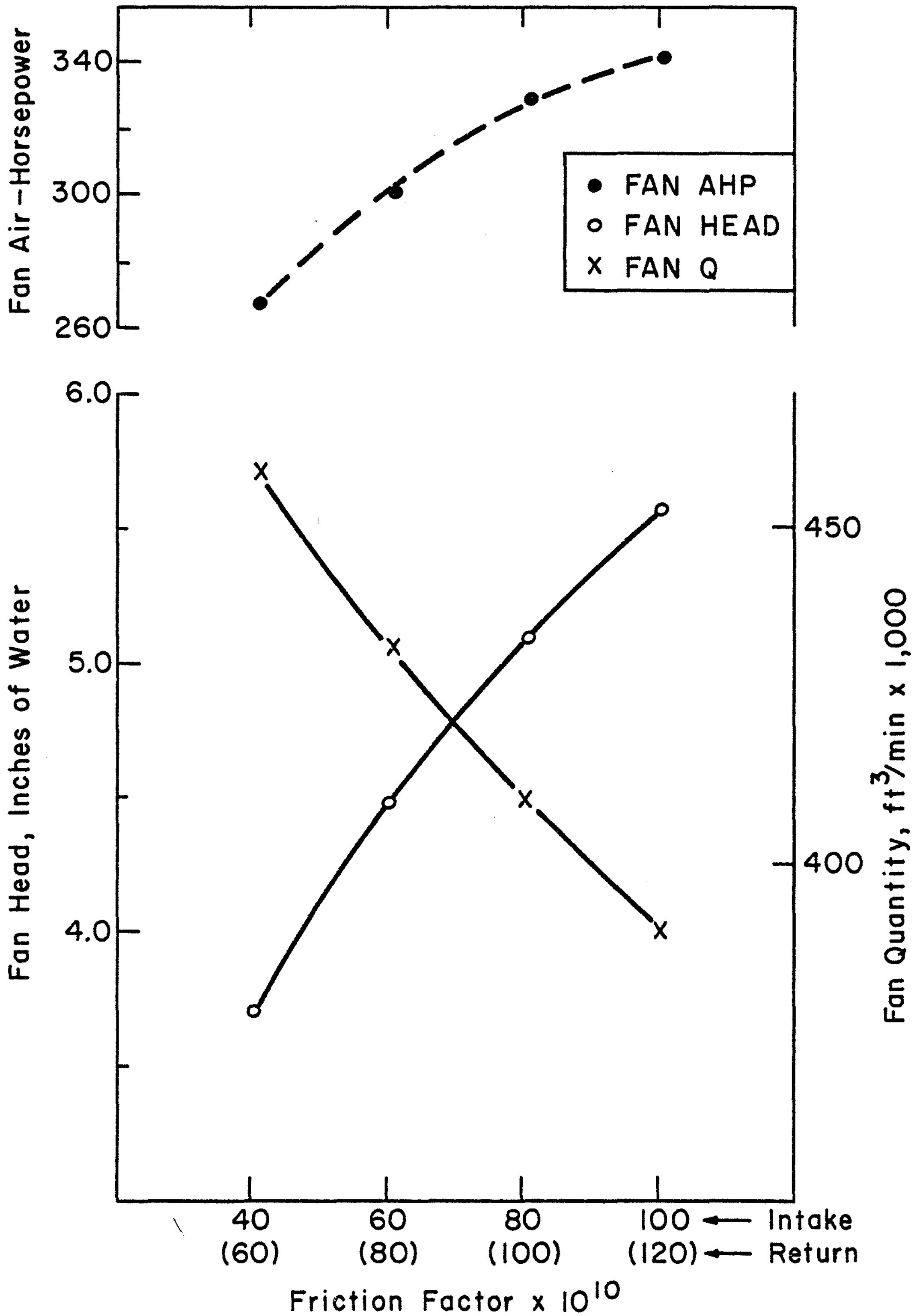

FIGURE 4.9. Effect of Leakage and Friction Factors on Fan Parameters (Natural Splitting). (Leakage factor variations had no marked changes in Fan Quantity and Fan Head as shown in Figure 4.10)

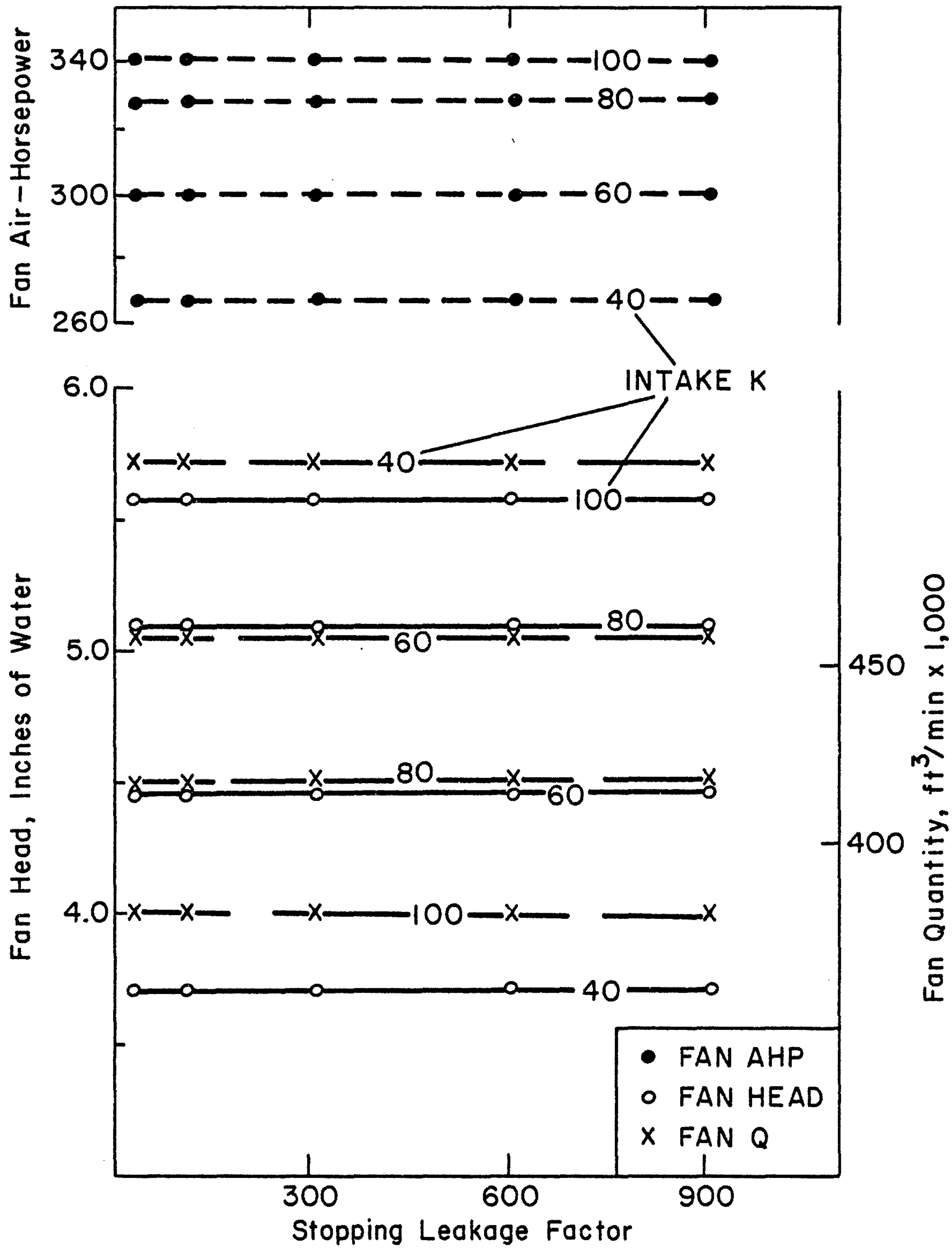

FIGURE 4.10. Effect of Leakage and Friction Factors on Fan Parameters (Natural Splitting)

