U. S. DEPARTMENT OF THE INTERIOR
HAROLD L. ICKES, Secretary
BUREAU OF MINES
JOHN W. FINCH, Director

Bulletin 385

ENGINEERING FACTORS IN THE VENTILATION OF METAL MINES

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UNITED STATES
GOVERNMENT PRINTING OFFICE
WASHINGTON : 1935

For sale by the Superintendent of Documents, Washington, D. C. - - - - Price 30 cents

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ENGINEERING FACTORS IN THE VENTILATION OF METAL MINES [1]

By G. E. McElroy [2]

INTRODUCTION

The ventilation of a mine has to do with the coursing of air currents through underground openings and may be accomplished by purely natural means or by a combination of natural and mechanical means. The object of ventilating mines is to provide comfortable, safe, and healthful atmospheric conditions at all places where men work or travel.

In all metal mines there is the necessity of removing the fumes of explosives used in blasting. In many (and this group includes most of the important producing metal mines of the world) there is the necessity of providing air conditions that are comfortable so as to maintain a high efficiency of labor. In some there is the need of diluting and removing rock dusts dangerous to the health of the miners or inert but harmful gases, such as carbon dioxide and nitrogen. In a few mines explosive gases, such as methane, or toxic gases, such as hydrogen sulphide, are encountered and require dilution and removal. And in all metal mines it may be necessary at times to contend with underground fires and the toxic gas carbon monoxide produced by them.

During 1929 and 1930, in connection with a comprehensive study of mining methods conducted by the United States Bureau of Mines in cooperation with the leading metal-mine operators of the United States, the author was enabled to make detailed studies of ventilation methods, practices, and costs at a number of large metal mines throughout the western United States. This intimate contact with the officials directly responsible for the ventilation of the mines revealed that the generally high degree of effectiveness of the systems in use was due more to long-continued and expensive development through experience than to the application of available engineering data. That practice lags behind the development of theory and the accumulation of empirical data is due not so much to lack of available information as to its slow dissemination. During the last 10 years interest has revived in the engineering aspects of metal-mine ventilation, owing primarily to the economic necessity of employing larger volumes of air flow to combat the effects of higher-temperature air conditions encountered with increase in depth of many of the world's largest producing mines.

[1] Work on manuscript completed Mar. 30, 1932.
[2] Mining engineer, U. S. Bureau of Mines.

Sufficient theoretical or empirical material is now available to apply to average conditions, but it is scattered through a wide variety of technical publications that are not readily available to mine-operating engineers and is confused by the lack of standards governing the units and the designations used in ventilation literature.

PURPOSE AND SCOPE OF REPORT

In the belief that the quickest and cheapest method of obtaining effective ventilation of metal mines with regard to air conditions that promote efficiency, health, and safety is through more general application of the engineering factors involved rather than by trial and error and that this method of attacking ventilation problems will be facilitated by a simple, concise statement of these factors and the methods of using them, an attempt is made in this bulletin to present the subject of metal-mine ventilation to mine operators in a practical way.

Although the physiological factors of gases, dust, and heat in mine air determine both the necessity for ventilation and the results achieved by it these subjects are not considered within the scope of the present report; and no attempt is made to cover the subject of coal-mine ventilation, for although the general principles are the same as for metal-mine ventilation the particular applications of these principles differ to a considerable degree. An attempt has been made to present the material in as nontechnical a form as is possible without sacrificing necessary accuracy or completeness, for which reason it is thought that it should prove of value to all mine officials and students interested in mine ventilation.

ACKNOWLEDGMENTS

The author has necessarily drawn on knowledge acquired piecemeal from a large variety of sources, including books and periodicals which can only be acknowledged en masse and the publications and files of the Bureau of Mines. For data on present practice and specific conditions he is deeply indebted to mine officials and Bureau mining engineers throughout the western United States for their cooperation in the underground studies recently made as well as in similar studies and specific investigations dating back to 1920. His thanks are also due to many manufacturers of ventilating equipment for special data freely furnished through correspondence and personal conferences.

The courtesy of Prof. W. S. Weeks, of the University of California, whose book [3] on this subject is a standard reference; of A. S. Richardson, ventilation engineer of the Butte mines of the Anaconda Copper Mining Co., which have long served as a pattern for mechanical-ventilation systems applied elsewhere; and of F. Ernest Brackett, in critically reviewing the manuscript, is gratefully acknowledged.

VENTILATION SURVEYS

GENERAL REQUIREMENTS FOR METAL MINES

Ventilation is for the most part unseen and unobserved; surprisingly little accurate knowledge results from casual observations, and many preconceived ideas are upset by the simplest type of systematic obser-

[3] Weeks, W. S., Ventilation of Mines: New York, 1926, 221 pp.

vations. Systematic and periodic surveys, involving physical measurements, are required for all mines to reveal actual conditions. Where hot and humid working conditions are encountered the value of systematic surveys is increased so largely that the benefits have been recognized by many operators, and surveys are made at regular or irregular intervals. Few surveys have been made in naturally ventilated mines, mainly because of a fixed impression that natural ventilation is not subject to control, although a considerable degree of control can be exercised both as to volume and direction of natural-draft currents.

The most direct application of the information gained by a survey comes at the time of a mine fire, when it is vitally important to have the maximum amount of correct information regarding ventilation conditions, particularly the direction of the air currents. Unless a regular procedure of ascertaining and recording ventilation conditions has been followed the necessary information may not be available when and where required. Data so acquired may also be used to advantage in preplanning methods of fire control. Probably the next most important use of such information is in planning for the betterment of present or future conditions of comfort to improve working efficiencies. Other important applications are in connection with fan installations or changes in fan installations of mechanical-ventilation systems, changes in airways, and control of natural draft.

Such surveys demand speed rather than absolute accuracy and involve only simple measurements; but for the information to be properly recorded on maps and otherwise it is preferable for them to be intrusted to a technical engineer who is thoroughly familiar with the mine openings.

Individual requirements, of course, govern particular mines as to how often surveys should be made and what should be included. The essential requirements are that they should be complete enough and be made at sufficiently frequent intervals to keep mine officials posted so that plans for maximum safety and efficiency may be made. In general, the minimum number of surveys to begin with and to continue until the seasonal effect of natural draft is accurately gaged would be two a year, preferably in the late summer and late winter. For shallow, naturally ventilated mines at least four surveys a year are required to give the desirable initial information, whereas for very deep mines yearly surveys will give enough data after the first year or two. Supplementary data are always desirable, however, to ascertain the effect of all major changes in distribution systems.

SURVEY METHODS

The main requirements of a survey are to ascertain the directions of flow and the approximate magnitude of the various air currents. Next to the air-flow information, air and rock temperatures and pressures on doors and stoppings are most useful records for all mines. For mechanically ventilated mines approximate performance data covering all ventilating appliances are of primary interest, and records of the physical characteristics of the airways are required for air-flow computations. In hot, humid mines air conditions are of primary importance, and records of air temperatures and cooling powers are particularly useful.

Exact measurements of all physical conditions involved in mine ventilation would require skillful technique, and descriptions of all the methods that might be used would fill many books. However, refined methods are required only for research and exact tests of appliances. For ordinary surveys of the complex and ever-changing conditions of the ventilation of a mine at relatively infrequent intervals a sufficiently exact picture of the conditions obtaining can be determined by use of a limited number of approximate methods that require little time and more common sense than skill.

A complete set of instruments required for simple ventilation surveys would cost approximately $100. Average prices for good-grade instruments are about as follows:

Prices of instruments required

Low-speed vane anemometer, with leather case and shoulder strap	$35
Sling psychrometer, with leather case and shoulder strap	25
Stopwatch, ⅕ second, 30-minute register	15
Pitot tube, ¼-inch O. D., about 15 inches long	10
U-tube water gage, with rubber tubing	5
Spare psychrometer thermometer for rock-temperature measurements	3
Revolution counter	3
25-foot tape in case	2
Aspirator bulb with hard-rubber valves and rubber tubing	2

These are minimum requirements for simple surveys. For accurate air-density calculations a barometer would be needed for determining the absolute pressure, but the latter may be estimated approximately from altitude and temperature or estimated closely from the reading at some nearby point with which a comparison has been made previously with borrowed instruments.

For more accurate and more inclusive surveys additional instruments that are desirable are an aneroid barometer, a kata-thermometer, and a sensitive pressure gage. The types and designs of instruments that may be used to advantage for indicating and recording ventilation conditions are legion and beyond the scope of this bulletin. In the main, they are special instruments satisfying a rather limited demand and therefore relatively costly. Practically all can only be used under a comparatively small range of conditions, and the number required for even limited research or test requirements is large.

LOW-VELOCITY AIR MEASUREMENTS BY SMOKE-CLOUD METHODS

For determining the direction and velocity of air currents flowing at velocities of less than 150 f. p. m., which observations will ordinarily comprise the bulk of the air-flow observations that should be made in any metal mine, smoke clouds offer the most readily available and practical means. The absolute accuracy of the method, while higher than it is commonly rated, is still very low but not lower than conditions demand. More accurate instruments, including special types of anemometers and the (dry) kata-thermometer, are available, but their use is not justified for mine air-flow conditions except for purposes of research.

A most convenient device for generating smoke clouds under mine conditions is the smoke-tube device developed [4] by United States

[4] Katz, S. H., and Bloomfield, J. J., Sulphur Trioxide Smoke Tubes for Determining Air Currents: Rept. of Investigations 2505, Bureau of Mines, July 1923, 2 pp.

Bureau of Mines engineers. (See fig. 1.) This device produces a dense white cloud which travels with the air currents. Air is forced through a glass tube containing pumice stone saturated with anhydrous tin or titanium tetrachloride, preferably the latter, as it is less corrosive. A rubber aspirating bulb, with hard-rubber valves, is used for forcing air through the tube. To protect pocket linings from the corrosive action of the tube contents and to prevent self-sealing action when the tube is not in use the broken tip should be covered with a rubber tip, either a small cane tip or a pencil-eraser tip, depending on the size of the tube. The tubes may be made up locally as needed or may be obtained from one of the mine safety-appliance companies. This is one of the most useful devices carried by mine-ventilation engineers, since it can also be used to trace low-velocity air currents. A tube of the size shown usually will suffice for the needs of at least a full shift underground.

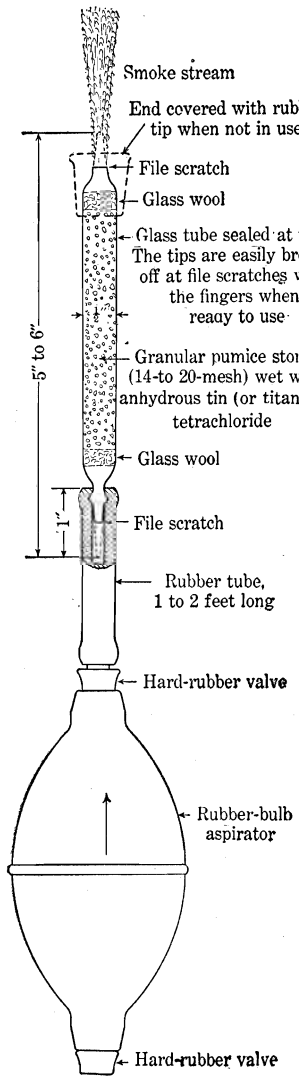


FIGURE 1.—Bureau of Mines type of smoke-cloud tube.

Other methods of smoke-cloud production are, of course, available. Burning fuse has been widely used but is not very satisfactory for many situations. Apparatus for aspirating liquids and finely divided powders are available but are at best somewhat messy and usually have other disadvantages as to bulk, safety, or capacity.

To measure velocities by smoke-cloud travel measure or pace off a known distance along as straight and as uniform a section as can be found on the airway. Station the smoke-cloud producer at one end of this section and an observer with a stopwatch at the other. The smoke-cloud producer should call the starting time of the cloud production, which is made across the airway rather than along it and about halfway between the center and the walls. In slow-moving currents the original cloud can be reinforced with another, or several clouds can be started in different positions in the cross-section by producing the succeeding clouds in line with the advancing cloud first produced. The observer should call the finishing time, which is taken arbitrarily as that of the passing of the middle of the main body of the cloud or clouds. In straight airways of uniform cross-sectional area the velocities found about one seventh of the height or the width of the airway from the walls approximate the

average velocity of the section, but smoke clouds generated to travel so close to the walls would usually be disturbed by eddy currents due to the rough walls. With travel at about the quarter points the readings will average about 10 percent high, and determined average velocities should be decreased by approximately this amount (multiply by 0.9). In irregular airways the distribution of velocities is irregular, and the time of travel of clouds started in different positions in the cross-section should be averaged. These results will also average high and should be decreased arbitrarily about 10 percent. With very low velocities of flow through airways where rock and air temperatures differ considerably there are often large differences between the velocities in the upper and lower parts of the cross-section, but these can be roughly averaged from separate cloud travels. The smoke clouds, however, are affected to a relatively larger extent by the higher-velocity air streams, and deductions of about 20 percent from observed average velocities (multiply by 0.8) are required for approximate accuracy. Disturbances due to car, cage, and skip travel in nearby airways should be expected in the vicinity of such activities, and an attempt should be made to obtain measurements during normal periods of flow only. One of the great advantages of smoke-cloud methods over more accurate methods for mine work is the fact that the effects of disturbances are readily observed. It is also possible to take measurements in airways virtually inaccessible or dangerous to enter without safeguards that would take time to provide and might interfere unduly with mining activities.

To determine quantities of flow it is, of course, necessary to know the average area of cross-section of the length of airway used for the velocity observation. Under ordinary conditions the length is best taken as 25 feet. With good light and with velocities near and above 100 f. p. m. this distance might be increased to 50 feet in a regular airway. With very low velocities and where required by the airway conditions it might be reduced to 10 or 15 feet. In untimbered rock airways average areas based on a few measurements may be in error as much as 10 percent, but more refined measurements ordinarily will not give the area actually occupied by the flowing current much more accurately. Individual determinations of quantities of flow in rough-walled airways may be in error as much as 30 percent by smoke-cloud methods of measurement, due to combined errors in average velocity and average area determinations; but, in general, it will be found from check measurements that relative agreement within 10 percent of more precise air measurements is usually obtained in rapid, though careful, work.

MODERATE-VELOCITY AIR MEASUREMENTS BY VANE-ANEMOMETER METHODS

For measuring velocities of 150 to about 2,000 f. p. m. in mine airways (a range that includes the major air currents of the average mine) ordinary commercial types of low-velocity vane anemometers are most practical, convenient, and accurate. Other methods of accurate measurement of velocities in this range are available but have been developed mainly for application to small ducts and have serious disadvantages for rapid surveys of mine-ventilation conditions.

The vane anemometer is a small windmill geared to a totalizing mechanism through a small clutch, which can be engaged at will. The usual type used in American metal mines is the Biram type shown in figure 6, *A* (p. 17); the common size is about 4 inches in diameter. Late models are provided with a threaded hole through which, by means of a small attachment usually provided with the instrument, it can be mounted on a small-diameter shaft. For precise work the anemometer is mounted on a shaft, but for ordinary rough measurements it is held in the hand as shown. In all cases it is important that the dial side face away from the air current.

Another type of vane anemometer, which is useful in that it can be more easily carried around for occasional rough measurements, is the watch type, also shown in figure 6, *A*.

There is a general impression among mining engineers that anemometers are unreliable, an impression created partly by their own attempts to use them improperly and partly by the expressed opinions of engineers in other lines of endeavor who are more familiar with methods of measurement applicable to small ducts of uniform cross-section and who have no idea of the variations in mine-air flow. The author has had an extensive and intimate acquaintance with anemometric air measurement in mines, with facilities for comparing methods of air measurement, and has found anemometers very reliable when properly used. The accuracy of results is affected to a large extent by the exact method of use but not to a much greater degree than any other device used under similar circumstances. The anemometer has the important advantages that its readings are practically independent of air density and (for the range encountered in mines) of air temperature and that it is affected but very slightly by nonparallel flow. Accurate measurements of air flow can be obtained only by carefully traversing a cross-section at right angles to a straight air passage of regular area a considerable distance downstream from any disturbance of the flow and from a position outside the airway and then applying correct calibration factors to observed readings. However, an average accuracy within 5 percent, which is close enough for mine-air distribution records, can be obtained with less-refined methods if calibration and method factors are used and areas are carefully determined.

Anemometer calibrations.—Vane anemometers usually register a velocity that is somewhere within 10 percent of the true velocity; the latter has to be determined by referring to the results of calibration tests. In this country the National Bureau of Standards, Washington, D. C., is the recognized authority on anemometers and is prepared to calibrate these instruments for a nominal fee. Calibrations are made on a horizontal whirling arm for low velocities, with correction for the air swirl set up by the apparatus, and in a horizontal wind tunnel for high velocities. The use of a shaped nozzle or orifice to give uniform parallel flow at the discharge of a small fan-pipe installation would appear to provide a simple and accurate field method of calibrating anemometers for both horizontal and vertical flow, but it has not been used, to the author's knowledge.

Calibrations furnished by the manufacturer with new instruments are usually in the form of tables of plus and minus corrections to be made to observed velocities per minute, as figured from the total reading and the total time of operation, and usually agree with Bureau

of Standards calibrations within 2 percent. The latter are issued in the form of a curve, figure 2, *A*, in which a factor, by which the observed velocity must be multiplied to ascertain the true velocity, is plotted against the observed velocity.