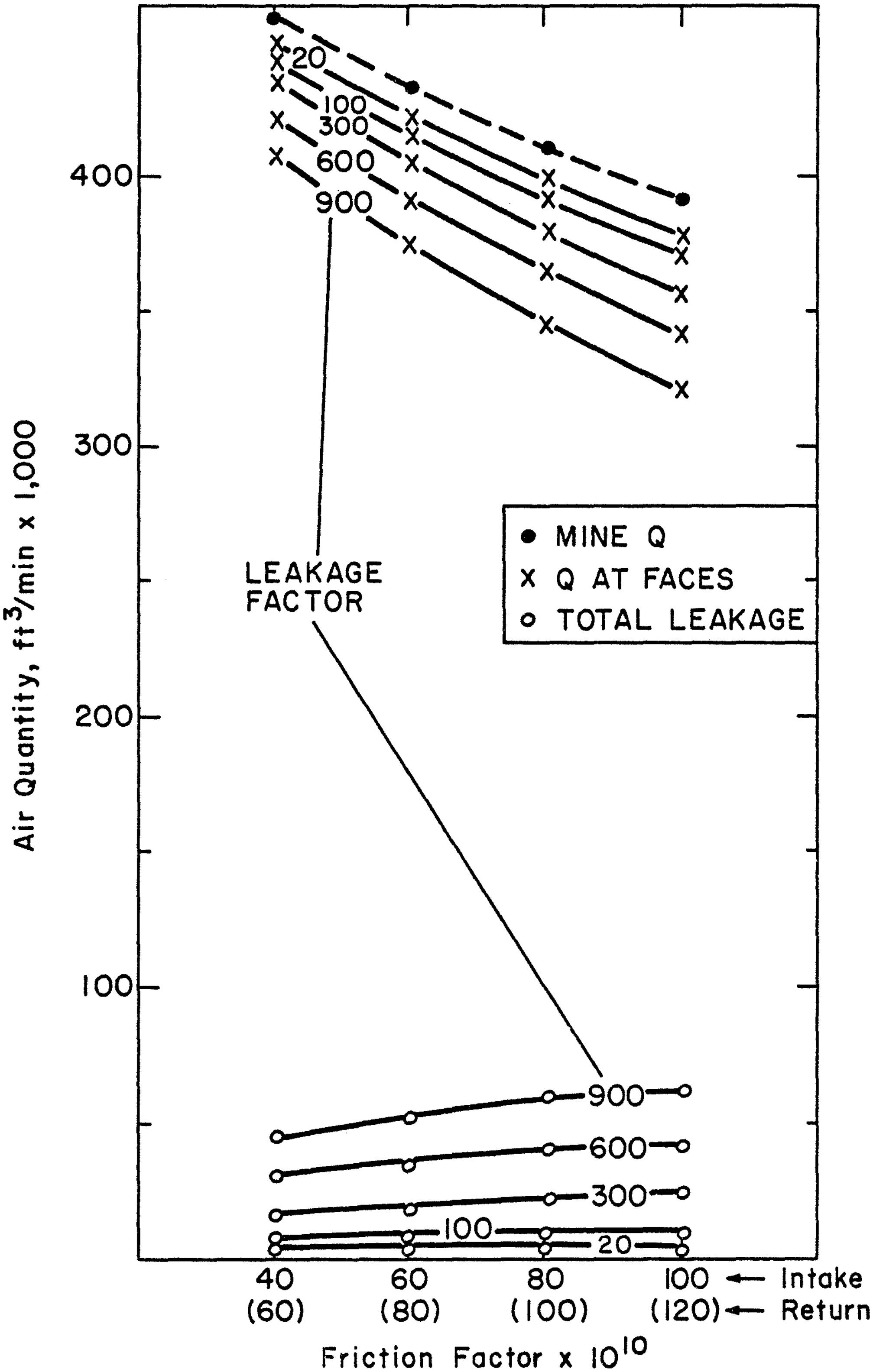

FIGURE 4.11. Effect of Leakage and Friction Factors on Mine Air Quantities (Natural Splitting)