It has been found by experiment [5] that the registered velocity of a vane anemometer is a straight-line function of the true velocity, as shown in figure 2, *B*, which can be represented by the general equation

$$V_T = A + BV_R, \tag{1}$$

where V_T is the true velocity, V_R is the registered velocity, and A and B are constants for the particular anemometer. Apparently, according to calibration curves for a great variety of anemometers, A remains constant for any one instrument, is virtually the same for all instruments of a general type, and averages about 30 for the common 4-inch, Biram-type, low-velocity (8-blade) instrument. The author has records of calibrations in which A ranges from 0, for a Rosenmuller special low-velocity instrument, to 60, for a Biram-type, high-velocity (4-blade) instrument. Apparently, also, the value of B in the calibration equation is practically constant for any one instrument that is kept fairly clean, well-oiled, and not damaged mechanically and varies but a few percent between instruments of the same design and make, but varies largely—at least over the range 0.90 to 1.15—for different designs and makes. It appears possible that an instrument could be made with B equal to 1.00, in which case it would only be necessary to add constant A to the registered velocity. The influence of design on constant A makes it appear probable that an instrument could be designed with values of 1.00 for both A and B, in which case no calibration would be required.

The straight-line method of plotting calibration results has a number of advantages. First, the straight line will be a "faired-up" average of individual calibration-test results; second, the calibration may be based on tests at but two velocities or, if constant A is known from previous calibrations, on one (preferably high) velocity only; third, the constants for a particular instrument may easily be memorized and applied directly to observed velocities.

Calibrations are available for horizontal flow only, and it is assumed that these hold also for vertical flow upward and downward. Complete comparative calibrations to prove this point are desirable but are not available. The author has made a few comparative traverses in shafts where average velocities were measured by more exact methods and found no perceptible discrepancies at moderate to high average velocities (1,000 to 2,000 f. p. m.). Comparison of air measurements made in horizontal and vertical airways in mines has likewise indicated no larger discrepancies than similar comparisons of horizontal-flow measurement.

The effect of air density on calibrations, according to Ower, is negligible, except for very low velocities and large density differences. If d is the density where the observation is made and d_s the density at which the calibration was made (usually about 0.075 pound per cubic foot), constant A in the calibration equation should be changed to $A\sqrt{\frac{d^s}{d}}$ to allow for the air-density effect. For the Biram type of

[5] Ower, E., Measurement of Air Flows: London, 1927, pp. 101–134.

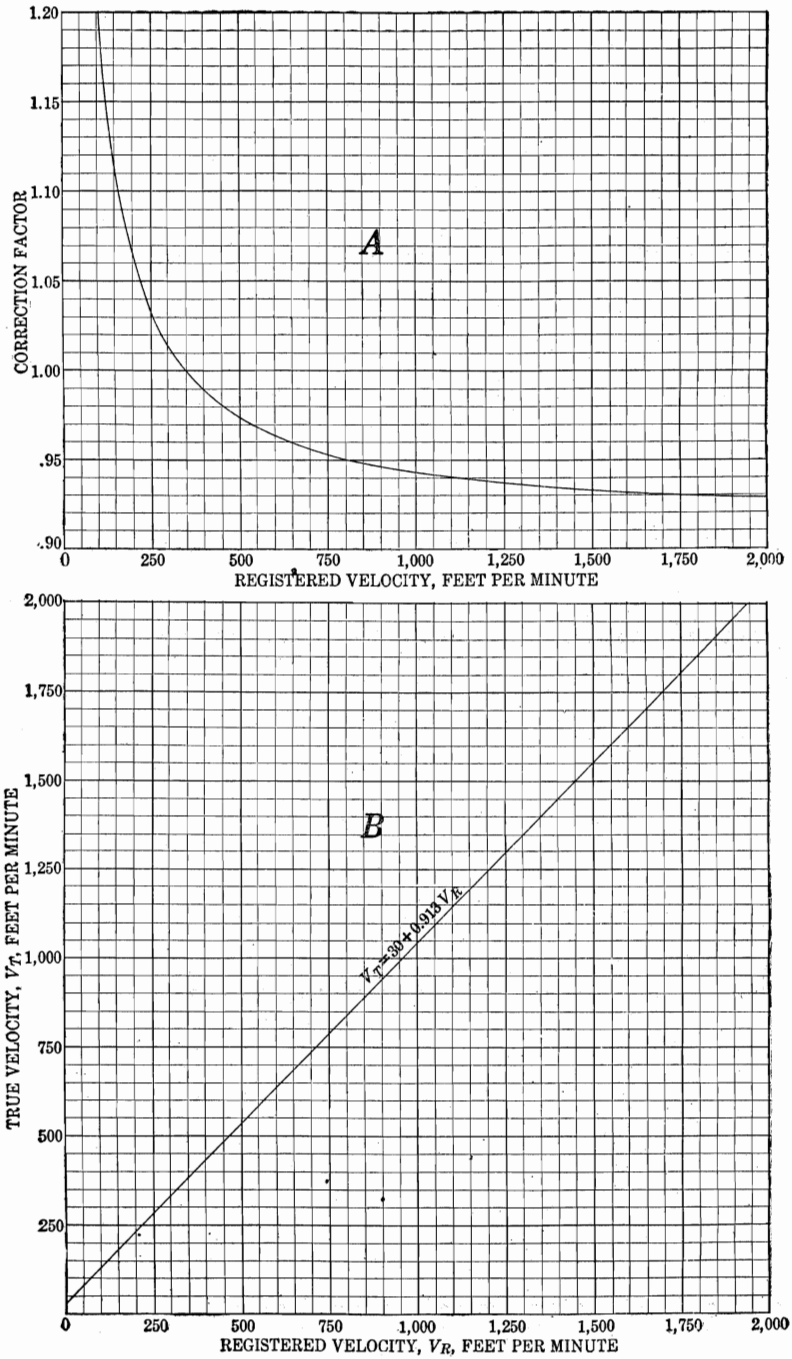


FIGURE 2.—*A*, Calibration curve for vane anemometer; *B*, calibration chart for vane anemometer.

low-velocity anemometer this change results in a difference of but approximately 3 f. p. m. as between sea level and points 5,000 feet above or below sea level.

Ower's tests have also shown that the accuracy of an anemometer is affected to but a slight extent when its face is not exactly at right angles to the direction of flow. With the face up to 20° off line the effect on the accuracy of readings was less than 2 percent, a result that is especially significant for mine work where the directions of flow are commonly quite variable.

Method factors.—For rapid mine-ventilation survey work the anemometer must be used under conditions differing from those under which it was calibrated, and allowances must be made for such differences to obtain approximately accurate results. After differences between individual anemometers are eliminated by applying calibration factors to the registered velocities, it is necessary to multiply the result by another factor, termed the "method" factor, to get the true average velocity over a particular cross-section to allow for certain changes between conditions of use and conditions of calibration. Important considerations are: Whether average velocities are based on one-point readings or on traverses of the section; whether the traverse is made by a precise or by an approximate method; whether the anemometer is mounted on a shaft or held by hand; whether or not the air stream itself has some abnormal characteristics at the place of measurement; and whether or not the anemometer itself causes an appreciable change in the normal conditions of flow. Particular methods of use usually involve a number of these considerations to a varying degree, but the author has found by experiment that average method factors can be applied for a number of common methods of use to give satisfactory accuracy.

ONE-POINT MEASUREMENTS

The simplest and fastest method commonly used is that in which the average velocity over the cross-section is based on a measurement at but one point in the section. With the operator's body out of the air current and the anemometer mounted on a shaft, fair results may be obtained in straight airways of uniform cross-sectional area. With the anemometer held at the center of the section, a method factor of 0.80 should be used. When it is held about one seventh of the wall-to-wall dimension from one wall, or preferably two, the method factor is 1.00, but results are not as consistent as for the center position on account of the effect of minor irregularities usually present in the wall surfaces. However, mine airways that are straight and uniform enough to permit accuracy within 5 percent by this method are most often shafts carrying the main intake or return air currents, for which it is generally desirable to obtain more accurate measurements by careful traversing.

In irregular airways the velocity distributions cannot be foretold with any degree of accuracy, and the method is not applicable. Actual tests in mine openings under comparable conditions have resulted in calculated method factors ranging from 0.65 to 0.90, with very little correlation of factors to observable differences in conditions. However, one-point measurements of velocity in irregular airways do indicate relative changes in quantity of flow, provided they are made at the same point in the same cross-section.

APPROXIMATE TRAVERSING METHODS

Most of the measurements made by anemometers in mines are on minor airways where approximate traverse methods, combined with the use of method factors, are best suited to the conditions encountered and give sufficient accuracy, that is, within 5 percent on the average. Continuous traverses made by moving the anemometer slowly through a plane at right angles to the axis of the airway and along lines approximately parallel to the walls and not more than 1 foot apart, as shown in figure 3, *A*, are recommended. The total time should be gaged by the dimensions of the opening and the relative accuracy of the results desired; 1 minute for a 30-square-foot opening and up to 3 minutes for a 100-square-foot opening are average figures. The speed of traversing should be uniform throughout the section, and for this reason it is practically necessary to have an assistant call out the 5- or 10-second intervals. Any regulation of speed to time should preferably be done when the anemometer is about one seventh

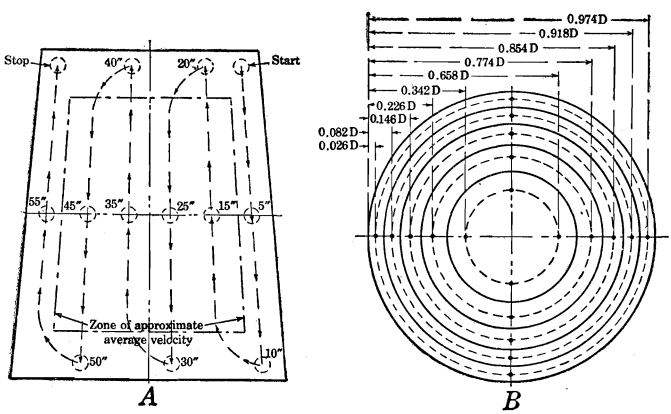


FIGURE 3.—*A*, Diagram showing path followed with anemometer in the continuous approximate method of traversing air-measurement sections of mine airways; the positions of the anemometer are indicated for 5-second intervals of a 1-minute traverse. *B*, Diagram showing position of traverse points in a circular air-measurement section for a 5-area 20-point traverse.

of the dimensions from the walls, that is, in the zone of approximate average velocity. With a little practice it will be found that check measurements within 2 percent can be obtained under almost any conditions. As an extreme illustration of this point it may be cited that the author obtained this degree of relative agreement in 3 sets of duplicate traverses of arched tunnel sections 27 feet wide and 20 feet high with an anemometer mounted on a 14-foot shaft and operated from the top of a 12-foot stepladder.

For horizontal or slightly inclined openings the anemometer may be held by hand at arm's length while the observer moves slowly across the section as far as is necessary to complete the traverse. The method factors to be used are not very well authenticated but have proved satisfactory in the author's experience. They depend mainly upon whether or not the traverse is started with the operator's body in or out of the current. If started with the operator's body in the current, as in unlined rock sections, the first part of the traverse is made close to the body in a zone of locally increased velocities, and

the factor to use is 0.85. If the traverse can be started with the operator's body shielded by a timber or a doorframe the factor to use is 0.90. This same factor can be used if the operator stands in the airway and traverses each half separately, starting each with his body at arm's length from the center. These factors are for ordinary metal-mine drifts of 35- to 45-square-foot areas. For larger openings the ratio of area of body to area of opening is less, and the relative effect on method factors is less. For 80-square-foot openings the normal factors of 0.85 and 0.90 should be increased to 0.88 and 0.92, respectively, and to 0.90 and 0.93, respectively, for openings of 100 to 120 square feet.

The fact that an obstruction, such as the operator's body, changes the normal velocity distribution of the air stream both alongside and for a considerable distance upstream is rarely recognized; only the downstream disturbance is commonly considered. To take a special case, anemometers are sometimes operated on the end of a short shaft by an operator standing downstream and in line with the anemometer position. Readings taken in this way will give results probably 5 to 10 percent low. Then, again, where separate readings are taken at each of a number of properly arranged traverse points the starting and stopping of the anemometer, unless some method of remote control is used, results in some obstruction of the air currents twice for each reading, which will have some effect on the accuracy of the observations, depending on the time used for each observation.

For vertical and steeply inclined airways it is desirable to mount the anemometer on a shaft. With the anemometer mounted on a shaft and started and stopped from points close to the wall on one side of the airway very little interference with the air currents results, and no method factors are required. Actually, on account of exaggerated looping in low-velocity areas close to the walls, the results in a passage of uniform area will usually average about 2 percent low and require a factor of 1.02; but at constrictions, such as doorframes, the velocities close to the walls are relatively high, and the results will average about 2 percent high and require a method factor of 0.98. However, by avoiding excessive looping close to the walls in continuous traverses, the necessity for applying a method factor is avoided.

PRECISE TRAVERSING METHODS

Precise traversing methods with anemometers can yield results within 2 percent of absolute accuracy but are justifiable only on major air currents or in connection with tests of primary fan performance to determine pressure-quantity relations; they should be applied only at sections of regular area on straight airways well away from disturbing influences. Traverse points should be located—by strings, wires, or marks on guideframes—at the centers of approximately equal areas of the cross-section, which is assumed to be rectangular. Their number depends on the shape of the section and the conditions of flow and should not be less than 16 for a square section or less than 24 for a section in which one dimension is twice the other. Although a maximum spacing of 6 inches is often recommended for test work in relatively small ducts a minimum spacing of 1 foot is satisfactory in mine airways. If the conditions of distribution are such that closer spacing would seem desirable, it is very probable that

the conditions of turbulence would prevent a high degree of accuracy by any method of measurement.

Assuming that it is possible to work from without the section or from a recess, precise traversing is most practically and conveniently carried out under mine conditions by what might be termed "continuous point traversing." In this method the anemometer is not started and stopped at each traverse position but is started at the first position, is slowly moved to a new position at the end of each 10-, 20-, or 30-second period, and is stopped at the last position. Traverse points may be arranged so that the start and stop positions are on the same side of the airway; or the traverse points may be covered twice to bring the stop position at the same point as the start position. Narrow airways, up to 5 feet in width, can be covered from one side with an anemometer attached to a 30-inch shaft with but little interference from the operator's arm. Airways up to 10 feet in width can be covered in halves, one half from each side, and where conditions permit narrow airways can be covered similarly, with possibly a slight gain in accuracy. For airways wider than 10 feet it is desirable to stretch taut wires across the airway and to use a small hook attachment to suspend the anemometer, which is placed as to position laterally by means of a shaft, as before, either from one or both sides. Pulley and guide arrangements have also been used for traversing very wide or very high airways.

Separate-point traversing methods—that is, methods involving separate velocity observations for each traverse point—are often used for precise work but are too slow for ordinary mine work, since at least 1- or 2-minute readings are required at each point to minimize inaccuracies due to interference with the air currents in starting and stopping the anemometer, and further time is required to change the position of the anemometer and secure it in the new position. The time intervals may be shortened by using some system of remote control to start and stop the instrument, such as a flexible wire in a flexible-coil casing, a small air-operated piston, or a small solenoid; but anemometers usually are not so equipped, and if the added device is at all large a new calibration is required to gage its effect. However, where it is impossible to work from a recess communicating with the section some such method is required for precise work; and these methods can, of course, be applied from a point without the section. Nevertheless, with either continuous-point or separate-point traversing, results accurate within about 2 percent can be obtained with careful work at good sections without the use of method factors, and the slowness of separate-point traversing usually is not justified.

At doorframes which constrict the area of the airway abruptly about 50 percent or more the ordinary number of points do not cover the higher-velocity areas close to the walls closely enough, and a method factor of about 0.95 is ordinarily required for 16-point traversing.

In order that traverse points on circular sections may represent equal areas the circular section is divided into equal areas by concentric circles, as shown in figure 3, *B*, and traverses are made on two diameters perpendicular to each other at points determined by their intersection with the alternate circles shown dotted in the figure. The full circles are the boundaries of equal areas, and the dotted circles are the mean radii of these equal areas. Four points are

taken for each ring of equal area—that is, 12 for a 3-area traverse and 20 for a 5-area traverse. The latter is recommended for precise traversing. If D is the diameter of the airway the distances from the wall to the points on each diameter will be 0.044 D, 0.146 D, 0.296 D, 0.704 D, 0.854 D, and 0.956 D for a 3-area 12-point traverse and 0.026 D, 0.082 D, 0.146 D, 0.226 D, 0.342 D, 0.658 D, 0.774 D, 0.854 D, 0.918 D, and 0.974 D for a 5-area 20-point traverse.

With anemometers mounted on a shaft operated on a carefully constructed guideframe relative agreement of successive readings to 0.1 percent can be obtained under uniform flow conditions, and absolute accuracy better than 2 percent can be obtained under the best conditions of mine-air flow.