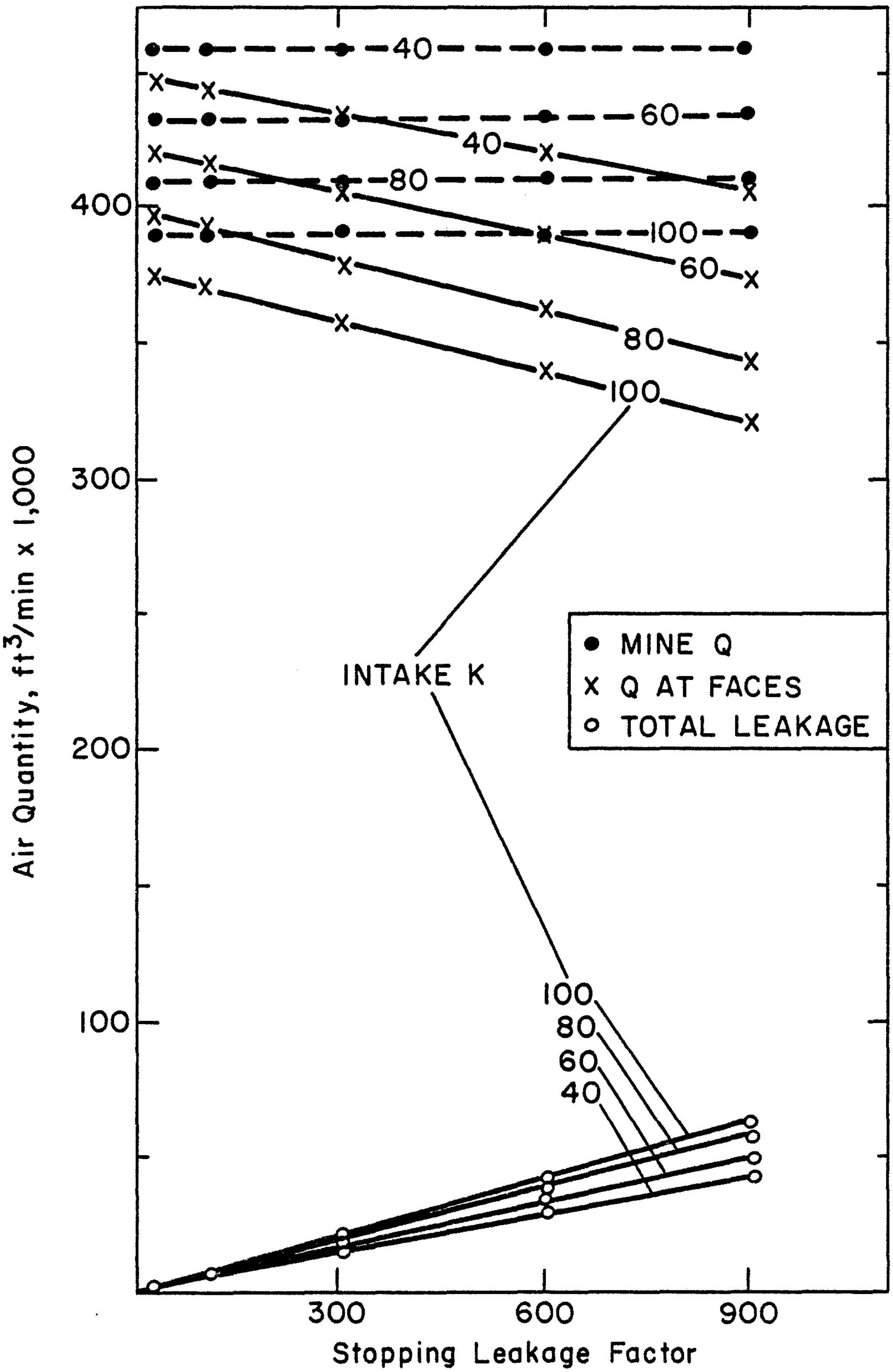

FIGURE 4.12. Effect of Leakage and Friction Factors on Mine Air Quantities (Natural Splitting)

Quantities: It is well known that the quantity of air at a face cannot be allowed to decrease below a certain desired level. In fact, the criterion for mine ventilation planning is to determine the minimum quantity of air required at the faces and then plan an outby system for this quantity. Therefore, the air quantities at the faces are fixed, and the effect of leakage and friction increases is analyzed.

When the face quantities are fixed, studies show that: 1) as the friction factor increases (keeping leakage factor constant), fan head, and fan air horsepower (Figure 4.13) increase, while fan quantity and total mine leakage (Figure 4.14) decrease. 2) as the leakage factor increases, (keeping friction factor constant) fan head, fan air horsepower and air horsepower decrease, while fan quantity and total mine leakage increase.

The fixed face quantities in this mine were lower than those that would be flowing under the natural splitting. Therefore, in this mine, regulators were needed at the faces.

4.5 Summary

Leakage and friction factors affect the head requirement of the fan, and the amount of air delivered at the last open crosscuts. Since both of these factors change over the life of a mine, it is necessary to study the effects of these on the ventilation system. The computer simulation models of ventilation systems are ideal for such analyses as has been determined in this chapter.

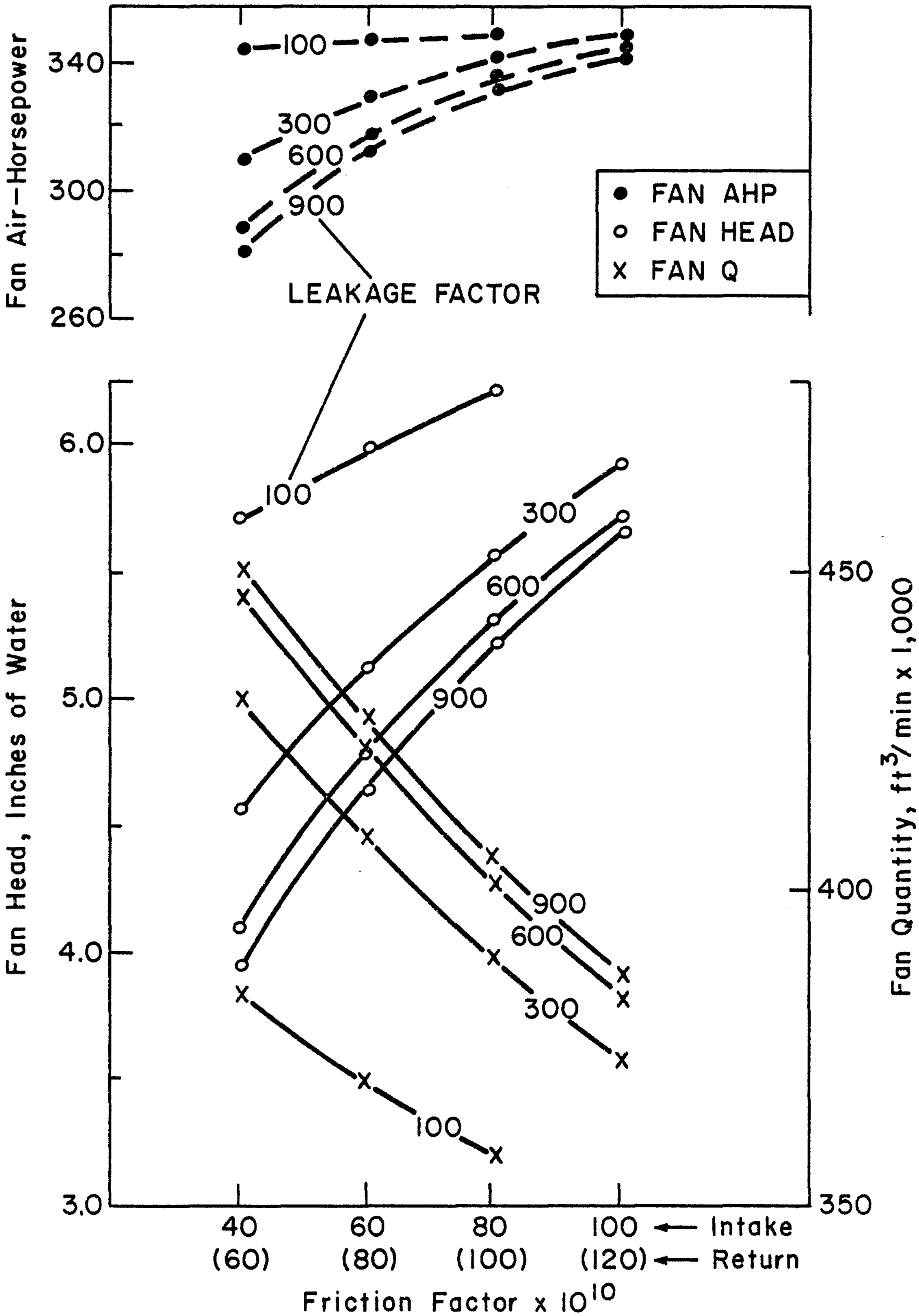

FIGURE 4.13. Effect of Leakage and Friction Factors on Fan Parameters (Fixed Face Q)