HIGH-VELOCITY AIR MEASUREMENTS BY PITOT-TUBE METHODS

The use of the ordinary vane anemometer in velocities over 2,000 f. p. m. is liable to crack the jeweled bearings, and at velocities above 2,500 f. p. m. this result is almost certain. Special high-speed anemometers, usually with half as many vanes as the standard type, are made for this work but seldom are required for mine measurements. The high velocities that it is most often desirable to measure are those that occur inside small pipes or tubing where it is impossible to use an anemometer. Here the velocities are high enough to permit the use of methods involving simple pressure-measuring instruments; and the use of a Pitot tube, which can be inserted through a small hole punched in the pipe or tubing, is convenient for obtaining the pressures, which can be measured on a U-tube water gage. Since a Pitot tube and a pressure gage are desirable for certain other pressure measurements in mine-ventilation surveys, the use of these instruments, rather than a high-speed anemometer, is recommended for this work, even though the special high-speed anemometer is more convenient to use.

The Pitot tube is a primary standard instrument for measuring velocities. Commercial types are generally accurate within 1 percent, and types based on recent research can be made to be accurate to within 0.1 percent. The usual form, figure 4, is two concentric tubes, one within the other, with the end bent at right angles to the shaft. The inner tube is open on the end and receives the total pressure of the air stream. The outer tube is closed on the end and receives the static pressure of the air stream through a number of small pinholes set back from the end. The tip is tapered or is made hemispherical to reduce interference with the air stream to a minimum. The opposite end of the tube has two connections, one to each tube. The difference in pressures obtained with the instrument is the velocity pressure at its tip when it is pointed directly against, and parallels, the air stream; it is usually determined and expressed in inches of water pressure. Velocity pressures vary as the square of the velocity and directly as the density of the air. The constants in the formulas connecting velocity with velocity pressure vary slightly with location owing to slight variations in the acceleration due to gravity, but sufficiently accurate formulas are:

$$V=1097.4\sqrt{\frac{H_v}{d}}, \text{ or } H_v=0.000000831\, d\, V^2, \tag{2}$$

where V=velocity, feet per minute;
H_v=velocity pressure, inches of water (at 60° F.);
d=density of air, pounds per cubic foot.

At standard air density of 0.075 pound per cubic foot,

$$V=4008\sqrt{H_v}, \text{ or } H_v=0.0000000623\, V^2. \tag{3}$$

Velocity pressures ordinarily are so small in mine openings that sensitive differential-pressure gages are required to measure them accurately. Even with a velocity as high as 2,000 f. p. m. the velocity pressure is only about one fourth inch at standard air density, and with a gage reading to hundredths approximate accuracy only is obtained.

Since velocity pressures vary as the square of the velocities the mean velocity at a section cannot be obtained accurately from the direct

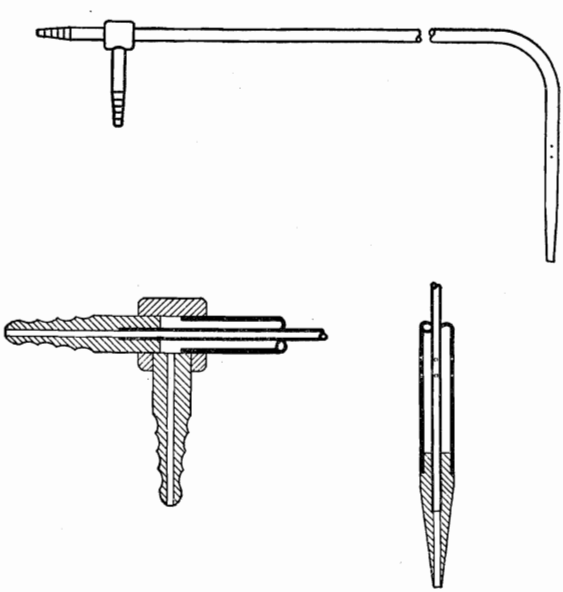


FIGURE 4.—Pitot tube, with details of ends.

mean of the velocity pressures obtained at traverse points but must be obtained as the mean of the separately computed velocities or from the square of the mean of the square roots of the separate traverse-point pressures. The error in the velocity determined from the direct mean of pressures, however, is usually less than 2 percent, and this refinement can be neglected in approximate measurements. Approximate traverse methods for fan-pipe air measurement are discussed in a later part of this section.

ACCURACY OF MINE AIR-FLOW MEASUREMENTS

Good relative agreement is, of course, not the same as absolute accuracy. In general, rough measurements of velocities and areas for minor air distribution will give flow measurements with an average

accuracy of about 10 percent, and the sum of a group of 10 or more measurements will probably be within 5 percent. Careful measurements of velocities in major air courses by approximate traversing methods, with the use of the method factors given, combined with careful area measurements, will give flow measurements with an average accuracy of about 5 percent, and the sum of a large group will probably be within 2 or 3 percent. Considering the normal variations in air quantities in mines due to changing natural-draft conditions, variations in the total amount of compressed air liberated, and the effect of many other uncontrollable variables, these degrees of accuracy are precise enough for practical purposes. However, in determining fan performance and for certain other purposes, more accurate flow measurements may be desirable, and an accuracy within about 2 percent can be obtained by precise traversing methods and close attention to details. Under mine conditions it is hardly possible to obtain accuracy much under 2 percent, even with precise methods of air measurement; such work calls for specially skilled technique, expensive preparations, and instruments not warranted except for research.

Considering the inaccuracies of most mine air-flow measurements, the practice of reporting quantities to the last cubic foot per minute is unwarranted. It is suggested that quantities below 20,000 c. f. m. be reported to the nearest 500 c. f. m. only and that above 20,000 c. f. m. even thousands should be considered sufficiently precise.

In considering the relative agreement of the quantities of flow as measured in the various parts of a mine the effect of compressed air liberated in the workings and of changes in air density on quantities of flow should be taken into account, since either may account for differences of at least 5 to 10 percent in almost any mine.

TEMPERATURE OBSERVATIONS

Except on intake airways within 1,000 feet of the surface, mine-air temperatures change very slowly; readings at 500- to 1,000-foot intervals in shafts and at 1,000- to 2,000-foot intervals on working levels, combined with readings at the face of every third or fourth working place, give a sufficient indication of air-temperature conditions throughout the mine, although additional readings will be required to establish the effect of local conditions. Wet- and dry-bulb temperature observations are required for determining relative humidities, absolute humidities (water-vapor content), total heat contents, and air densities.

SLING PSYCHROMETERS

Wet- and dry-bulb mine-air temperatures are conveniently obtained with a sling psychrometer, which is merely an assembly of two thermometers mounted on a frame equipped with a pivoted handle which permits whirling of the frame to obtain the necessary air movement over the bulb of the wet-bulb thermometer. The Bureau of Mines type, figure 5, is in common use in American metal mines, and the design includes both an aluminum (*a*) and a leather (*d*) case for full protection during transportation underground. It carries 10-inch thermometers mounted on a thin aluminum frame with stiffening edges. A recent improvement, not as yet incorporated in the stand-

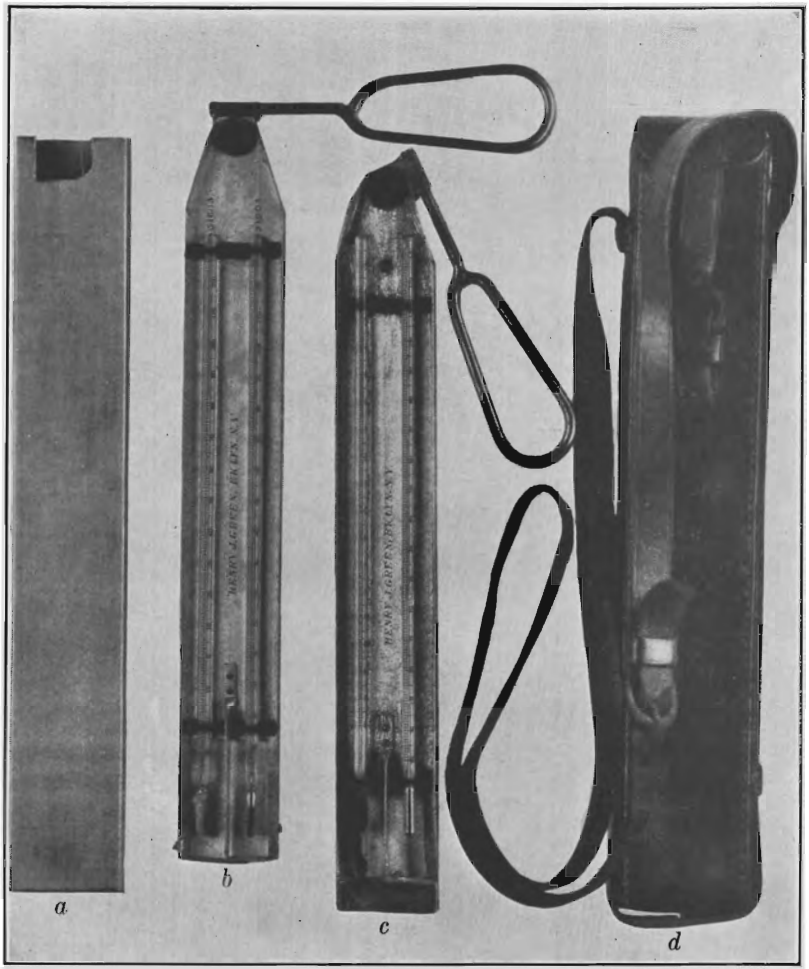
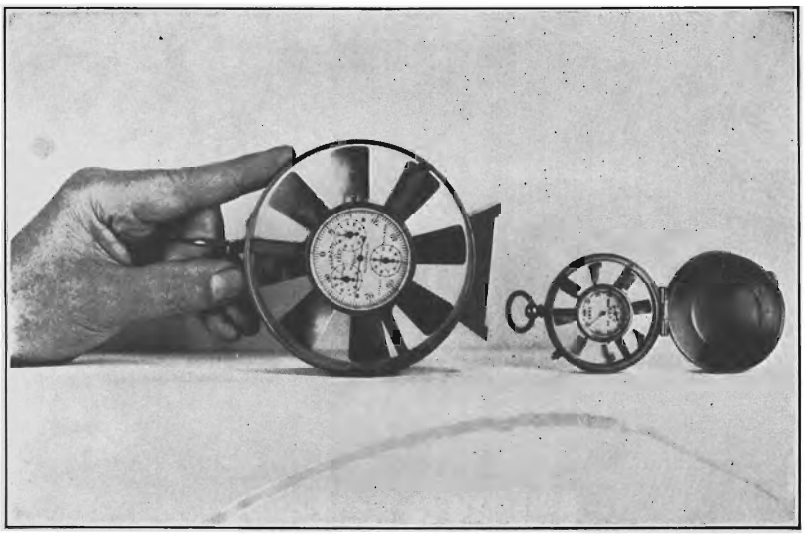
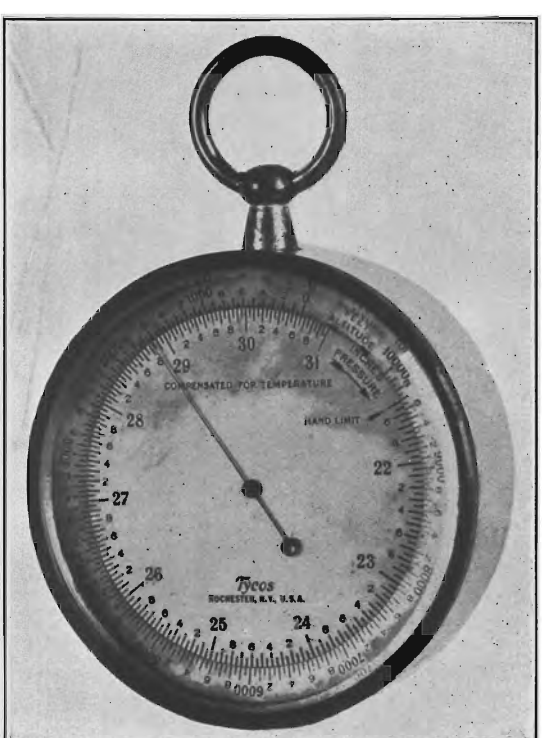


FIGURE 5.—Bureau of Mines type sling psychrometer and carrying cases: *a*, Aluminum case; *b*, standard design; *c*, water well added to standard design; *d*, leather carrying case.



A



B

FIGURE 6.—*A*, Biram-type vane anemometers; *B*, standard type of small aneroid barometer.

ard design (*b*), is the addition of a water well across the bottom (*c*). The well can be made of thin brass the width and thickness of the frame and three eighths inch deep. The top edge has a one-sixteenth-inch slot across the width to admit the tubular wick of the wet bulb, preferably of "china silk" about 6 inches long and about the diameter of the bulb. The wick is doubled up in the small well and completely fills it. A small hole in the back of the well permits filling it with a medicine dropper and is closed with a large-head milled-edge screw. The use of a dropper is not required if clear water in which the end may be immersed is available. A few minutes' immersion saturates the wick enough for a day's work in the ordinary damp to wet mine. With a saturated wick in the well no trouble has been experienced in low-temperature weather, and there is no leakage through the wick slot. With free water in the well there might be leakage, and freezing of the water might burst the well.

Thermometers with the scale engraved on the glass are required for accurate work. The standard 10-inch psychrometer thermometers, graduated from about −20° to +130° F., are usually accurate within 0.2°; readings can be estimated in good light to 0.1°, but they usually are not read closer than 0.5° under mine conditions. For close readings, involving holding a light close to the instrument, a flashlight should be used, and in air currents the operator should stand sidewise to or face the current to prevent the heat of his body from affecting the reading. The frame usually is not at air temperature when the reading is started; and whirling, with occasional observations, is required until both wet- and dry-bulb thermometer readings are constant. The wet-bulb reading should always be made first, as it changes quickly when the instrument is brought to rest. It is desirable to record the readings always in the same order, as 60.0–61.5 for 60° wet bulb and 61.5°, dry bulb. For ventilation purposes the wet-bulb temperature is more significant than the relative humidity, and it should always be retained in records.

Obtaining wet-bulb temperatures for temperatures below freezing is very slow work, as the evaporation of ice on the wet bulb is very slow, and there is an initial heating of the wet-bulb thermometer due to the heat release accompanying the change from water to ice on the bulb. Persistent whirling will, however, give results; but if the ice is evaporated before the true wet-bulb temperature is determined, indicated by a rise in the reading, it is necessary to add more water to the bulb, let it freeze, and repeat the process. Correct wet-bulb temperatures a few degrees below 32° F. may be obtained without the wet bulb freezing.

Ordinary surface drinking water is clean enough for psychrometer use, although distilled water is preferable if conveniently obtainable. Mine water should not be used, as it often contains a high enough salt, or other chemical, content to affect the wet-bulb reading seriously. Dirt gradually accumulates in the wet-bulb covering and finally prevents it from absorbing water readily. A new covering is then desirable and is easily applied as follows: Cut a piece of muslin (from a roller bandage) a little longer than the bulb and wide enough to make a good lap; wash it thoroughly and wrap it around the bulb while wet; tie the upper end at the narrow groove above the bulb, and tie the lower end with a loop started just where the bulb starts to round off. As the loop is drawn taut it will slip down over the rounded

end and draw the muslin taut; the excess muslin below the bulb should then be cut off. Any strong thread will do for tying.

RECORDING THERMOMETERS

Various instruments for recording wet- and dry-bulb temperatures and relative humidities are available. Those for dry-bulb temperatures and relative humidity are relatively simple and cheap. Those for wet-bulb temperatures usually are combined wet- and dry-bulb recorders for either quiet or moving air and are relatively complex and expensive. The ordinary types of wet-bulb instruments require water wells and do not operate much below freezing temperatures.

Maximum and minimum indicating thermometers, which may be obtained in a variety of designs, are convenient instruments for determining temperature fluctuations on the surface or underground and where the fluctuations are quite uniform are useful for approximating average temperature conditions over long or short periods.

ROCK TEMPERATURES

Rock-temperature observations are commonly made for determining the geothermic gradients or the rate at which the temperature of the rock in undisturbed areas increases with increase in depth below the surface. Gradients of 20 to 250 feet per degree (F.) have been reported, but most metal mines have gradients that fall within the range of 50 to 100 feet per degree.

Although many methods and instruments may be used a 10-inch psychrometer thermometer mounted in a groove in one face of a three-fourths-inch square softwood block makes an easily carried and robust instrument. It can be inserted in any ordinary drill hole, assumes the rock temperature within about 10 minutes, and changes temperature very slowly on being withdrawn from the hole.

Determination of rock-temperature gradients.—On account of the major effect of circulating air currents, water flow, drainage, and many other factors it is difficult to get good data in short holes extending from mine openings that will show the temperatures in undisturbed areas or the virgin-rock temperatures; and many misleading data have been published. Lacking precise data on the effect of various factors on the accuracy of temperature-gradient determinations, experience is the best guide. The author has found that temperature observations made in 5- to 6-foot holes in rapidly advancing development headings on the lowest levels of a mine, correlated with Weather Bureau or private records of the average mean dry-bulb surface-air temperature over a period of years, yield approximately accurate gradients. Direct comparisons of observations made on different levels underground usually are confined to relatively small temperature and depth differences, with resulting greater effect of small inaccuracies on deduced gradients. The temperature of the ground 40 to 50 feet below the surface has been found to average within a degree or two of long-time averages of the dry-bulb temperature of surface air. A better base figure would be a virgin-rock temperature close to the surface, but access to undisturbed areas usually is impracticable. Where the surface is very irregular the correct depth figure is not the

vertical depth but the shortest distance between the point of observation and the surface, since the temperature contours of the rocks follow the surface contours.