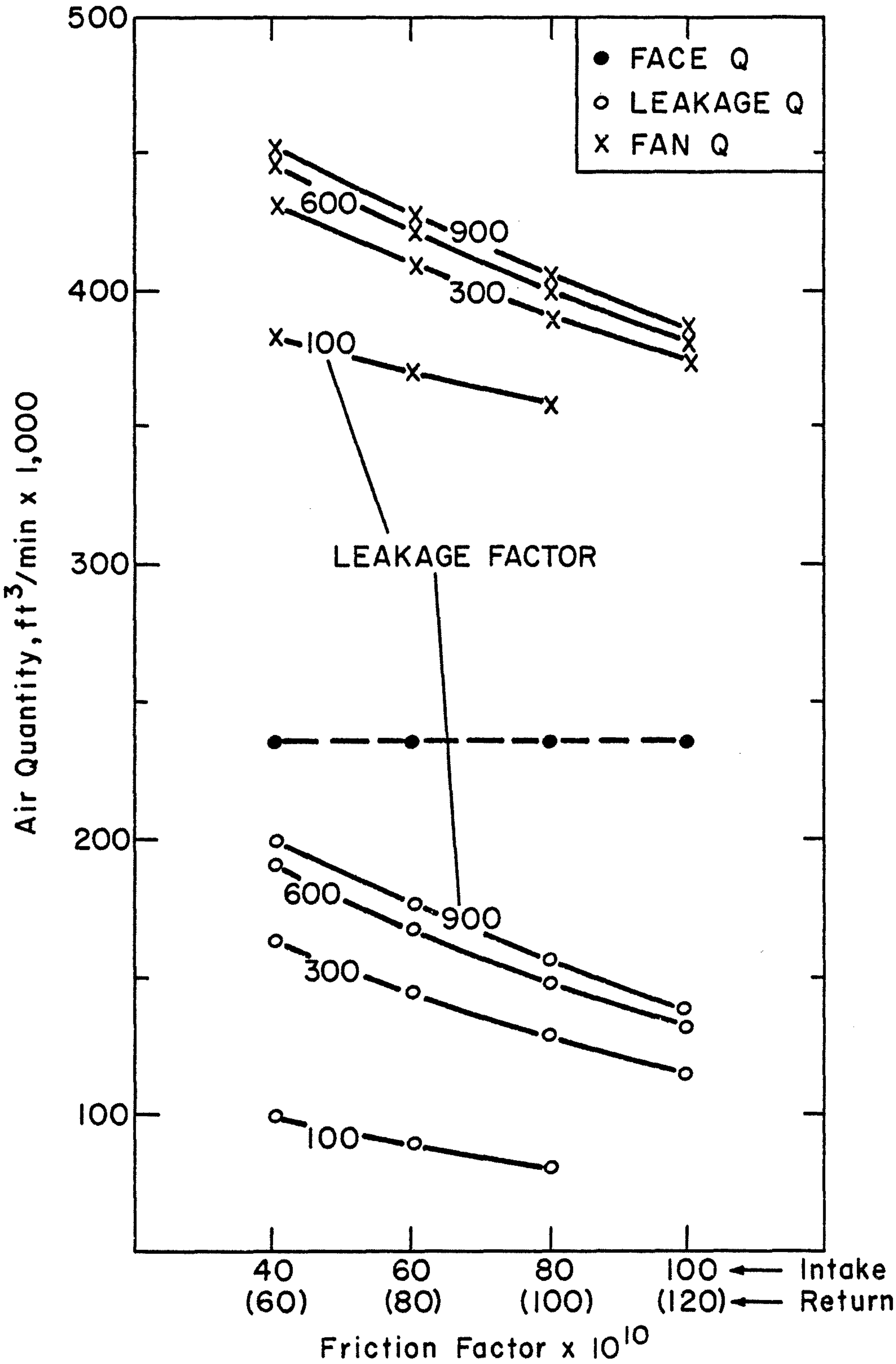

FIGURE 4.14. Effect of Friction and Leakage Factors on Air Quantities (Fixed Face Q)

V. ECONOMICS OF MINE VENTILATION

The previous sections on friction factor and leakage analysis would not be complete without some discussion of the economic considerations in ventilation. In mine ventilation, as in any other branch of engineering, economics is mainly a choice between alternatives. The choice between a fan having a high efficiency and high capital cost and another fan having a lower efficiency in addition to a lower capital cost should be made based on the lowest total costs over the mine life. Often, however, economic criteria are pre-empted or replaced by other more practical considerations such as roof control or equipment dimensions. An excellent but extreme example of this is the ventilation system at the Kaiser Resources, Ltd., coal mine in Canada. The intake air enters the mine through a blowing fan and is heated. The air is forced to exhaust the mine through the caved stopes to the surface rather than through a return aircourse system. This system of ventilation cools the gob area and prevents spontaneous combustion (Parks, 1976). When ventilation is the primary purpose for an aircourse or shaft, engineering considerations dictate that airway design must minimize the ventilation cost.

5.1 Cost Comparisons

Any airway design for a given air quantity must minimize the total ventilation cost. This cost can be broken down into two parts: the capital (fixed) costs and the operating (variable) costs. The capital costs are accrued in borrowing money and repaying the original investment (Hartman, 1961), and so include the interest and amortization charges on the capital invested in the airway. The operating costs in ventilation are mainly power costs. To make a comparative assessment between the capital and operating costs a common basis is needed. This basis can be the net present value, the annual cost, or the total cost over a specified period of years (Lambrechts, 1974). No matter which basis is chosen for comparison, given the same conditions, the same minimum will be found. For mine ventilation, annual costs are usually used.

5.2 Quantity Requirements

The power required to move air through the ventilation system or circuit is directly proportional to the product of quantity and head loss in the system [equation (4.1)]:

$$AHP = \frac{H \times Q}{6350}$$

Since $H = RQ^2$, where R is the resistance of the mine, the power requirements are proportional to the quantity cubed:

$$AHP \propto Q^3 \qquad (5.1)$$

It is obvious that, from power considerations alone, only as much air as required should be circulated through the system. To minimize power costs basic air requirements should be set prior to the actual system design. These requirements should be based on the quantities required for the removal of airborne dust and harmful gases (both noxious such as nitrous oxides and explosive such as methane), and to provide suitable climatic conditions (moisture content and effective temperature). The air quantity required for the removal of dust and gases is based only in a minor way on economic criteria due to the dangers involved in too little airflow for effective dilution and dispersion (West, 1959). Suitable climatic conditions involve an economic trade-off between working efficiency and the cost of air conditioning. Tabor (1969), after reviewing the ventilation standards for 76 longwall faces in Great Britain and comparing the standards with the methane ignitions occurring at the face, suggested the development of air quantity standards based on risk analysis. On the premise that too high a quantity is wasteful financially, and too small a quantity may precipitate hazards he felt standards based on risk analysis would "virtually guarantee that business objectives will not be restricted and ...(such standards) are almost certain to prevent injury or loss of life from environmental causes." Standards initially developed in this manner would be a function (through time) of the ability to measure and monitor the levels of the dust, gases, and environmental factors. Risks develop because measurements made at statutory intervals may not reveal hazardous situations in the ventilation of both active and waste.