An allowance of 1 hour is assumed to be ample for the heat of drilling a hole and the effect of cool water used in drilling to be dissipated into the rock so as not to yield a substantially inaccurate observation. Ventilation of the heading, of course, cools the rock, but under ordinary circumstances the cooling does not penetrate to the bottom of a 5-foot hole during the drilling cycle. High-velocity ventilation of the face with air well below rock temperature, a condition usually confined to places where the rock temperatures are over 90° F., will cool the rock rapidly, and observations should be confined to rapidly advancing faces shortly after exposure. The thermometer needs to be left in only long enough to be certain that its casing has risen to rock temperature.

ABSOLUTE-PRESSURE MEASUREMENTS

Absolute pressures of the atmosphere are required mainly for air-density determinations, particularly in connection with natural draft, fan performance, air sampling, and change-in-quantity calculations. For approximate purposes they may be estimated from tables of average values for various altitudes and temperatures;[6] but for any point the absolute pressure may change as much as 1 to 2 inches of mercury during the year, and actual observations are therefore more satisfactory for general use.

The standard apparatus is a laboratory-standard mercurial barometer of the wall type in which the pressure of the atmosphere holds the mercury at a certain level in an evacuated tube, and various precautions must be observed both in the construction of the barometer and in correcting observed readings to get accurate results.

ANEROID BAROMETERS

For mine work the aneroid barometer (fig. 6, *B*) is commonly used, and a small rugged type is preferable. Inside the outer case is an evacuated chamber of corrugated metal. Atmospheric pressure tends to collapse this metal chamber, which is partly resisted by a spring. However, partial collapse occurs, the extent varying with the pressure, and is communicated to a pointer on the face of the instrument through levers and a small chain and sprocket. Compensation for temperature effect on the lever mechanism is accomplished by making 1 of the levers of 2 or more metals such that lengthening is compensated by bending. Very cheap instruments often are not compensated for temperature, although so marked; and most aneroids require time for compensating large temperature and pressure changes, since it takes time for the operating mechanism to take the new temperature, and an effect termed "hysteresis" causes a lag in the pressure indication. The most expensive and sensitive types of the standard aneroid, which are provided with reading lens and vernier reading attachment and are usually quite large—4 or 5 inches in diameter as against about 2½ inches for the simpler instrument—suffer more frequent changes in calibration from the rough handling incident to mine use and from the sudden large pressure changes in

[6] Buffalo Forge Co., Fan Engineering: Buffalo, N. Y., 2d ed., 1925, p. 20.

vertical shaft transportation than do simpler, more rugged instruments. For accurate work it is essential that aneroids be compared frequently with a laboratory-standard wall-type mercurial barometer such as is usually available at any United States Weather Bureau office or physics department of a school or university.

Average sea-level barometric pressure usually is taken as 29.92 inches of mercury, and the limits of the standard aneroid ordinarily are 20 and 31 inches. For mine work a higher or lower range than this sometimes is required, necessitating a special instrument.

An aneroid barometer occasionally is graduated in feet or other units of altitude, since differences of absolute pressures may be used for roughly determining differences of altitude; it is then termed an "altimeter." The ordinary aneroid usually is graduated in inches of mercury on the fixed inner circle and for convenience of use as an altimeter is graduated in feet or other linear units on an outer adjustable circle.

Paulin aneroid.—A new type of sensitive aneroid, perfected in recent years by Paulin System, Inc., makes use of a null, or zero-gaging, principle of indicating pressures in a manner that eliminates virtually all inaccuracies of the standard type, in which the vacuum capsule is allowed partly to collapse. In this instrument the top of the capsule is brought back to zero position for each reading by a spring operated by a slow-motion screw; and by means of two or more concentric scales a very fine and accurate reading may be obtained, the readings being made to one one-thousandth of an inch mercury compared with one one-hundredth on the standard aneroid.

SMALL ABSOLUTE-PRESSURE DIFFERENCES

The measurement [7] of absolute pressures with enough precision to permit direct determination of small pressure differences due to flow of air in mine openings has always been desirable on account of the relative ease of such methods of measurement, but instruments with which even approximate results may be obtained have been developed only recently. In addition to the precision aneroids other types of instruments are being developed for this particular purpose. One type, represented by the Askania Statoscope, makes use of the null method in an aneroid-type instrument in which the base pressure can be set and small differences from this base pressure measured. Another type, of which a number of designs have been used and which has been termed a "contrabarometer", is patterned after the mercurial barometer but makes use of a second lighter liquid and of differences in tubing areas to increase the differences of vertical levels corresponding to small pressure changes. Both types are rather bulky in their present forms; but the first appears to have greater advantages, as to ease of handling and portability, for mine work.

In determining small pressure differences accurately by absolute-pressure methods it is necessary to realize that such instruments measure only the absolute static pressure. At any point there is some energy in the air in the form of velocity, but it is ordinarily so small that it is not considered in ordinary absolute-pressure measure-

[7] Williamson, J. N., Ritson, J. A. S., Cooke, W. E., and Statham, I. C. F., Air-Pressure Surveying in Mines: Trans. Inst. Min. Eng. (London), vol. 81, 1931, pp. 313–339.

ments. However, it may form a considerable proportion of small differences in total absolute pressures, and whether or not it is negligible for a particular condition should always be taken into consideration.

Where the points concerned in the measurements are at different elevations it is necessary to calculate the actual average density of the air over the intervening vertical height so that the difference due to the normal increase in the pressure of the atmosphere (0.0144 inch of water per foot for 0.075 air density) may be separated from the effect of air flow.

DIFFERENTIAL-PRESSURE MEASUREMENTS

Small pressure differences due to flow of air in mine openings usually are measured on a liquid gage by connecting both pressures to the gage and measuring the differential height of the liquid supported by the difference in pressure, which is a direct measure of the intensity of the pressure difference. Water is commonly used; or, if another liquid is used its gravity is determined, and all readings are commonly expressed in inches of water. Since the unit weight of water changes but very slightly with change of temperature no temperature is commonly specified, although for accurate work the weight of water at 60° F. is ordinarily used, which is closely equivalent to 5.2 pounds per square foot or 0.0361 pound per square inch.

All common pressure measurements in mine ventilation are differential-pressure measurements, but since the most common measurements are those made to ascertain the difference in pressure between the atmosphere the gage is in and some other point at a different pressure which is connected to the gage through tubing or pipe one leg of the gage is open to the atmosphere and such measurements are usually termed merely "pressure" measurements. The term "differential-pressure" measurements is therefore commonly reserved for those measurements in which both pressures differ from that at the gage position, and both are communicated to it through tubing.

U-TUBE WATER GAGE

The most common pressure-difference measuring device for mine-ventilation surveys, and the only type required under most conditions of routine work, is a simple glass U-tube partly filled with water. With the tube in a vertical position the difference in level in inches between the water levels in the two legs is measured against a scale, preferably graduated in inches and tenths. With steady pressures the difference in levels can be measured by estimation to the nearest one hundredth inch, which is close enough for the constantly changing pressure conditions in mines. Commercial types are made with an adjustable scale, so that the zero position of the scale can be adjusted to the no-pressure-difference position of the water, and the readings both ways from the zero are added to obtain the total difference. On the assumption that the distances above and below the zero position will be equal, which is only true for absolutely equal bores not found in ordinary glass tubing, the scale on some commercial types of plain U-tube gage is made double so that only one reading is required. This type of gage causes confusion and may be quite inaccurate.

For occasional measurements of the pressure difference between the two sides of a door or stopping, which are matters of interest in either naturally or mechanically ventilated mines, no great degree of accuracy is required, and the readings can be made on a simple home-made U-tube. A very simple type for rough work can be constructed by mounting a 5-inch U-tube, of one-fourth-inch O. D. glass tubing with the legs on about 1-inch centers, on one half of a piece of white cardboard on which a black-line scale of inches and tenths has been ruled. The tube can be secured to the card with rubber bands and protected by folding the other half of the card over and securing it with another rubber band. The scale should be protected from dirt by a piece of flexible, transparent material, held in place by mounting tissue of some sort. The water can be left in the gage during transportation if the two ends are connected by a piece of rubber tubing—one-eighth-inch I. D. tubing for a tight fit on one-fourth-inch O. D. glass tubing. For use, a short length of rubber tubing—preferably 2 to 3 feet of heavy-wall tubing to avoid excessive kinking and to permit some flexibility in position of the gage for reading—is attached to one leg of the U-tube. This is pushed through an opening to the other side of the door or stopping against which the tube and card are held rigidly and vertically while the difference of the two liquid levels is read. The flexible mounting makes it possible to bring one level to an even division of the scale. It is desirable to arrange the reading to come about at the height of the eye, and it is necessary to take the precaution to have both the open end of the U-tube and the open end of the rubber tubing in quiet air, which means making sure that neither is opposite a crack or hole where a high velocity of flow might exist.

For measuring fan pressures, however, a larger range and a more rigid mounting are desirable for better average accuracy. Commercial types, though cheap and good for fixed positions, usually are too heavy and bulky for rapid survey work. A light, easily carried gage may be made by mounting a 10-inch glass U-tube, similar to that just described, on a ¼- by 1½- by 11-inch block of nonwarping wood, with each leg wired to the wooden mounting near each end and at the middle so that the U-tube is movable vertically but cannot bow away from the block. The scale should be ruled accurately on heavy white paper after it has been smoothly glued to the block and is thoroughly dry. This gage can be protected most easily and carried in a leather case similar to that of the Bureau of Mines standard sling psychrometer, with the rubber tubing attached and hanging down outside the case. A short extra length of tubing should be carried where differential-pressure readings, such as velocity-pressure readings from a Pitot tube, are required.

MANOMETERS

Although all differential-pressure instruments are correctly termed "manometers" and a water gage is thus a vertical manometer, the term is quite generally applied to gages in which one or both legs of the U-tube are inclined to obtain more accurate readings of the difference in vertical level of the liquid in the two legs of the gage. These are seldom used for routine mine work but are often desirable and are used largely for experiment and research. Types with uniform bores in both tubes, requiring readings on both legs, are more suitable to

work where calibrations are infrequent than types in which the bore of one leg, which is vertical, is so large in comparison with the small bore of the other leg, which may be vertical but is usually inclined, that virtually all of the movement takes place in the small-bore leg and the total movement is read on this leg by means of a properly graduated scale. The slope of one or both legs may be adjustable, and other liquids than water may be used to increase the reading and to give better liquid-flow conditions in small-diameter glass tubing. Weeks [8] has fully described the construction of a portable, mine-type, uniform-bore, adjustable-slope gage which requires calibration only to check mechanical accuracy of construction. Many designs of non-uniform-bore instruments, with both fixed slope and range and adjustable slope and range, are commercially available. Because glass tubing does not have a uniform bore throughout they require calibration, and types in which the tubing is unsupported or rigidly fixed at both ends require frequent recalibration. With proper calibrations readings accurate within one or two thousandths of an inch of water may be obtained with these instruments, and accuracy within a hundredth of an inch is certain without calibration.

MICROMANOMETERS

Instruments arranged to make very accurate measurements of minute pressure differences commonly are referred to as micromanometers and involve micrometer arrangements and null methods of operation to insure accuracy. These are laboratory rather than field instruments and are used as primary standards for calibrating field instruments. The simpler types, such as the hook gages, are accurate to one one-thousandth inch of water; more complicated types, of which the Wahlen gage,[9] developed at the University of Illinois and largely used in Bureau of Mines ventilation investigations, is a good example, can be made accurate to one ten-thousandth inch of water; very special types with even greater sensitivity and accuracy can be made.

RECORDING PRESSURE GAGES

All the instruments so far discussed are indicating gages or manometers. Recording designs are not used to any extent in metal mines but are available in a variety of commercial types and undoubtedly could be used to advantage. The laws of most coal-mining States require recording gages for main-fan pressures, but there are no legal requirements of this type for metal mines and none are required. Most fans for both coal and metal mines are of the forward-curved-blade type and have such flat pressure characteristics that the record of fan pressures, particularly when taken from the inlet side of fans in the exhaust position, has little relation to fan performance or mine-airway conditions. Records of quantities of flow would be much more informative for any fan installation; but records of quantities of flow are more difficult to obtain than pressures, and the types of instruments available are few and relatively expensive and require the use of a calibration curve to interpret the record.

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[8] Weeks, W. S., Ventilation of Mines: New York, 1926, p. 15; A Precise Manometer for Ventilation Measurements: Min. and Met., vol. 4, 1923, pp. 19–22.

[9] Williard, A. C., Kratz, A. F., and Day, V. S., Investigations of Warm-Air Furnaces and Heating Systems: Univ. of Illinois Exp. Sta. Bull. 120, 1921, pp. 91–98.

PRESSURE OBSERVATIONS

Pressure observations often require transmittal of pressures through long lengths of tubing and must often be obtained in moving air currents. Any length of horizontal tubing suffices to transmit pressures accurately, but where the tubing extends to points at different elevations it is necessary to have the same air density within and without the tubing to balance the normal change in absolute pressure that accompanies differences in elevation, which means that in vertical or inclined openings the tubing should be suspended in the airway for which the measurements are being made and not in a warmer or colder airway.

Mine pressures are always fluctuating, and there is a time lag in communication of the fluctuations through the openings of pressure instruments and through tubing, which sometimes causes large fluctuations in gage pressures and for good results requires an average to be made of a number of readings. Where both pressures are communicated through long lengths of tubing the gage fluctuations can be largely minimized by equalizing the openings of the pressure-obtaining instruments and by adjusting the tubing lengths, by trial, to a ratio that gives minimum fluctuations.

To obtain correct static pressures in a moving air stream it is necessary to take the pressure lead from a connection to a small hole or holes in a surface parallel to the flow and free from projections or burrs, for which purpose the static-pressure member of the Pitot tube is convenient for survey work. For permanent installations larger holes, up to one fourth inch in diameter in side walls or in beveled-edge plates, are preferable because they are not so quickly clogged.

FAN-PERFORMANCE DATA

In mechanically ventilated mines there are usually three types of fan installations for which relative performance data are desirable: Main fans, booster fans, and auxiliary fan-tubing units. The rapidity of change of conditions of normal operation dictates the degree of accuracy desirable in routine records, which ranges from about 2 percent for main-fan installations to 5 percent for booster-fan installations and 10 percent for auxiliary fan-tubing installations.

The relative mechanical efficiency of fan operation—that is, the degree to which the operating condition approaches that for which the fan was designed—can be determined, usually within about 1 percent, from approximate measurements of pressures and quantities by reference to curves showing the manufacturer's test data, but the correlation of pressure measurements to rated catalog or test-curve pressures usually requires some analysis, as is discussed in detail in a later section on fan pressures and fan ratings. Usually this information on relative efficiency suffices for the efficient use of the fans available and for the selection of new fans.

Determinations of absolute mechanical efficiency are difficult to make in the field and are liable to be very inaccurate unless supervised by someone with a knowledge of the technique necessary for obtaining the measurements, for analyzing the results, and, if required, for applying the pressure corrections almost always involved in field tests. Unavoidable leakages are also a confusing factor in such tests, and

since all errors are cumulative the final results expressed as absolute efficiencies may contain large errors.

Fan-speed measurements may be made accurately and easily with ordinary revolution counters or Veeder counters, and instruments are available for determining speeds accurately by measurements requiring only a few seconds. Speed recorders are not always reliable and should invariably be checked for accurate work.

Field measurements of power are difficult to make accurately. Virtually all metal-mine fans are electrically driven, and even though the power input to the motor is determined accurately by the mine electrical department the motor efficiency for the exact condition of operation must be known and the drive losses estimated or determined separately before the power input to the fan is known.

In all field tests it should be remembered that fan ratings are based on a standard sea-level weight of air or air density, usually taken as 0.075 pound per cubic foot. Quantities of flow at the fan are independent of air density, but pressures and powers vary directly with it.

MAIN FANS

Quantity of flow should be determined by precise traversing methods wherever it can be accomplished most conveniently with due regard to turbulence and possible leakage. The discharge of surface exhaust fans, either with or without tapered (evasé) discharge connections, is usually so turbulent that measurements at this most conveniently accessible point are seldom reliable unless a method factor has been determined for the particular type of discharge; and these easily may vary from 0.7 to 1 for good types. If measurements are made at a distance from the fan, particularly if any great difference of elevation is involved, the air density should be determined both at the point of measurement and at the fan inlet and the measured quantity corrected on an equal-weight basis to the inlet air density. All surface-fan installations should be provided permanently with suitable arrangements for getting a good static-pressure reading inside the casing—near the inlet of an exhaust fan or near the discharge of a pressure fan—and one free from turbulence effects. A position as close to the fan as possible is desirable, but it is even more desirable that it be in a section where the flow sensibly parallels the walls of the duct. The difference in static pressure between the atmosphere and the inlet of an exhaust fan or the discharge of a pressure fan should be measured, as well as the area of the section at which it is obtained. This last measurement, as well as the density at the fan inlet, is required for conversion of the data to standard conditions for comparison with rated performance data and determination of relative efficiency of operation.

BOOSTER FANS

Quantities of flow usually are determined in the vicinity of the fan by approximate traversing methods. The fan discharges through a stopping which should be provided with an opening through which to pass the tubing of the pressure gage. The pressure on the stopping, close to both sides of which there is virtually quiet air, is measured and may be taken directly as the approximate differential static

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pressure produced by the fan. The latter should be based more correctly on the static pressures at the fan inlet and outlet, neither of which is quite the same, although usually within a tenth of an inch of water, as the pressure on the adjacent side of the stopping. Corrections for these differences, of which only that on the discharge is usually material, can be made by making separate determinations of the difference in static pressure between the inlet or discharge and the stopping by using the static member of a Pitot tube for obtaining the pressure in the moving air current.