The statutory limits or threshold values for dust, gas, and climatic conditions are determined by a compromise between various groups such as governments, industry, and labor unions and include safety factors to minimize the risks. Automatic monitoring of mine ventilation, despite the high cost, can be justified by the reduction of the statutory or threshold levels due to an elimination of the safety factors (Greuer, 1973).

This same logic can be carried forward to justify automatic controls of airflow. Automatic control can smooth out variations in both the air quality and quantity. Work stoppages occurring due to the peak values of methane, dust noxious fumes, temperature, and humidity can be eliminated, increasing productivity. In addition, the safety, comfort, and morale of the workers would increase.

5.3 Circuit Design

Once air quantity requirements have been set the ventilation system or circuit design can be developed.

The design of the mine ventilation system can be broken down into three phases: the downcast circuit, the mine workings, and the upcast circuit. The mine workings are a function of the mining method, ore body geology, etc., so that each system is different. Airway design from economics criteria is difficult in this area due to practical considerations already mentioned, except for airways whose sole function is ventilation. Economic considerations can be utilized to their full extent in the upcast and downcast circuits, limited only by other services such as hoisting and escapeways (West, 1959).

5.3.1 General Considerations. The headloss for an airway or simple ventilation circuit is given by Atkinson's equation, which was discussed in section 2.1 [equations 2.1 and 2.2]:

$$H_f = \frac{KLPQ^2}{5.2A^3} \quad \text{or} \quad H_f = RQ^2 \quad \text{where} \quad R = \frac{KLP}{5.2A^3}$$

The air horsepower is related to the headloss in a circuit by [equation 4.1]:

$$AHP = \frac{HQ}{6350} = \frac{KLPQ^3}{33000A^3} = \frac{H_f^{3/2}}{R^{1/2}} \qquad (5.2)$$

From equation (5.2), it can be deduced that

$$AHP \propto Q^3$$

$$AHP \propto H_f^{3/2} \qquad (5.3)$$

Assuming that the quantity requirements are fixed prior to the system, design power requirements can only be lowered by minimizing the static (H_f) and velocity heads, (H_v) in the system. Although the system headloss has less effect on the power requirements than the quantity, substantial savings can still be achieved by headloss reduction. Often when quantity requirements increase in the later stages of a mine some work must be done on the system headloss to help offset the tremendous increase in power costs. With the quantity fixed, from equation (2.1), it can be seen that

$$H \, \alpha \, R \text{ where } R = \frac{KLP}{5.2A^3} \qquad (5.4)$$

implying that either K, L, or P must be reduced or that A must increase for reductions in headloss.

5.3.2. Size. The size of an airway is a function of the perimeter (P) and area (A), both of which appear in Atkinsons Equation. This can be expressed as (Hartman, 1961; Mishra, 1962):

Circular Section: diameter, D
$$H_L \, \alpha \, {}^{P}/A_3 \, \alpha \, \frac{\Pi D}{(\Pi/4D^2)^3} \, \alpha \, {}^{1}/D^5 \qquad (5.5)$$

Square Section: Side, D
$$H_L \, \alpha \, \frac{4D}{(D^2)^3} \, \alpha \, {}^{1}/D^5 \qquad (5.6)$$

Rectangle Section: Sides A & B
$$H_L \, \alpha \, \frac{2A + 2B}{(A^2B^2)^3} \, \alpha \, {}^{1}/A^5, \; {}^{1}/B^5 \qquad (5.7)$$

If the shape of the entry is held constant (Mishra, 1962):

$$H_L \, \alpha \, {}^{1}/A^{5/2} \qquad (5.8)$$

because the perimeter is proportional to the square root of the area ($p \, \alpha \sqrt{A}$). From this it can be seen that to reduce headloss either larger entries or more openings (in parallel) are required.

5.3.3. Surface Characteristics. The surface characteristics of an airway are expressed by the friction factor, K. Since $H_L \alpha K$ the headloss can be lowered by reducing the friction factor of the entry. This can be done by entry cleanup (reduction in roughness, obstructions, and sinuosity) or by lining. It is common practice to line shafts, slopes, and some main intake entries.

5.3.4. Shape. Shape can be expressed as the hydraulic radius (Area/Perimeter):

$$R_H = A/P \qquad (5.9)$$

From Atkinson's equation [equation (2.1)] it can be seen that:

$$H_L \alpha 1/R_H \qquad (5.10)$$

A circle has the least perimeter for a given area (hence the smallest hydraulic radius). Other shapes of entries are compared to a circle in Table 5.1. The variation of power cost and headloss due to shape is negligible, as long as a rectangle with a side ratio of below 1 to 1.5 is used.

5.3.5. Length. In a system or circuit, $H_L \alpha L$. In most systems the length is fixed; however, the most direct route of airflow should always be utilized.

5.3.6. Shock Loss. Shock loss can be expressed as equivalent length, Le. Head loss is then proportional to the equivalent length ($H_L \alpha Le$). Approximate equivalent lengths for various sources of shock loss are given in Table 2.2.

Shock losses account for only 10 to 30 percent of the total head loss in a mine ventilation system. Shock losses can be minimized by avoiding sharp bends (by using a large radius or angle of deflection)

TABLE 5.1

Shape Factors and Relative Head Loss

Description of the Shape	Hydraulic Radius/ft	Relative Head Loss
Circle	3.55	1.00
Regular hexagon	3.66	1.03
Arched roadway with a side ratio of 1:1 and a radius of curvature of 5 times the side for the arch	3.86	1.09
Square	4.00	1.13
Rectangle with a side ratio of 1:1.25	4.03	1.14
Rectangle with a side ratio of 1:1.5	4.08	1.15
Rectangle with a side ratio of 1:2	4.26	1.20
Rectangle with a side ratio of 1:3	4.62	1.30

and by streamlining area changes or obstructions (such as overcast approaches).

5.3.7. Shaft Design. For a shaft of under 2000 ft in depth, assuming a 10-year life, the following cost breakdown has been developed (West, 1959):

	Low power costs	High power costs
Shaft capital cost	64%	65%
Fan capital cost	14%	13%
Power costs	13%	17%
Fan maintenance costs	9%	5%

Mining cost and power costs are the two biggest items in the cost breakdown. Fan capital costs and fan maintenance costs are usually neglected in the development of an optimum size shaft since these costs are incurred no matter which shaft is selected. If an airshaft larger than optimum is selected, the increased capital charges exceed the reduction in power costs. An airshaft smaller than optimum wastes more power than can be justified by the reduction in capital costs (Mancha, 1946). At the optimum shaft size the sum of annual capital costs (amortization including depreciation and interest on the capital investment) and the annual power costs is at a minimum.