AUXILIARY FAN-TUBING UNITS

For an auxiliary fan-tubing unit it is desirable, in a survey, to determine the approximate quantities both at the end of the line and close to the fan to keep informed of the approximate leakage condition of the line of tubing or pipe which carries air to or from the working place. It is suggested that, for rigid tubing, the end reading be taken by holding an anemometer, mounted on a short shaft, at the center of the tubing end and applying a method factor of 0.85 for a discharge end or of 0.7 for an inlet end. With canvas pipe the discharge-end condition is such that only the most approximate measurements can be made, and it is suggested that the reading be taken by holding the anemometer by hand at the center of the discharge and applying a method factor of 0.8.

Both the pressure and quantity in the tubing near the fan can be ascertained most conveniently with a small Pitot tube inserted through a small hole punched in the tubing. With the static connection only attached to the gage the static pressure in the tubing is first measured. Then both connections are attached to the gage, and a rough traverse is made on one diameter of the tube with the traverse points gaged by a mental picture only of the correct traverse positions (see fig. 3, *B*) accompanied by a rough mental average of the observed velocity pressures for later computation, in connection with a calculation of the actual area based on the measured circumference, of the average velocity and quantity. The method is admittedly very rough but sufficient for accuracy within 10 percent, which is all that conditions ordinarily justify. Greater accuracy may be obtained by the use of a sensitive gage combined with more attention to the details of procedure, such as the use of guideframes for locating traverse points on two perpendicular diameters.

AIR DENSITY

The weight of 1 cubic foot of a mixture of air and water vapor, commonly designated as "air density", may be found accurately enough for ventilating problems from the formula

$$d=\frac{1.325}{459+t}(B-0.378f), \tag{4}$$

where d is the weight of 1 cubic foot, pounds;
t is the dry-bulb temperature, °F.;
B is the barometric pressure, inches of mercury; and
f is the vapor pressure at the dew point, inches of mercury.

With the same units the weight of the dry air (d_a), in pounds, in 1 cubic foot of a mixture of air and water vapor may be found from

$$d_a = \frac{1.325}{459+t}(B-f), \tag{5}$$

and the weight of the water vapor (d_w), in pounds, in 1 cubic foot approximately from

$$d_w = \frac{1.325}{459+t}\,0.622\,f = \frac{0.824}{459+t}\,f. \tag{6}$$

Slightly different values of the constants in these equations and other formulas are given by various authorities; and tables of densities and steam data are also available. The standard weight of air for mine-ventilation work is that of dry air at 70° at sea level and is 0.07495 pound per cubic foot; 0.075 is sufficiently accurate for ventilating purposes. A chart [10] for the approximate determination of air densities from (4) and their ratio to standard (0.075) air density is given in figure 7, in which the method of procedure is indicated by the solution of the following example, with arrows to show the direction of procedure; with a dry-bulb temperature of 93.5°, a wet-bulb temperature of 84°, and a barometric pressure of 29.25 inches of mercury the weight of moist air is 0.0692 pound per cubic foot and its ratio to standard air 0.923.

For some problems the weight of the air only in 1 cubic foot of the mixture of air and vapor, from (5), is desired. This may be obtained from figure 8, which is similar in many respects to figure 7 and in which the method of procedure is indicated by the same example, which shows that the weight of the dry air is 0.0676 pound per cubic foot and that 14.79 cubic feet of the mixture contain 1 pound of dry air.

PSYCHROMETRIC TABLES AND CHARTS

Tables and charts for determining the physical properties of moist air from wet- and dry-bulb temperatures and barometric pressures are available. Vapor pressures at saturation (which are independent of atmospheric pressure) and relative humidities and dew-point temperatures for the range of 23 to 30 inches of mercury, atmospheric pressure, may be determined from wet- and dry-bulb temperatures by means of the psychrometric tables [11] of the United States Weather Bureau. Data for higher pressures, which may be encountered in deep mines, have been published [12] abroad. Although these data, as well as absolute humidities (weight of water present in air as vapor) and total heat contents, may be determined approximately, according to the scale of presentation, from psychrometric charts, such as the Carrier, Hill, and Bulkeley charts, the charts available are limited to data for an atmospheric pressure of 29.92 inches of mercury or average sea-level pressure.

[10] Based on American Blower Co. Data Sheet 1425, April 1927.

[11] Marvin, C. F., Psychrometric Tables for Obtaining the Vapor Pressure, Relative Humidity, and Temperature of the Dew-Point: U. S. Department of Agriculture, Weather Bureau No. 235, 1915, 87 pp.

[12] Jones, J. S., Hygrometry for Deep Mines: Trans. Inst. Min. and Met., vol. 34, pt. 1, 1924–25, pp. 3–67. Jeppe, C. W. B., Psychrometric Tables, with Special Reference to Deep Mining Ventilation: Jour. Chem. Met. and Min. Soc. of South Africa, vol. 31, pp. 88–118, and supplementary 30-page pamphlet, October 1930.

Dew-point temperatures are required for air-density calculations as well as for the direct determination of absolute and relative humidities and are therefore involved in virtually all psychrometric determinations. The relation of dew-point temperature to wet-bulb depression is quite involved and is a function of the barometric pres-

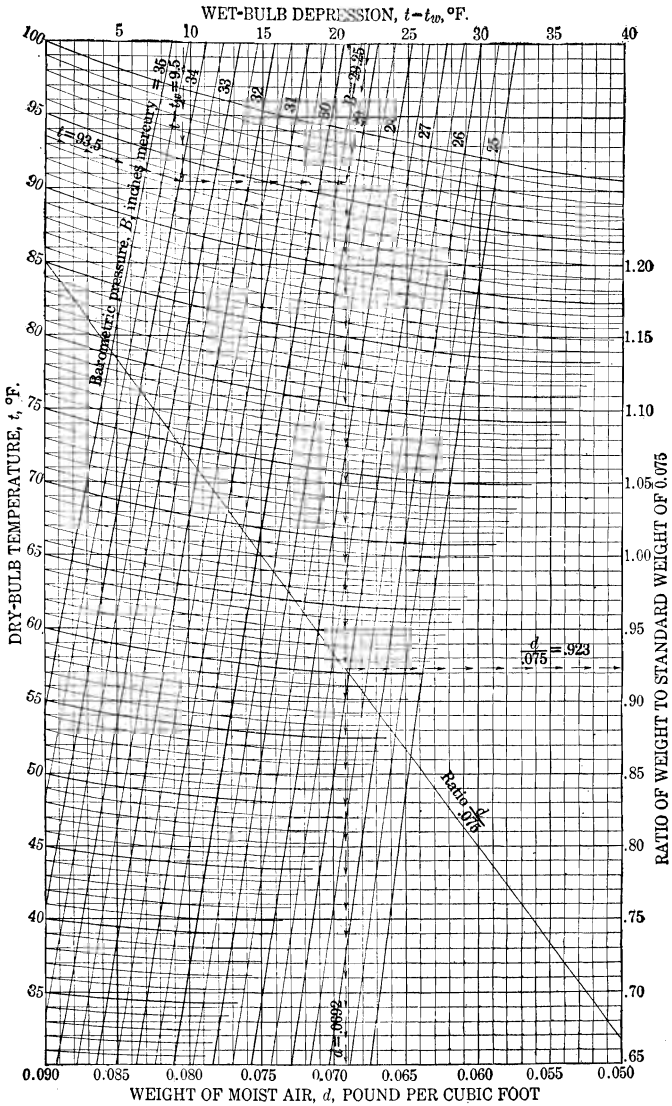


FIGURE 7.—Chart for determining weight of moist air and ratio to standard weight.

sure and temperature conditions. It must therefore be determined from tables or charts. In figure 9 the Weather Bureau data are plotted in the form of a chart, with the values extrapolated to a certain extent to cover a larger range of barometric pressures. It serves for the approximate determination of vapor pressures, dew-

point temperatures, and relative humidities from wet- and dry-bulb temperatures and barometric pressures. The method of using the chart is indicated by the arrows which trace the solution of the following example: With a wet-bulb temperature of 84°, a dry-bulb temperature of 93.5°, and a barometric pressure of 29.25 inches the

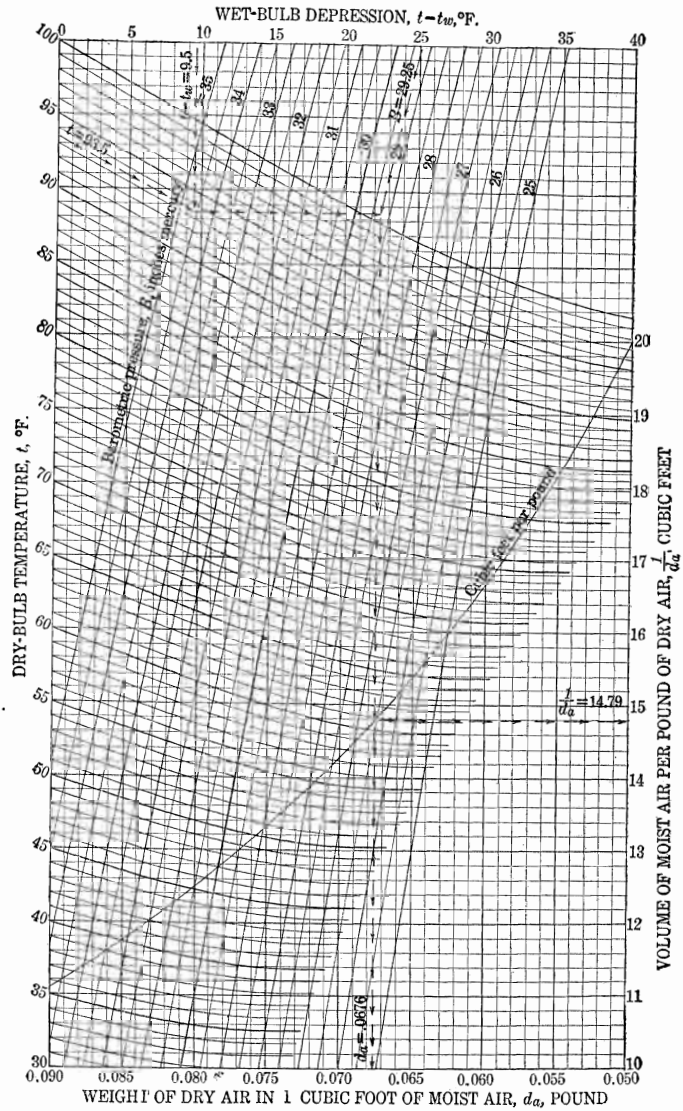


FIGURE 8.—Chart for determining weight of dry air in moist air and volume per pound.

pressure of the vapor is 1.058 inches, the dew-point temperature 81.1°, and the relative humidity 67.4 percent.

Absolute humidities and total heats, when expressed on a pounds-of-dry-air basis, vary with the absolute pressure, but at constant pressure depend only on the dew-point and wet-bulb temperatures,

respectively, since the absolute humidity for any wet- and dry-bulb temperature condition is the same as for saturation at the corresponding dew-point temperature, and the total heat for any wet- and dry-bulb temperature condition is the same as for saturation at the wet-bulb temperature.

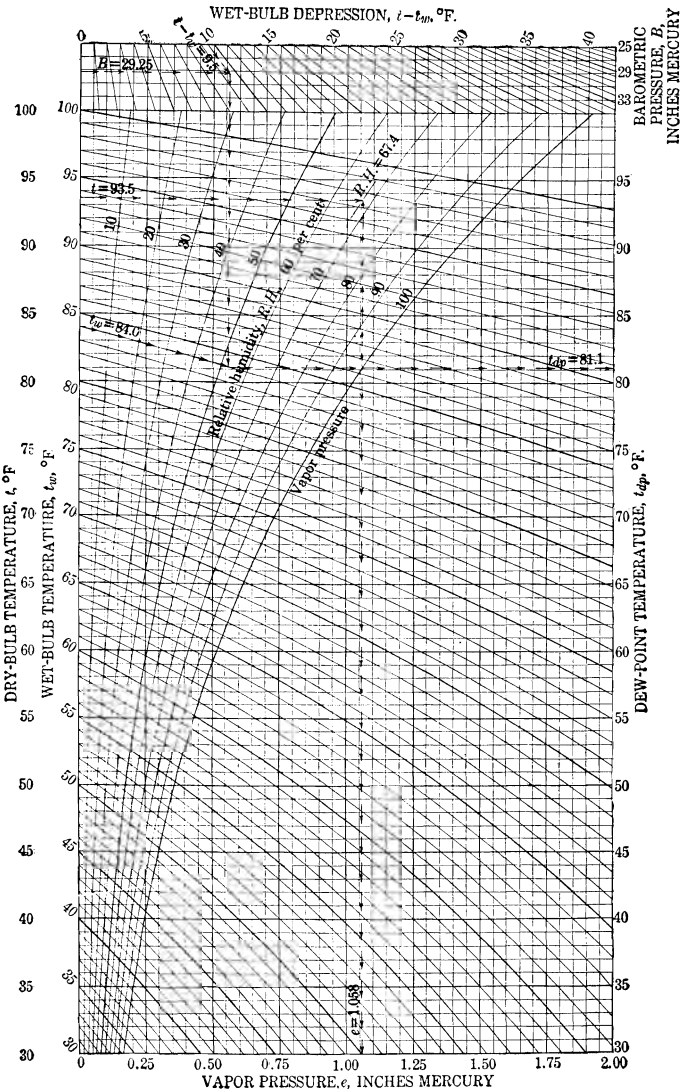


FIGURE 9.—Chart for determining vapor pressure, dew-point temperature, and relative humidity of moist air.

With the wet-bulb and dew-point temperatures known, a simple chart, figure 10, suffices for determining total heat and vapor contents of unit quantities, since these involve direct relations. The method of procedure is shown by the continued solution of the same example traced by arrows, which shows that there are 47.5 B. t. u. (above ° F.)

and 0.0237 pound of water vapor per pound of dry air in the moist air.

For most mine air-flow conditions (almost saturated air) the pounds of the mixture of air and water vapor, found by multiplying the quantity in cubic feet by the weight per cubic foot, or air density, from (4) or figure 7, may be used for the pounds of air in the mixture with small error. The actual pounds of air in the mixture is the number

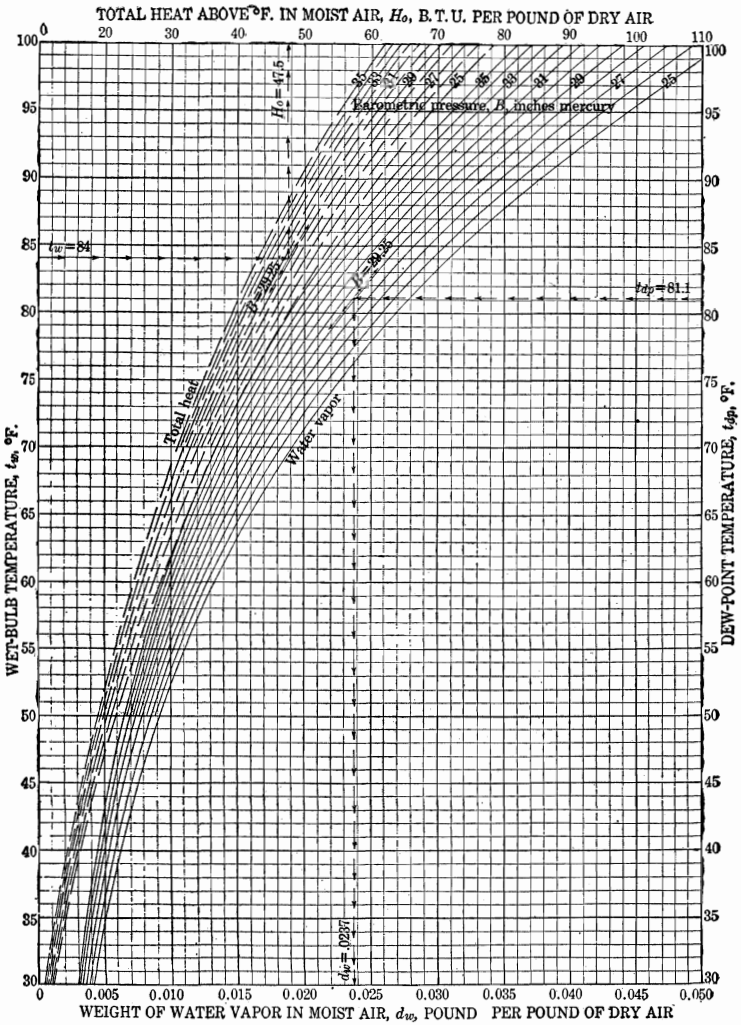


FIGURE 10.—Chart for determining water vapor and total heat in moist air.

of cubic feet multiplied by the weight of dry air per cubic foot of the mixture, from (5) or figure 9.

COOLING-POWER OBSERVATIONS

The main interest in the ventilation of metal mines has always been devoted to those mines where the ventilation requirements are

determined by lack of enough cooling power in the air rather than its chemical composition. Until efficiency of working is perceptibly reduced through reduced cooling power little attention is ordinarily paid to ventilation, and few metal mines would be mechanically ventilated were it not for the effect of higher rock temperatures in reducing the cooling power of the air and the efficiency of labor. By improving the cooling power of the air comfort is improved, and efficiency follows suit.

The cooling power of the air, or its capacity for cooling the body, depends on its dry-bulb temperature, wet-bulb temperature, and rate of motion and is the major factor in determining the rate at which the waste heat of the body may be transferred to its surroundings, although the exact condition of the body also is important in determining the rate. If this rate is too slow or too fast sensations of discomfort are experienced which directly affect efficiency of labor.

KATA-THERMOMETER

The kata-thermometer is a large-bulb alcohol thermometer devised by Dr. Leonard Hill, of England, to measure its own rate of cooling and thus serve as an index of the relative rate of cooling of the body under various conditions. The instrument may be used with the alcohol bulb bare, in which case "dry-kata" cooling powers are obtained; or a wet cloth sack may be fitted over the bulb, and "wet-kata" cooling powers are then obtained. The stem of the thermometer is etched at the 95° and 100° points; and the time in seconds required for the instrument to cool through this temperature range, close to body temperature, is divided into a factor, etched on the stem of the instrument, to obtain its rate of cooling in millicalories per square centimeter per second. The use of these metric units has become standard for this instrument, but they can be converted to British thermal units per square foot per hour by multiplying by 13.27. Electrical indicating and recording kata-thermometers may also be obtained.

The use of the kata-thermometer or similar instruments [13] that may be developed for actual determinations of relative cooling powers of the air and surroundings on a dry or a wet body is about the only way in which actual cooling powers can be determined under mine conditions because air motion is almost always an important factor in actual cooling rates and is often of such an eddying and turbulent nature that no approximation to the equivalent effect of a uniform velocity, such as might exist in a pipe or wind tunnel, can be made.

The actual cooling of the body, in either a dry or a wet state, is entirely different in rate from that of the kata-thermometer—roughly about 5 to 1—but the instrument serves an excellent purpose in providing relative measurements for all except possibly the most unusual conditions.

Where there is no cooling power, but rather a warming power, the relative warming power may also be determined by the kata-thermometer by measuring the time required for the instrument to heat through the same temperature range. The frail nature of the present instrument and the time consumed by observations have hindered its

[13] Weeks, W. S., A New Instrument for Measuring Cooling Power; the Coolometer: Jour. Ind. Hyg., vol. 13, September 1931, pp. 261–265.

general adoption in mine work, but its use is gradually extending in the hot mines and, to a very much smaller extent, in other mines.

EFFECTIVE-TEMPERATURE INDEX

Studies on human comfort at the research laboratory of the American Society of Heating and Ventilating Engineers and the Bureau of Mines at Pittsburgh have determined equivalent conditions of dry- and wet-bulb air temperatures and air motion, with surrounding walls at the dry-bulb air temperature, for which persons feel equally com-

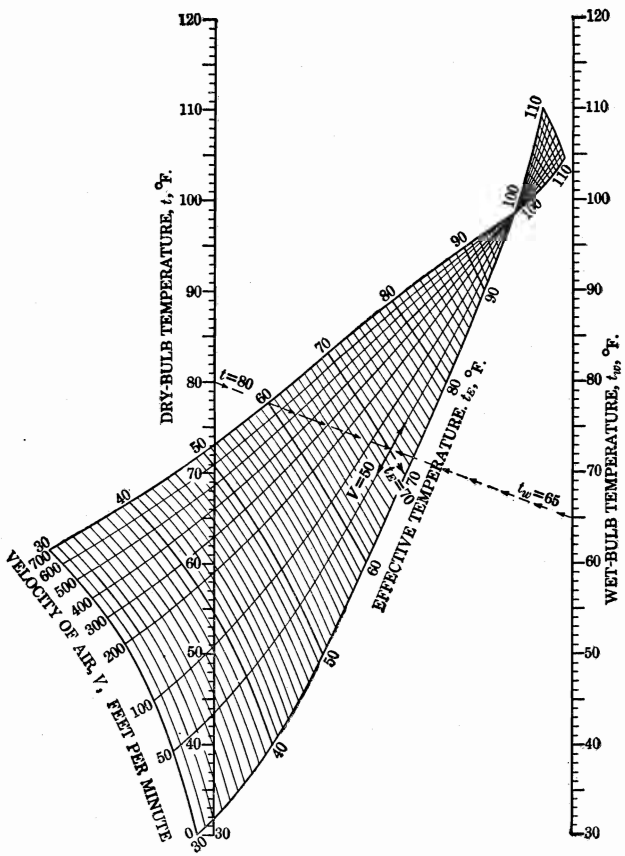


FIGURE 11.—Effective-temperature chart for men at rest and stripped to the waist.

fortable. The temperature of still, saturated air that produced the same sensation of comfort is used as an index for other combinations of air temperatures and air motion and termed the "effective temperature." The results of separate series have depended mainly upon the amount or lack of clothing worn; several charts of the possible combinations are available. For hot mines the results of tests made with the subjects stripped to the waist and at rest are most applicable to underground conditions and are reproduced, in the chart form devised by C. P. Yaglou, as figure 11. The method of

using the chart is shown by an example, with arrows to indicate the direction of procedure. In the example solved the effective temperature (t_E), corresponding to a wet-bulb temperature (t_w) of 65°, a dry-bulb temperature (t) of 80°, and a velocity (V) of 50 f. p. m., is 70°. The results of tests on men acclimated to high temperatures and working moderately hard while stripped to the waist and covered with sweat would be valuable for comparative mine purposes but are not available.

The effective-temperature index offers a method of comparing variations in air conditions for average underground conditions simply and accurately when the three variables of dry-bulb temperature, wet-bulb temperature, and rate of air motion are known, since the walls may be assumed to be at the same temperature as the dry-bulb air temperature. But the rate of air motion is often indeterminable, under mine conditions, just where it is desirable to institute comparisons, and other than the test conditions of radiation of heat and condition of the body may often be met. For these reasons the kata-thermometer type of instrument cannot be equaled for direct determinations. The relations of effective-temperature scales and kata cooling powers are complex on account of the number of variables involved and have not been satisfactorily solved, although much work has been done on this subject.

SURVEY RECORDS

Records of mine-ventilation surveys should be systematic and complete and should be preserved for future reference, since much of the detail is such that it cannot be presented in sketch or map form. However, as much of the data as possible should be entered on prints of the mine maps or on special ventilation maps or diagrams in order that it may be used to best advantage. In many cases actual relations are derived only by a study of mapped data. Such a map or sketch is required primarily to show the distribution of air currents and the positions of devices for producing or controlling the distribution. Special sketch maps, on which the data are entered by means of a definite series of symbols, are generally desirable. Since distribution is the important factor in this form of record discussion of this phase of the subject is deferred to a later section on distribution and control of air currents.

FLOW OF AIR IN MINE OPENINGS

Air flowing through ducts, such as mine openings, encounters resistance due to interference of the individual air streams with each other caused by the wall surfaces. The result is that part of the kinetic (velocity) energy of the flowing stream is converted to heat; and constant renewal of the kinetic energy from the total energy of the flow is required to maintain it. The flow therefore acts to cause a continuous decrease in the total energy of the air in the direction of flow; and the flow automatically adjusts itself as to quantity so that the total energy used on any circuit of connected openings is equivalent to the total energy generated on that circuit.

The sea of air that surrounds the world has weight, and the pressure exerted by this weight increases with depth at a gradually increasing

rate. The total energy in the air in any air column therefore increases with vertical depth; but the increases in separate columns are not necessarily the same, and the energy representing the differences in these natural increases in energy content is used up by flow through openings connecting the columns so as to maintain a condition of equilibrium. This naturally generated energy available for producing flow is called "natural draft." Energy may also be added to a circuit at any point mechanically, as by a fan, and is then termed "mechanical draft."

PRESSURE FORMS AND DESIGNATIONS

These energy losses and energy gains involved in air flow are evaluated and measured in terms of pressure energy, even though the actual changes in pressure energy do not correctly evaluate the total changes in energy content. In a flowing stream part of the energy is in the form of velocity or kinetic energy, but this kinetic energy can be measured as a pressure exerted only in the direction of flow, as distinguished from a true pressure, which is exerted in all directions. The actual energy content of the air at any point in a flowing stream is therefore represented correctly only by the sum of the actual pressure and the pressure corresponding to the velocity of flow; this sum is termed the "absolute total pressure." The true absolute pressure is termed the "absolute static pressure" to identify it similarly. Air flow depends on differences of absolute total pressures rather than their actual values, and pressure measurements are measurements of differences in absolute pressures. In quiet air, such as the atmosphere on the surface, the only pressure exerted is the absolute static pressure. In a flowing stream the pressure exerted on the open end of a tube pointed against and parallel to the flow is the absolute total pressure, whereas the pressure exerted on a hole in a smooth surface parallel to the flow is the absolute static pressure, which in practice is somewhat difficult to obtain entirely free from velocity effects. The difference between the absolute total pressure and absolute static pressure at any point is termed its "velocity" pressure.

Although pressure differences must be measured between points the differences in average pressures across sections at right angles to the flowing stream are chiefly involved in this work. It so happens that the absolute static pressure is virtually constant across such a section where the flow is sensibly parallel to the walls; but the velocity across such a section is not uniform, and therefore the velocity pressures and the absolute total pressures across the section are not uniform. Absolute static-pressure differences are therefore the basic pressure measurements because easily ascertained. Average velocity pressures and average absolute total-pressure differences can be obtained by traversing such sections, but in practice such traverses are made only as a method of determining average velocities from velocity pressures. When the quantity of flow is known the average velocity pressure for any section may be calculated accurately enough from the average velocity based on the area of the section; and in practice these calculated average velocity pressures are used to calculate average absolute total-pressure differences from absolute static-pressure differences.

The pressure difference commonly measured is that between the absolute static pressure at a section in a flowing stream in a duct and the absolute static pressure of the atmosphere outside the duct or that between the absolute static pressures of two separated quiet atmospheres. The result is commonly referred to as the "water-gage" pressure, but this is an indefinite expression because various pressure differences may be measured on a water gage. The fact that a difference of absolute pressures is involved is, however, ignored, and the pressure difference is designated as the "static" pressure of the air stream in the first case and as the "static-pressure difference" between the atmospheres in the second. Where the difference in absolute static pressure between different sections in flowing streams is desired the difference may be determined directly, or it may be determined indirectly from the separate differences from the pressure of the atmosphere where it is available; and the same designation of "static-pressure difference" is commonly used. The corresponding terms for absolute total-pressure differences, usually calculated from the foregoing and calculated velocity pressures, are "total pressure" and "total-pressure differences." Unless stated to the contrary, average total pressures over a section, rather than pressures at points, are implied.

Actual absolute pressures in a flowing stream are not only changed by flow but also change with change in elevation due to the increase in absolute static pressure that accompanies increase in vertical depth; but these changes are automatically allowed for, in the methods of measuring pressure differences, by equivalent absolute static-pressure differences in the tubing which connects the point to the gage position and need not be considered as affecting the flow except through the agency of natural draft or unbalanced absolute-pressure changes due to differences in air density.

PRESSURE CHANGES

The weight flow through an air circuit, in pounds per minute, is constant; and if the air density remains constant the quantity of flow in cubic feet per minute also remains constant throughout the circuit. In a mine-ventilating circuit differences in elevations and temperatures produce small changes in density; and the quantity of flow is only approximately constant throughout the circuit, although the same number of pounds of air enter the mine as leave it, except for actual additions underground, such as compressed air and strata gases. With the quantity constant every change in area of cross-section requires a corresponding change in average velocity and in average velocity pressure. Changes in total pressure are affected only by energy losses and energy gains, so, assuming no change in total energy, changes in area must be compensated by the interchange of static and velocity pressure; this is actually the case, although some energy is lost at each change of area. The velocity pressure, for constant quantity, changes only with the area of cross-section, but the static pressure is influenced by changes in both area and energy.

In an airway of uniform area of cross-section the velocity distribution remains practically uniform and the average velocity pressure likewise, so that the static-pressure changes equal the total-pressure changes. Because static pressure is the basic pressure measurement

and because airways of uniform area are the rule rather than the exception in commercial air-flow installations, the whole subject of energy losses and energy gains involved in air flow usually will be found to be presented on an artificially simple basis of static-pressure changes rather than of total-pressure changes, which are directly related to energy changes. Velocity pressures are not ignored but are considered by means of various separate conventions determined by the particular conditions, so that the final results are the same to those acquainted with the conventions; however, those not acquainted with the particular conventions in use are much confused. Since the energy changes can be considered correctly on the basis of total-pressure changes, this method will be followed throughout this bulletin. Wherever the word "pressure" is used without qualification total pressure is implied; and static-pressure changes are considered only in their true significance of pressure changes brought about by changes of both energy and area occupied by the flow.

Although the change in static pressure in mine openings usually is in the same direction and approximately of the same magnitude as the change in total pressure, this is not always true for high-velocity flow through variable areas of opening, as at mine fans. In fact, under certain conditions of enlargement of area the air may be found to be flowing from a lower to a higher absolute static-pressure area. Pressure losses, on the basis designated above, are always decreases of absolute total pressure in the direction of flow which are caused by the flow; and pressure gains always represent increases of absolute total pressure generated in the direction of flow. In both cases the natural changes in absolute total pressures due to differences in elevation are considered to be balanced. The effect of lack of balance in these changes is considered as a generated pressure—natural draft.

Positive and negative pressures.—It was necessary to qualify the foregoing statements with the word "absolute" because pressure differences, which are always positive absolute-pressure differences, are virtually always considered with reference to a fixed absolute pressure, that of the atmosphere. In this case both total and static pressure may be positive or negative, that is, either greater or less than the pressure of the adjacent atmosphere; and although the amount of the change in absolute total pressure and the direction of the change are not affected by the position in the circuit at which the pressure-generating source operates, the actual values of the pressures relative to a fixed pressure, such as atmospheric pressure, are controlled by the position of the pressure-generating source in the circuit, which is open to the atmosphere at both ends. Pressure may be generated at the entrance, at an intermediate point, or at the exit. For a fan these would be designated as the "blower", "booster", and "exhaust" positions, respectively.

A graph of the pressure changes in the same simple airway circuit with three positions of the one pressure-generating source is shown on both an absolute and a relative basis in figure 12. It will be noted that the pressure losses have been assumed to be uniform throughout, that absolute total pressure always decreases in the direction of flow, and that, although the pressure-generating source produces the same absolute total-pressure gain in the direction of flow at each position, the relative magnitude of the total pressure and of the static pressure at any section with reference to the atmospheric pressure is determined

by the position of the pressure-generating source with reference to the system as a whole. The total pressure is always zero at the entrance to the system and positive and equal to the velocity pressure at discharge to the atmosphere; the static pressure is always zero at a free discharge to the atmosphere but negative and equal to the velocity pressure at the entrance to the system. Although pressures are considered and plotted ordinarily on this relative basis with respect to atmospheric pressure, as shown in the upper parts of the graphs, because they are so measured in actual practice for most conditions of air flow, it must be kept in mind that pressures so measured are concerned merely with pressure changes with respect to atmospheric pressure, whereas actual pressure losses and pressure gains are expressed by total-pressure differences that have an absolute-pressure origin. This conception of pressure relations is particularly necessary in considering the pressure changes at an underground fan, where there is no fixed atmosphere available as a constant-pressure base.

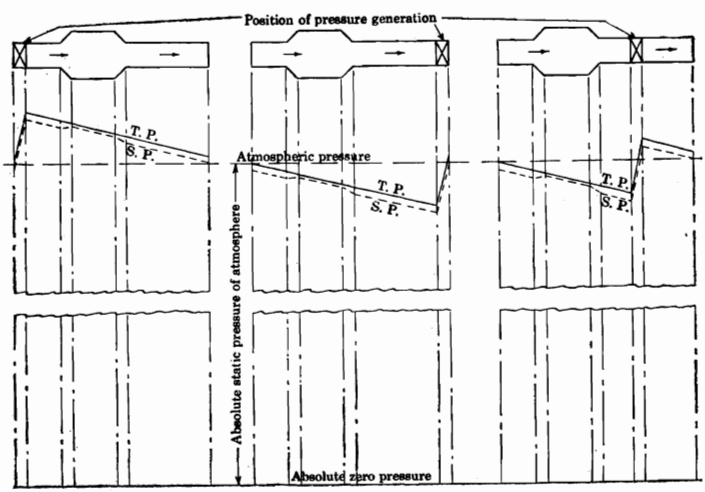


FIGURE 12.—Graphs of absolute and relative pressure changes in the same uniform-resistance system of airways for three positions of the same pressure-generating source.

In considering the relation of pressure losses to the total-pressure differences from the atmosphere, it should be noted that this difference decreases in the direction of flow on the pressure side but increases in the direction of flow on the exhaust side of a pressure-generating source but that both sides must be considered to evaluate the pressure generated. This necessity for considering pressure changes on the relative basis of their difference from atmospheric pressure results in some confusion but is sanctioned by universal use.