If C is the cost of power, ¢/KWH, n is the efficiency of the fan and the drive (percent), and 8760 hours of operation per year are assumed, the annual air shaft power cost (M) in dollars can be calculated (1 HP = 0.746 Kw) (Mancha, 1946):

$$M = \frac{H_f Q \ (0.746) \ (8760) \ (C)}{6350 \ n} = \frac{1.03 \ H_f QC}{n} \qquad (5.11)$$

Substituting for H_f from equation (5.3), it is obvious that

$$M = \frac{1.03\ KLPQ^3.C}{5.2n\ A^3} \quad (5.12)$$

The shaft sinking costs can be broken into three parts, the move in charge (cost of transporting, erecting and removing the shaft sinking equipment), the cost of shaft lining, and the excavation cost. In this analysis the move in charge is assumed to be fixed and independent of the airshaft size. The cost of shaft linings is also assumed to be fixed at a cost per cubic yard of lining material (Mancha, 1946). The capital costs represented by the shaft sinking costs are treated as expense items and are amortized over a 15-year life by most coal companies. Amortization is a special form of straight line depreciation for capital expense items (Stermole, 1973). For purposes of size comparison interest on the investment will be included with the depreciation expenses.

The shaft sinking cost can then be expressed as (Mancha, 1946):

$$\frac{(A_L - A)\ L_L\ xa}{27} + \frac{(A_L L_L + AL)\ ca}{27} + ya \quad (5.13)$$

Where A_L = area of lined airshaft, ft^2

A = area of airshaft, unlined, ft^2

L_L = lined length, ft

L = unlined length, ft

a = amortization rate (%/100)

c = excavation costs ($/cu yd)

y = move in costs

The total annual cost is then the sum of the airshaft power and amortization costs. The optimum shaft size can be determined graphically using the power and amortization cost equations as shown in Figure 5.1.

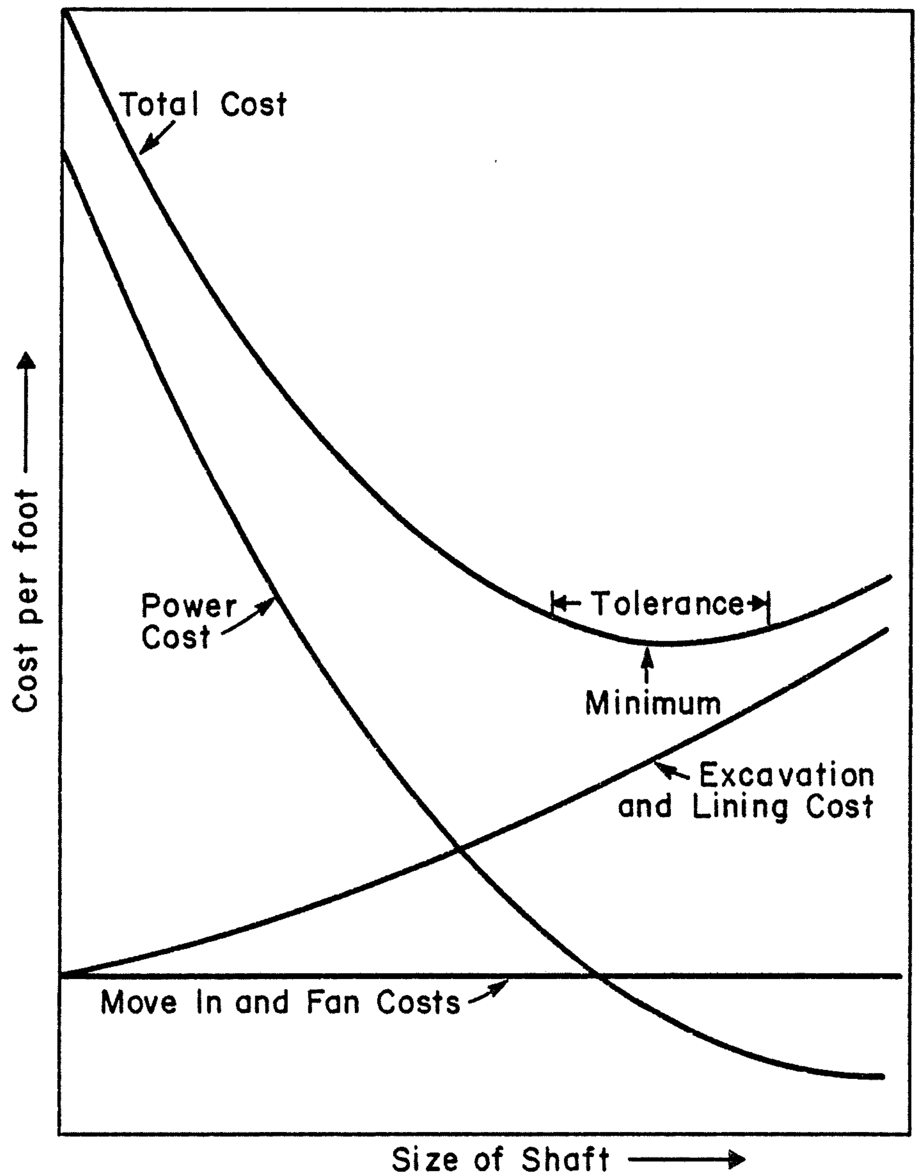

FIGURE 5.1

Optimum Shaft Size Determination

Adapted from (West, 1969)

The curve representing the total annual cost flattens out slightly at the minimum point. This range about the minimum point gives a tolerance to the results and enables other considerations such as hoist size to be taken into account (West, 1959).

By equating the first derivative of the total annual cost equation with respect to D (inside principle dimension) to zero, the optimum shaft size can be found using an equation instead of a graph. Formulas developed by Mancha (1946) for the optimum size for circular, elliptical, and rectangular airshafts are given in Appendix I. For these formulas the thickness of the lining has been assumed as one tenth of the optimum finished shaft size, but could easily be modified to any desired thickness. The formulas also take into account the length of shaft lining, in that some shafts might be lined only in the parts of the shaft passing through incompetent strata. If the shaft is totally lined or unlined the length drops out of the formula and the optimum shaft size is then independent of shaft length.

From a comparative analysis of optimum circular, elliptical and rectangular shafts at a given quantity, using these formulas, the following generalizations were made (Mancha, 1946):

Size - If unlined airshafts are considered, the circular airshaft shape has the least area. If lined airshafts are considered the elliptical and rectangular airshaft might have the least area due to the greater ratio of perimeter to area.

Cost - For a specific depth of lining circular shafts are most economical, followed by elliptical and then rectangular shafts. Regardless of shape, unlined airshafts are cheaper than lined airshafts (in spite of the greater area required to compensate for the rougher surface).

The optimum shaft size equations take all the factors (such as excavation cost, amortization rate, efficiency of fan and drive, etc.) to the seventh root so that extreme accuracy is not needed in developing these parameters. The quantity is taken to the 3/7 root so it should be accurately determined. Figure 5.1 shows, however, that the additional cost resulting from an over-estimate in shaft size (based on an excessive quantity) is far less than that resulting from an under-estimate of the same degree. For an under-estimate of ten percent on quantity requirements the additional costs would be three times that resulting from a ten percent over-estimate. It should be noted from this discussion that shaft size selection by permissible air speeds is completely arbitrary and fails to take into account the type and amount of lining and the shape of the airshaft.