PRESSURE LOSSES

Precise evaluation of the pressure losses accompanying mine-air flow requires knowledge of the general theories of fluid flow, and for purposes of research and experiment it is desirable that all available data be correlated by means of a complete theory. But for practical application a more simple presentation suffices, restricted to average conditions of mine-air flow, since the pressure losses required to be

calculated occur in comparatively rough airways for flows of moderate to high velocities where empirical factors cannot be applied rigidly. The main argument revolves about the question of whether the pressure losses can be considered to vary as the square of the average velocity of flow. Actually, this condition does not hold exactly for all pressure losses in mine airways, particularly for low rates of pressure loss; formulas based on this condition, however, are precise enough for practical application of empirical data to mine ventilation, and the normally small variations that are known to occur, some of which are discussed in a later section, can be handled mathematically where desired, more easily by small variations in estimated empirical factors than by introducing fractional exponents in basic formulas.

The basic formulas for pressure losses, thus simplified, still appear to be the bugaboo in teaching mine ventilation, and many ventilation examinations degenerate into mathematical exercises rather than a test of ventilation knowledge. As a reaction to this condition continued attempts are made to displace basic formulas with still more simplified formulas which, although serving as mathematical aids in certain problems, cannot logically replace the basic formulas for solving pressure-loss problems.

Pressure losses may be divided into two separate groups according to their mode of origin: "Friction" pressure losses caused by the drag of the walls on the air streams, which depend primarily on the condition of roughness of the individual wall surfaces, and "shock"[14] pressure losses caused by changes in the area occupied by the air streams, which depend primarily on the relative positions of the individual wall surfaces with respect to the lines of flow.

All formulas for both friction and shock pressure losses include an empirical factor whose value varies with conditions and is determined only by actual experiments. These empirical factors are the basic data for air-flow calculations; unless approximately correct factors are used the calculations are worthless. Yet in spite of the fundamental importance of such data, one looks for them in vain in most textbooks on mine ventilation. Methods of using these data are given in great detail, but few are available. Since the results of Murgue's experiments were published[15] in 1893, a considerable mass of information has been made available both on factors for calculating friction pressure losses and on factors for calculating shock pressure losses. However, there has always been a practical preference for a constant factor, similar to Atkinson's, that could be used for a mine as a whole, regardless of the fact that no such single factor can be used with any pretense to accuracy even where the airways involved are roughly similar in general type. Attempts to make practical use of such single or average factors for a mine as a whole have led to such large discrepancies between theoretical calculations and actual results that the practical application of the data available has been very limited until recent years; as a matter of fact, rule-of-thumb methods still largely govern the field of mine ventilation.

Friction and shock pressure losses apparently take place concurrently in almost all types of mine airways and can be arbitrarily

[14] The author has selected the term "shock" to designate this group of pressure losses, for which there is no commonly accepted term, after carefully considering such terms as "miscellaneous", "velocity", "turbulence", and "impact"

[15] Murgue, D., Experimental Investigations on the Loss of Head of Air Currents in Underground Workings: Trans. Am. Inst. Min. Eng., vol. 23, August 1893, pp. 63–112.

segregated only for the purpose of calculation. Segregation is required because the two types of pressure losses obey different laws. Present practice is to calculate the major shock losses separately but to include all the minor shock losses in the friction-loss calculation. In the discussion presented here shock losses are segregated from friction losses as far as seems feasible in practice, but even so some minor shock losses are involved in determining the range of pressure losses considered as friction losses. The pressure losses in straight airways, of as uniform cross-sectional area as is normal for this particular type, are considered as strictly friction pressure losses and are so calculated. Any departure from this condition of straight airway of approximately uniform area is considered to involve shock losses which may be calculated approximately as friction losses by means of certain allowances but which should preferably be calculated separately as shock losses. The relation of calculated pressure losses to actual pressure losses in mine airways is only a rough approximation for two reasons: In the case of friction losses there are well-established empirical factors, but it is difficult for the individual to visualize the exact conditions of application; and in the case of shock losses, although the conditions of application are easily visualized, there are no well-established empirical factors for many mine-airway conditions. Moreover, for certain types of mine airways the average cross-sectional area of the airway, which has a preponderating influence on pressure losses, is not accurately determinable.

FRICTION PRESSURE LOSSES

The friction pressure losses in straight sections of approximately uniform area comprise the major part of the pressure losses in almost any type of air-flow system. In mechanically ventilated metal mines the friction losses in the main airways often account for 70 to 90 percent of the total pressure loss sustained in the system, although the figure may be much lower for a few mines. They are therefore of considerably greater practical importance than the shock losses as far as metal mines are concerned.

FRICTION-LOSS FORMULAS

The basic law for the turbulent flow of a fluid through a straight duct of uniform cross-section, assuming that the pressure loss varies as the square of the velocity, is that first developed for the flow of water but later found applicable in general to fluids, either liquid or gas:

$$h_f=\frac{flv^2}{2gr}, \tag{7}$$

where h_f is the decrease of pressure, head in feet of fluid flowing;

f is an empirical factor based on experiment and depending chiefly on the roughness of the wall surfaces and the system of units used in the formula;

l is length of duct, feet;

v is average velocity, feet per second;

r is the hydraulic or mean radius, a term used in hydraulics for designating the result of dividing the area of cross-section by the perimeter in contact with the flowing fluid; and

g is the acceleration due to gravity, a figure that varies slightly from place to place but is usually taken as constant at 32.2.

For mine ventilation this formula usually is used in the form:

$$p=\frac{kSV^2}{A}, \tag{8}$$

where p is decrease in average total pressure, or total pressure loss, pounds per square foot;
k is an empirical factor similar to f and termed "friction factor";
S is the rubbing surface, or square feet of wall surface, equal to the perimeter times the length;
V is velocity, feet per minute; and
A is cross-sectional area, square feet.

This formula contains no density term. The latter is included in friction factor k, which thus actually depends on the air density considered standard, which is close to 0.075 pound per cubic foot for most published values, a value that is assumed as standard throughout this bulletin. For horizontal ducts or airways at or near sea level differences from standard density usually are negligible compared with the uncertainty regarding the exact value of the friction factor, but in metal-mine airways they usually cannot be disregarded; it is therefore desirable to retain the density factor. It is also desirable to express pressures in the same unit in which they are measured, that is, in inches of water. However, the values of k as used in this basic formula should not be changed, since many engineers in English-speaking countries have these values firmly fixed in mind. A more convenient form of the basic formula for mine ventilation is:

$$H_F=\frac{kPLV^2}{A\times 5.2}\times\frac{d}{0.075}, \tag{9}$$

where H_F is total pressure loss due to friction, inches of water, and equals $p\times 5.2$;
k is the friction factor for a density of 0.075 pound per cubic foot;
P is perimeter, feet;
L is length, feet;
V is velocity, feet per minute; and
d is weight of air, pounds per cubic foot.

Since mine-ventilation engineers usually are concerned with quantities of flow in cubic feet per minute rather than velocities of flow and since velocity is equal to quantity divided by area, the basic formula may also be written:

$$H_F=\frac{kPLq^2}{5.2A^3}\times\frac{d}{0.075}, \tag{10}$$

where q=quantity of flow, cubic feet per minute.

Other forms of friction formulas are in use in other divisions of the field of fluid flow, as well as other units for the various factors, as dictated by convenience or utility. A factor similar to k appears in all these formulas; however, the numerical values differ widely, and full knowledge of both formulas and units used is required to transform the friction factors derived for use in one equation to equivalent values for use in a different form of the equation, unless the transformations have been made, as in the present case, without disturbing the numerical values of the factor. Any constants developing during the transformation from the basic formula for fluid flow customarily are included in the friction factor, since the latter has no significance except as a factor; such a wide variety of values of friction factors for similar conditions has resulted that casual comparisons of experimental

results are impossible. Considerable care is therefore necessary to insure that the values used correspond to the formula and exact units used in their determination.

LAWS OF PROPORTION

The last two forms of the basic formula presented contain in a convenient form most of the laws of proportion required for ordinary ventilation problems; with all remaining factors constant, the friction pressure loss varies directly as the friction factor, perimeter, length, density, square of the velocity, and square of the quantity and inversely as the area for constant-velocity conditions or as the cube of the area for constant-quantity conditions.

However, it is seldom necessary in ventilation problems to consider a condition where either perimeter or area remains constant while the other changes, as this implies a concurrent change of shape. The normal condition is the consideration of similar shapes, in which case a change of area is accompanied by a change in the perimeter. Although the two basic forms so far given are suitable for calculations they do not include directly the laws of proportion as to dimensions and area for similar shapes of cross-section.

For similar shapes the formulas show that the pressure loss varies directly as $\frac{P}{A}$ on a constant-velocity basis and directly as $\frac{P}{A^3}$ on a constant-quantity basis. By expressing both P and A in terms of a common dimension, O, it will be found that

$$\frac{P}{A}=\frac{\text{constant}}{O}=\frac{\text{constant}}{\sqrt{A}} \qquad \text{and that}$$

$$\frac{P}{A^3}=\frac{\text{constant}}{O^5}=\frac{\text{constant}}{A^{5/2}}.$$

The foregoing constants are not the same in each case and differ for each different shape of section, but constants do not affect the proportions; therefore the pressure losses for similar shapes vary inversely as the side or square root of the area on a constant-velocity basis and inversely as the fifth power of the side or five-halves power of the area on a constant-quantity basis.

Laws of proportion, often termed "ventilation laws", can be deduced for any term of the basic equation by transforming the equation so that the desired term stands alone on the left; but the result, except in particular instances, is more of a mathematical exercise than an aid to the understanding of ventilation principles and is likely to lead to confusion. It is better, therefore, to memorize the basic forms and to make the desired transformation only when required.

FRICTION FACTORS FOR MINE AIRWAYS

During 1923 and 1924 the author directed a large number of carefully conducted experimental determinations of friction factors for mine airways—a few in coal mines and a relatively large number in the metal mines of Butte in cooperation with the Anaconda Copper Mining Co. Concurrently with the Butte epxeriments, the coal-mine

experiments were continued in the Bureau's experimental coal mine near Pittsburgh under the direction of H. P. Greenwald, and the results of these tests, although not published [16] until later, were available in reporting the results of the Butte work published [17] in 1927. Both programs of test work were designed primarily to investigate friction pressure losses, but numerous shock pressure losses were determined and other ventilation data obtained incidental to the main program.

Using the Butte results as a "framework" and amplifying by careful consideration of all the then available data as detailed in the report of that work, the author and A. S. Richardson developed a table of friction factors as applicable to both metal- and coal-mine practice. This table is repeated here (table 1), as the data that have since been reported [18] indicate no necessity for revising the values there given.

TABLE 1.—*Bureau of Mines schedule of friction factors for mine airways*

[Values in *italics* are recommended as good average values for theoretical computations]

Type of airway	Irregularities of surfaces, areas, and alinement	Values of $10^{10} \times k$[1]											
		Straight			Sinuous or curved								
					Slightly			Moderately			High degree		
		Clean (basic values)	Slightly obstructed	Moderately obstructed	Clean	Slightly obstructed	Moderately obstructed	Clean	Slightly obstructed	Moderately obstructed	Clean	Slightly obstructed	Moderately obstructed
Smooth-lined	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 rock (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	85	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	145
Igneous rock	Minimum	90	95	105	100	105	115	105	110	120	115	120	130
	Average	145	*150*	160	155	160	165	160	165	175	170	175	195
	Maximum	195	200	210	205	210	220	210	215	225	220	225	235

[1] 10 in table is equivalent to 0.0000000010; 100, to 0.0000000100. All values of k are for air weighing 0.075 pound per cubic foot.

In the construction of this table mine airways were grouped into four classes by major differences in type of surfaces, sedimentary rock including coal surfaces as of comparable degrees of roughness. Each class was then divided into 3 groups as to minimum, average, and maximum degrees of irregularity of surfaces, areas, and alinement that occur in practice, and the 12 values assigned to straight, clean airways of these 12 general types are not only the basic values upon which the table is constructed but also define the range of values for

[16] Greenwald, H. P., and McElroy, G. E., Coal-Mine Ventilation Factors: Bull. 285, Bureau of Mines, 1929, 104 pp.

[17] McElroy, G. E., and Richardson, A. S., Resistance of Metal-Mine Airways: Bull. 261, Bureau of Mines, 1927, 145 pp.

[18] Callen, A. C., and Smith, C. M., Measurement of Air Quantities and Energy Losses in Mine Entries, pts. I, II, and III: Univ. Illinois Exp. Sta. Bull. 158, 1926, 77 pp.; Bull. 170, 1927, 74 pp.; Bull. 184, 1928, 59 pp. Smith, C. M., Measurement of Air Quantities and Energy Losses in Mine Entries, pt. IV: Bull. 199, 1929, 48 pp.

calculating strictly friction pressure losses. They were determined upon mine airways of 20- to 60-square-foot cross-sectional area for the most part and at velocities of 500 to 2,000 f. p. m. Average spacing of timber sets was about 5 feet center to center, and areas were measured inside timbers. Effects of area, velocity, and other conditions on the factor, discussed in a later section, are slight and within the error of normal selection of factor, which should not be more than about 20 percent for general application and possibly within 10 percent for those personally acquainted with a variety of original test data—degrees of accuracy that are within the practical necessities of ordinary application.

Factors for airways that have wall surfaces of more than one type can be derived from these pure-type factors by weighting the factors for the types involved according to the proportional parts of the perimeter involved.

In extending the table from the 12 basic friction-factor values to values that include the effects of curvature and degree of obstruction, which cause shock pressure losses independent of the degree of roughness of the wall surfaces, strictly correct procedure has yielded to convenience, and the probable accuracy of selection is considerably less than that of the 12 basic factors. The irregularities of the airways involved, however, are so great compared to the error introduced and the increments used are so sketchily established that this procedure is justified for approximate calculations, even though a more accurate method is available through separate calculation of all shock pressure losses.

The actual increments to be made to friction factor k to include shock losses, as distinct from friction losses, vary directly with the ratio of area to perimeter of the airway. The extensions in the table are based on a mean value of this ratio for metal-mine airways of 1.5, for which shock pressure losses expressed in equivalent velocity pressure per 100 feet of airway, X, require an increment to k of about $0.0^{8}5X$. The exact relations are developed in the following general section on shock pressure losses, which also includes a description of conditions corresponding to the degrees or curvature and obstruction given in the table.

Effect of spacing of timber sets.—The factors for timber-lined airways are based on the assumption that, when the area used is that inside the timber sets, the effect of differences in spacing of sets can be considered as the effect of differences in the roughness of the surfaces responsible for the friction losses. No distinction is made between airways lined on 3 sides and those lined on 4 sides or between round and square timber, because the better alinement of the timbers in each of the latter cases balances the effect of the smoother floor or better edge condition in the former under average conditions, and the required friction factors are approximately the same. Although values for intermediate spacing are not available for average mine conditions we have the factor $0.0^{8}20$ for timber sets placed "skin to skin", $0.0^{7}100$ for timbering on 5-foot centers, and $0.0^{8}80$ for 10-foot centers. There is a certain spacing that will give a maximum pressure loss, because the effect of increasing the spacing is primarily an effect of increasing the intensity of the shock loss caused by each set while decreasing the number per unit of length. The effect of spacing on the friction factor under average conditions of metal-mine airways is

estimated to be about as indicated in figure 13, although confirmatory experimental data are lacking. For spacings greater than 10 feet approximate accuracy is obtained only by calculating the shock losses due to the timber sets separately, as explained in the following section on shock losses.

FRICTION FACTORS FOR TUBING

Experimental determinations [19] of friction factors for the types of tubing in common use in mines for auxiliary ventilation as attached to small fan units give factors that vary slightly with the material, area, frequency of joints, and (except for heavy canvas or jute tubing) velocity. The variations are small, however, except those between the comparatively rough canvas or jute tubing and the relatively smooth metal or wooden-stave tubing. That the differences between the rigid and nonrigid types are primarily a question of roughness rather than the requirement of inflation of the nonrigid types is apparently proved by the fact that the determined factors for tubing made of smooth, rubber-covered canvas were considerably lower than those obtained on rigid metal and wooden-stave tubing.

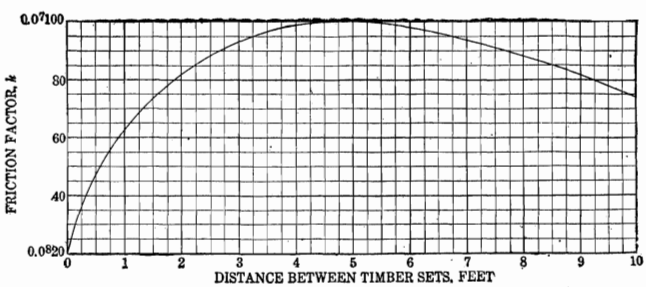


FIGURE 13.—Relative effect of spacing of timber sets on friction factor, as estimated for average conditions of metal-mine airways.

For a straight metal or wooden-stave line in good condition an average test factor of $0.0^{8}15$ seems directly applicable, as there is very little probable difference between test conditions of installation and the best conditions of underground installation. The best of the straight canvas or jute installations underground, however, are not as straight as test installations, and the constrictions at the couplings in general use are more pronounced; the average factors of about $0.0^{8}20$ to $0.0^{8}23$ obtained in experiments should therefore be increased to at least $0.0^{8}25$ for application to good, straight lines.