5.3.8. Airway Design. The foregoing analysis of optimum shaft size selection can also be applied to mine airway design. The concept of amortization of mine airways is not completely correct; however, it provides a practical method of comparison between power and mining costs and, as a result, is usually used in airway size determinations.

Mishra (1962) has modified the total cost formula to express the economic size of the airway as a function of quantity and airway life (assumed as 15 years for shafts). Although he used a sinking fund for amortization, his comparisons between coal measure strata, igneous rock, concrete lined airways, and timbered airways yield some general conclusions:

- an increase in the K factor causes an increase in the airway size, but the effect is not appreciable,
- for small quantities (under about 50,000 cfm) the life of the airway past 5 years has little effect on the size,
- the cost of excavation does not affect the area greatly for small quantities.

Once the economic size of an airway is determined the economic velocity can be found. Economic velocities can be useful if plotted for various size entries in that they give a rapid estimation of the airway size for a given quantity.

A problem with airway design develops due to the fixed length and (often) size (due to primary considerations other than ventilation) of the airways, the resistance of the airways leads to a natural split of the air entering the mine. Regulators or booster fans (passive and active control elements respectively) must be used to develop the air quantity required in each split. Regulators are practical and easily removed and adjusted, but the convenience is outweighed by the increase in mine resistance and resulting increase in power costs. Safety aspects of underground booster fans may often, however, preclude their use in hard rock mines due to recirculation patterns that develop and in coal mines due to the explosion and spontaneous combustion hazard . In addition, the temperature rise through a series of booster fans must be considered (Sheldon, 1952).

5.4 Use of Computers in Ventilation Economics

Although a simple mine ventilation system can be optimally designed using the economic parameters previously discussed, it is more difficult to locate the optimum design in complex systems which evolve from several years of operation. In the past decade computers have been used to accurately model the complex systems, predicting the airflow quantities and directions. The traditional empirical and intuitive techniques that have previously been applied to mine ventilation can then be dropped and replaced by simulation techniques that adequately deal with the interaction (Owili-Eger et al, 1973) of the ventilation system with dust, gases, heat and humidity (Didyk, 1974; Ramani, 1974; McPherson, 1976).

The most obvious use of ventilation simulation programs is for short term planning to evaluate changes in the ventilation system. As an example, high resistance segments of a mine can be easily identified by pressure-quantity surveys. The costs for improving these high resistance segments are, in general, high due to an inaccessibility of older mine workings. Ventilation simulations provide a means for assessing the effect of changes in the ventilation system and then comparing the effect of the change (increased air flow, lowered fan head, etc.) in economic terms with the anticipated costs. This method was used to assess the economics of driving new airways and determining airway size when a new ventilation system was introduced at Craigmont Mines in Canada (Press and Johnston, 1976). U. S. Steel has used a ventilation simulator to evaluate effects of the 1969 Coal Mine Health and Safety Act on its ventilation systems. Bethlehem Steel Corporation also utilizes ventilation simulations for short-term planning (Turpin and Weyher, 1976). Their ventilation simulator includes a routine to optimize the fan blade settings and fan speed adjustments for every fan in the mine network, resulting in a lower overall power cost for the same total quantity handled by the fans.

In long-term planning, ventilation simulation programs provide an excellent opportunity to optimize the mine network. White Pine has designed a reliable, effective and economical system based on future requirements in this manner (Tien and Bjork, 1976). Optimum shaft locations and fan sizes can be evaluated on the basis of the effect on mine resistance, the new volumetric efficiency of the system (due to leakage changes), and the airflow capacity for future needs (Gregory, 1970). Some simulators, such as the PSU/MVS, can be used to evaluate leakage as a parameter of stopping construction and analyze horsepower dissipation in the system (Didyk, 1974). Leakage characteristics (Holland and Skewes, 1962; Kawenski and Mitchell, 1963) of the stoppings in a projected segment can be inputed along with the physical parameters of the segment (length, area, perimeter, etc.) and the effects of various stopping types on the overall ventilation system can be noted. Combining this information with cost data will give the most economical type stopping to be used in the mine environment.

Colorado School of Mines is in the process of developing a thermodynamic ventilation simulation program which will be directly applicable to hard rock ventilation systems (with their complex heat and humidity changes). Morse (1976) has utilized an early version of this program to evaluate methods of changing the airflow from an economic standpoint. Using the Henderson Mine ventilation network he evaluated changes in the shock losses, modification of the fan curve (blade pitch), and changes in the thermodynamic state of the air (heating, cooling and evaporation rates) on a discounted cash flow basis. The results showed the greatest value in the reduction of shock losses. The

Henderson Mine at the time of the application was in development stages and had a very simplified ventilation system and high (2.5 w.g.) natural ventilation pressures.

5.5 Summary

The need for optimum and economical ventilation systems will grow over the next few years as costs continue to climb. Proper application of digital computer ventilation simulations combined with a thorough knowledge of the economic parameters of mine ventilation can produce the required optimum and economical system. The current trend toward user oriented programs and inter-active computer systems should help to revolutionise ventilation economics throughout the mining industry.

VI. CONCLUSIONS

6.1 Field Studies

The results of preliminary field investigations by Kharkar et al (1974) demonstrated that for individual mines, leakages and friction factors vary widely. Leakages were found to depend primarily on the pressure difference across a stopping; and that leakage losses are higher in the section of the airway farthest from the working face. For any given mine, the values of both friction factors and leakage losses must be experimentally evaluated periodically since they will normally change with changing ventilation configuration. The effect of changing leakage and/or friction factors cannot be generalized. It is necessary for ventilation engineers to perform sensitivity analysis on leakage and friction factors in their mines to assess the relative importance of each, and to develop guidelines for ventilation improvement.

6.2 Computer Analysis

The PSU/MVS program was employed to study the effect of leakage and friction factors on mine ventilation systems.

In leakage and friction factor analyses for the hypothetical case, it was found that the fan head, fan air horsepower, and mine leakage increased with increasing friction factor and decreased with increasing stopping leakage factor, while the total mine quantity showed exactly opposite relationships. Air quantity available at the face decreased with increasing leakage and friction factors. These results agree with the physical laws of ventilation.

In a more realistic mine layout, under natural splitting, leakage and friction factors affected mainly total mine leakage quantity and air available at the faces while friction factors affected all mine parameters considered. In this particular study, the effect of friction factors was more significant than that of leakage factors. When the face quantities were fixed, the effect of leakage and friction factors cannot be generalized.

6.3 Suggestions for Future Research

This study has only pointed out the importance of good input data for computer programs. The limited field studies on leakage and friction factors does confirm the need for more rigorous field studies. The use of computer programs to analyze recirculation patterns, leakage characteristics, air horsepower dissipations, and emergency escapeways are recommended. Economic analysis of mine ventilation systems is another area where work on the location of shafts, size of airways, number of airways etc. can be conveniently carried out with ventilation simulators. PSU/MVS has incorporated several features (leakage formulas, air horsepower dissipation in circuits, methane injections, methane distribution etc.) which make it a very useful tool for not only ventilation planning but also economic analysis on alternative designs.

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APPENDIX I

Nomenclature for Appendix I

A - finished inside airshaft area, sq. ft.

a - amortization rate, dollars per year per dollar cost of airshaft.

C - cost of electric energy, cents per kw-hr.

c - excavation cost, dollars per cu. yd. excavated.

D - inside diameter of circular airshaft or inside dimension of long side of either elliptical or rectangular airshaft, ft.

E - over-all efficiency of ventilating equipment from electricity to air, percent.

k - ratio of short to long inside dimensions of either elliptical or rectangular airshaft.

l - depth of unlined portion of airshaft, ft.

l_1 - depth of lined portion of airshaft, ft.

M - total annual airshaft cost including both electric energy and amortization, dollars per year.

Q - airshaft design air volume, cu. ft. per min.

R - air-friction coefficient for unlined airshaft.

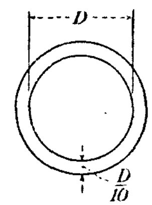
R_1 - air-friction coefficient for lined airshaft.

X - lining cost, dollars per cu. yd. lining material.

Y - move-in cost, dollars.

Appendix from Mancha (1946)

Derivation of Formula for Circular Shaft



$$M = \overbrace{\left[\frac{(R_1 l_1 + R\cdot l)\pi\cdot D\cdot Q^2}{5.2\left(\frac{\pi D^2}{4}\right)^3}\right]\cdot\left[\frac{Q\cdot 0.746\cdot 100\cdot 87.60\cdot C}{6345\cdot E}\right]}^{\text{Power cost 1 yr.}}$$

$$+ \overbrace{\left[\frac{\pi}{4}[(1.2D)^2 - D^2]\frac{l_1\cdot X\cdot a}{27}\right]}^{\text{Lining cost 1 yr.}} + \overbrace{\left[\frac{\pi}{4}[(1.2D)^2\cdot l_1 + D^2\cdot l]\frac{c\cdot a}{27}\right]}^{\text{Excavation cost 1 yr.}}$$

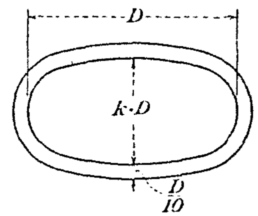
$$+ Y \times a \text{ (move-in cost per yr.)}$$

$$\frac{dM}{dD} = O = -\frac{2.04(R_1\cdot l_1 + R\cdot l)\pi\cdot Q^3\cdot C}{D^6\cdot E} + \frac{44\cdot\pi\cdot D\cdot l_1\cdot X\cdot a}{54}$$

$$+ \frac{\pi\cdot D(1.44 l_1 + l)c\cdot a}{54} + O$$

$$D = 1.960\sqrt[7]{\frac{Q^3\cdot C(R_1\cdot l_1 + R\cdot l)}{E\cdot a(0.44X\cdot l_1 + 1.44c\cdot l_1 + c\cdot l)}}$$

Derivation of Formula for Elliptical Shaft



$$K = (1 + k)\left[1 + \frac{1}{4}\left(\frac{1 - k}{1 + k}\right)^2 + \frac{1}{64}\left(\frac{1 - k}{1 + k}\right)^4 + \frac{1}{256}\left(\frac{1 - k}{1 + k}\right)^6 \ldots\right]$$

|←——Power cost 1 yr.——→|

$$M = \left[\frac{(R_1 \cdot l_1 + R \cdot l)\pi \cdot D \cdot K \cdot Q^2}{2 \cdot 5.2 \left(\frac{\pi k D^2}{4}\right)^3}\right] \cdot \left[\frac{Q \cdot 0.746 \cdot 100 \cdot 87.60 \cdot C}{6345 \cdot E}\right]$$

|←——Lining cost 1 yr.——→| |←——Excavation cost 1 yr.——→|

$$+ \left[\frac{\pi \cdot D^2(0.2k + 0.24)l_1 \cdot X \cdot a}{108}\right] + \left[\frac{\pi \cdot D^2 \cdot c \cdot a[(1.2k + .24)l_1 + k \cdot l]}{108}\right]$$

$$+ Y \times a \text{ (move-in cost per yr.)}$$

$$\frac{dM}{dD} = O = -\frac{1.023\pi \cdot Q^3 \cdot C(R_1 \cdot l_1 + R \cdot l)K}{D^6 \cdot E \cdot k^3} + \frac{\pi \cdot D \cdot (.2k + 0.24)l_1 \cdot X \cdot a}{54}$$

$$+ \frac{\pi \cdot D[(1.2k + 0.24)l_1 + k \cdot l]c \cdot a}{54} + O$$

$$D = 1.775 \sqrt[7]{\frac{Q^3 \cdot C(R_1 \cdot l_1 + R \cdot l)K}{E \cdot a[0.2k + 0.24)X \cdot l_1 + (1.2k + 0.24)c \cdot l_1 + k \cdot c \cdot l]k^3}}$$

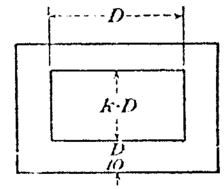
Substituting $(1 + k)$ for K gives the following close approximation:

$$D = 1.775 \sqrt[7]{\frac{Q^3 \cdot C(R_1 l_1 + R \cdot l)(1 + k)}{E \cdot a[(0.2k + 0.24)X \cdot l_1 + (1.2k + 0.24)c \cdot l_1 + k \cdot c \cdot l]k^3}}$$

Derivation of Formula for Rectangular Shaft, Fixed Side Ratio K

$$\overbrace{\left[\frac{(R_1 \cdot l_1 + R \cdot l) \cdot 2 \cdot D(1 + k)Q^2}{5.2(k \cdot D^2)^3}\right] \cdot \left[\frac{Q \cdot 0.746 \cdot 100 \cdot 87.60 \cdot C}{6345 \cdot E}\right]}^{\text{Power cost 1 yr.}}$$

$$+ \overbrace{\left[\frac{D^2(2k + 0.24)l_1 \cdot X \cdot a}{27}\right]}^{\text{Lining cost 1 yr.}} + \overbrace{\left[\frac{D^2 \cdot c \cdot a[(1.2k + 0.24)l_1 + k \cdot l]}{27}\right]}^{\text{Excavation cost 1 yr.}}$$



$$+ Y \times a \text{ (move-in cost per yr.)}$$

$$\frac{dM}{dD} = O = -\frac{1.975(R_1 \cdot l_1 + R \cdot l)Q^3 \cdot C(1 + k)}{D^6 \cdot E \cdot k^3}$$

$$+ \frac{(0.2k + 0.24)D \cdot l_1 \cdot X \cdot a}{13.5} + \frac{[(1.2k + 0.24)l_1 + k \cdot l]D \cdot c \cdot a}{13.5} + O$$

$$D = 1.600 \sqrt[7]{\frac{Q^3 \cdot C(R_1 \cdot l_1 + R \cdot l)(1 + k)}{E \cdot a[(0.2k + 0.24)X \cdot l_1 + (1.2k + 0.24)c \cdot l_1 + kc \cdot l)k^3}}$$