The average conditions under which tubing lines are installed in mines do not permit a good, straight line, so that shock losses of variable magnitude are imposed on the friction pressure losses, but the ratio of area to perimeter $\left(\frac{A}{P}\right)$ is small—0.125 for a 6-inch-diameter tubing and 0.50 for a 24-inch tubing—and the increments to k required to allow for them are small. Considering the major difficulty of handling leakage losses in computing pressure losses, no great

[19] Weeks, W. S., Goss, W. P., and Warren, G. H., Friction Factors in Ventilating Pipe: Eng. and Min. Jour., vol. 113, 1922, pp. 1001–1002. McElroy, G. E., and Richardson, A. S., Friction Factors for Fan-Piping Used in Mine Ventilation: Rept. of Investigations 2540, Bureau of Mines, October 1923, 15 pp.

degree of attempted accuracy is justified, and friction factors of $0.0^{8}20$ for metal or wooden-stave tubing and $0.0^{8}30$ for canvas or jute tubing are recommended for application to average metal-mine conditions. Pressure losses can be calculated more accurately by computing the shock pressure losses separately and adding them to the computed friction pressure losses for straight lines. No allowance is necessary for the effect of moderate distortion of flexible tubing, since the perimeter remains constant and the change in area that accompanies distortion from a circular to an oval section is very small.

SHOCK PRESSURE LOSSES

All departures from the condition of a straight airway of uniform cross-sectional area result in pressure losses that are here termed "shock" pressure losses, because analyses of experimental results indicate that they are always due to shock loss resulting from a faster stream expanding into a slower stream. Although all shock losses may be considered as caused by changes in the area actually occupied by the flow, for convenience they are divided into two general classes—those caused by changes in direction of the airway and those caused by changes in cross-sectional area of the airway.

SHOCK-LOSS FORMULA

Shock pressure losses have been found to bear a constant ratio to the velocity pressure corresponding to the mean velocity of flow, absolutely constant for most conditions of area changes and virtually constant for bends. They can be generally represented and computed, therefore, by means of the general formula,

$$H_S = XH_V, \tag{11}$$

where H_S is total shock pressure loss, inches of water;

H_V is the velocity pressure corresponding to the mean velocity of flow, inches of water (approximately $\left(\frac{V}{4{,}000}\right)^2$ at standard air density of 0.075 pound per cubic foot, p. 15); and

X is an empirical factor or the "shock factor" found by experiment.

Since H_S and H_V are always expressed in the same units and are equally affected by changes in air density, X is independent of both density and the units used, in which respect it differs from the friction factor used in formulas for computing friction losses. It is the number of velocity pressures equivalent to the shock pressure loss. When a summation of individual shock losses that occur regularly along an airway is desired X is the number per unit of length, for which 100 feet will be used throughout this bulletin. Whereas friction and shock factors each relate to but one condition of the airway—relative roughness of the surfaces and changes in the area occupied by the flow, respectively—the conditions for shock loss present an almost limitless variety as compared with the conditions of friction loss.

DATA AVAILABLE ON SHOCK FACTORS

The results of the author's examination and analysis of existing data on shock pressure losses at bends and area changes, in terms of shock factors applicable to mine-airway conditions, have been

reported [20] in Information Circular 6663, which forms the basis for the following brief presentation of the subject of shock pressure losses; for references, derivation of formulas, and more detailed discussion the reader is referred to this publication.

The examination reveals that, although no rational method of computing shock factors for bends has been developed, a large number of direct determinations of factors for bends in ducts of uniform area have been made, for which the general relations are consistent enough to permit their general application to bends in mine airways of uniform area. The test data are so inconsistent, however, that their application to normal flow conditions in mine airways is subject to a large degree of approximation. No rational method of computing factors for bends in airways of nonuniform areas and but few directly determined factors are available.

For symmetrical area changes in mine airways not only is there a rational method of computing shock factors for most of the conditions encountered, but also the agreement in the data required for the computations is good. However, area changes in mine airways are rarely symmetrical, and, as in the case of bends between different areas, the required shock factors cannot be computed until more data are available on unsymmetrical changes.

SHOCK FACTORS IN TERMS OF FRICTION FACTORS AND EQUIVALENT LENGTHS

Shock losses are independent of the roughness of the walls and therefore cannot be computed correctly as friction losses, although methods for obtaining approximate accuracy over a limited range of conditions are available. Many of the experimental results on shock losses are expressed in terms designed to facilitate their computation as friction losses, that is, as increments to the friction factor or as equivalent lengths of airway expressed in feet or diameters; formulas for expressing the relation between shock factors and these equivalent expressions of shock loss are therefore desirable.

Shock losses that are regularly intermittent are computed along with the friction losses by increasing the friction factor enough to make the result of the computation cover the total of both types of pressure losses. If k_s is the increment to the normal friction factor required for including a shock pressure loss of X velocity pressures per 100 feet, then at standard air density

$$k_s = 0.0^{8}324\frac{A}{P}X. \tag{12}$$

The increments to the friction factor thus vary directly as the $\frac{\text{area}}{\text{perimeter}}$ ratio, and since this ratio is largely a matter of the actual size of the airway the variation in increments for constant shock factors of X per 100 feet is large. For a 6-inch circle or square the A/P ratio is 0.125, for a 6-foot circle or square it is 1.5, and for a 20-foot circle or square it is 5.0. However, the A/P ratios for mine airways

[20] McElroy, G. E., Pressure Losses Due to Bends and Area Changes in Mine Airways: Inf. Circ. 6663, Bureau of Mines, November 1932, 34 pp.

of the sizes normally encountered range only from about 1.0 to 2.0, with an average of approximately 1.5, for which the increment to the friction factor is about $0.0^{8}5X$—the relation used in developing the table of friction factors (table 1) from the basic friction factors so as to include various degrees of intermittent shock losses. Compared to the normal friction factors the increments are relatively small for rough-walled mine airways, and the values of X available offer only an approximate degree of selection; the possible accuracy of results of calculations is therefore not affected seriously by this method of computing shock losses if such calculations are limited to the ordinary range of sizes of mine airways.

In computing the total pressure loss from the friction formula strictly local shock losses are allowed for by arbitrarily increasing the length over the actual length enough to make the result agree with the total of both types of pressure loss. The increment in length required often is given in terms of the equivalent number of diameters rather than directly in feet. If the equivalent feet of airway is represented by L, then at standard air density

$$L_s = 0.0^{6}324\frac{A}{kP}X. \tag{13}$$

Since the equivalent length in feet varies both directly as the $\frac{\text{area}}{\text{perimeter}}$ ratio and inversely as the friction factor the range of increments for constant values of X is so large that approximate accuracy of direct application of test data is possible only for a very limited range of values of k and A/P. For the limited range of average mine-airway conditions the equivalent length may range from about $300X$ to $15X$.

The expression of shock losses in terms of equivalent diameters of airway or duct merely means that increment of length L_s is divided by diameter D, and at standard air density

$$\frac{L_s}{D} = \frac{0.0^{7}81}{k}X. \tag{14}$$

The equivalent number of diameters is thus independent of the A/P ratio and, for constant shock factors, varies inversely as the friction factor. The range of direct application of test data is therefore limited to corresponding degrees of roughness of airway or duct; but for average mine-airway conditions, expressed by friction factors ranging from $0.0^{8}20$ to $0.0^{7}20$, the equivalent diameters may range from about $40X$ to $4X$.

FLOW CONDITIONS AT BENDS

Experimental determinations of shock losses at bends indicate that the air crowds to the far side of the bend and occupies less than the full area available at the departure end of the bend and that the excess of pressure loss over that due to friction, as determined by the rubbing surface, is a shock loss due to abrupt one-sided expansion from this contracted area to the full area following the bend, as pictured in figure 14, A. The degree of contraction in the area occu-

pied by the flow at the departure end of the bend apparently is determined not only by the position of the walls of the airway throughout the bend but also by the distribution of velocities over the cross-section at the entrance to the bend. The expansion takes place at a small angle and thus extends over a considerable length of airway downstream of the bend, depending on the degree of contraction preceding the expansion. Where expansion takes place from the contracted stream direct to the atmosphere, as from an elbow on the end of a duct, the shock loss is much larger than that for abrupt expansion to the airway area only, on account of the greater difference in the velocities involved in the shock loss. Where bends are close together the shock loss at the first bend may be affected through

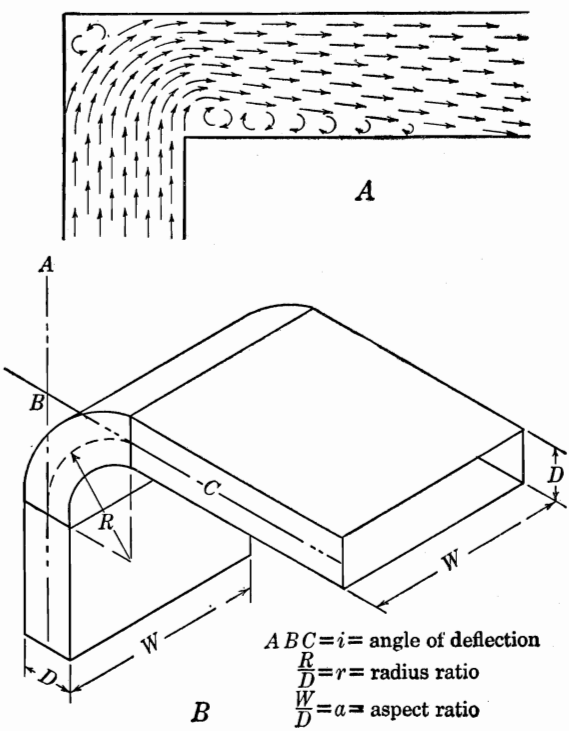


FIGURE 14.—*A*, Sketch showing general conditions of flow at bends in airways; *B*, sketch defining bend characteristics.

lack of complete expansion, in which case the normal velocity distribution at the entrance to the second bend and likewise its degree of contraction and shock loss are also affected to some extent.

In mine airways the area following the bend is rarely the same as the area preceding the bend, a condition that affects the degree of contraction and the resulting shock loss.

CHARACTERISTICS AND TYPES OF BENDS

The inner and outer corners of bends in ducts usually are rounded on concentric curves, because it has been found that the shock loss decreases as the degree of curvature is increased. The degree of

curvature is commonly designated by the ratio of the center-line radius to the width in the plane of turning, termed the "radius ratio", or R/D in figure 14, B. Just as it is harder to bend a plank on the wide dimension than on the narrow dimension, so it is easier as respects shock loss to turn the air flow through the larger dimension of the airway than through the smaller dimension. In rectangular airways the ratio of the side on which turning takes place to the other dimension is termed the "aspect ratio", or W/D in figure 14, B. The "angle" of a bend is the angle through which the air flow is deflected, or angle ABC in figure 14, B.

Right-angle bends in ducts are commonly termed "elbows"; since most of the test data available on bend losses relate to elbows the

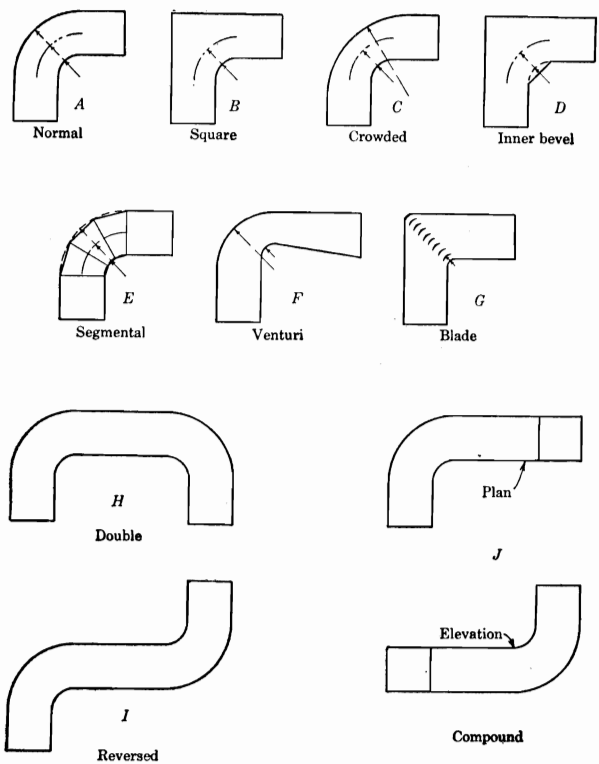


FIGURE 15.—Common types of right-angle bends.

type designations for bends in mine airways follow those used for elbows, as illustrated in figure 15. In a "normal" bend (A) the outer corner is a radial curve centered on the diagonal, and the inner corner is either square or commonly a similar concentric curve. In a "square" bend (B) the outer corner is square, and the inner corner may be square or a radial curve centered on the diagonal. These are the two common forms. Variants of these forms are: The "crowded" bend (C), where the radius of the outer curve is less than that of the normal bend; the "inner-bevel" bend (D), where the inner curve of the square bend is replaced by a bevel or chord subtending the normal curve; and the "segmental" bend (E) com-

mon to pipe and tubing lines. Special forms used for reducing the pressure loss where existing square inner or outer corners cannot be displaced are: The "venturi" bend (*F*), where the area occupied by the flow at the entrance is gradually reduced to about two thirds at the departure and then gradually expanded to normal beyond the bend; and the "blade" bend (*G*), where short, curved blades are installed along the diagonal of either a normal or square bend to divide the flow so that the separate sections through which it passes have better radius and aspect ratios than the original. Concentric-radius vanes installed in bends also reduce pressure losses by dividing the flow so that the separate sections through which it passes have different radius and aspect ratios than the bend without the vanes.

Two closely spaced bends in the same plane are termed a "double" bend (*H*) if they deflect the air flow in the same direction or a "reversed" bend (*I*) if they deflect it in opposite directions. Two closely spaced bends in different planes are termed a "compound" bend (*J*).

SHOCK FACTORS FOR BENDS IN AIRWAYS OF UNIFORM AREA

Virtually all data available on shock factors for bends in mine airways refer only to bends in airways of uniform area. Because of the variety of conditions that affect the pressure losses the data are not complete or consistent, and approximations must serve for the present.

NORMAL BENDS

Approximate shock factors for normal bends (fig. 15, *A*) may be computed from the formula

$$X=\frac{0.25}{r^2\sqrt{a}}\left(\frac{i}{90}\right)^2, \tag{15}$$

where X is the shock factor or shock pressure loss in equivalent velocity pressures, r the radius ratio, a the aspect ratio, and i the angle of deflection. For right-angle bends ($i=90$) in rectangular airways this reduces to

$$X=\frac{0.25}{r^2\sqrt{a}}, \tag{16}$$

and for right-angle bends in square or round airways ($i=90$ and $a=1$) it reduces to

$$X=\frac{0.25}{r^2}. \tag{17}$$

For the latter case the minimum value of r is 0.5 for a square inner corner, and the maximum value of X is 1.0. Good results are obtained with $r=1.5$ ($X=0.11$); there is little practical advantage in exceeding $r=2.0$ ($X=0.06$), that is, a center-line radius greater than twice the width or diameter.

SQUARE BENDS

Approximate shock factors for square bends (fig. 15, *B*) may be computed from the formula

$$X=\frac{0.60}{r\sqrt{a}}\left(\frac{i}{90}\right)^2, \tag{18}$$

which reduces to

$$X=\frac{0.60}{r\sqrt{a}} \tag{19}$$

for a right-angle bend in a rectangular airway and to

$$X=\frac{0.60}{r} \tag{20}$$

for a right-angle bend in a square or circular airway.

Bends in mine airways usually are square bends with a square inner corner, or $r=0.5$, which gives $X=1.20$ for a right-angle bend in a square airway. Fans are often so connected to multicompartment rectangular air shafts that the bend is made on the narrow dimension of the shaft where the aspect ratio may be about 0.5 and the shock factor 1.7, instead of on the wide dimension where the corresponding aspect ratio would be about 2.0 and the shock factor 0.85.

OTHER BENDS

Rough approximations of shock factors for other types of bends may be obtained from the two sets of formulas given. For the crowded bend the formulas for normal bends apply, but the factors must be increased arbitrarily; just how much is not definitely known. One set of tests indicates that a 40-percent increase is required when the outer radius is but three quarters the normal outer radius. The inner-bevel bend, often found at mine air shafts, may be treated as a square bend with a radius on the inner corner subtended by the bevel; and the segmental bend may be considered a normal bend with inner and outer radii circumscribing the segments, but a small increase over the factor for the equivalent normal bend is required to allow for a slight "crowding" of the outer corner. The common types of venturi bends may be considered normal bends having radius ratios of 0.8 to 0.9, depending on the details of construction. The factor for a blade bend may be taken as the factor for one of the equal sections considered as a single normal bend, but a small increase must be allowed for increased rubbing surface and edge effect. Factors for bends including radial vanes may be approximated by weighting the factors for the various sections according to the proportional parts of the area occupied and allowing a moderate increase for increased rubbing surface and edge effect. The best position for a single radial vane is about one third the width from the inner corner. Extensions to radial vanes increase the pressure loss slightly under most conditions and should not be used.

CHART FOR DETERMINING SHOCK FACTORS FOR BENDS IN AIRWAYS OF UNIFORM AREA

Figure 16 is a chart for determining shock factors for shock losses due to normal and square bends in mine airways of uniform area;

this is based on formulas (15) and (18). The method of using the chart is shown by the dotted-line solution of an example; the arrows indicate the direction of procedure. In the example solved in the chart it is shown that when the radius ratio of a normal bend is 0.6, the angle of deflection 45°, and the aspect ratio 0.5 the shock factor is 0.25. In using the chart it is important to note that the full lines for angle of deflection should be used for normal bends and the dash lines for square bends.

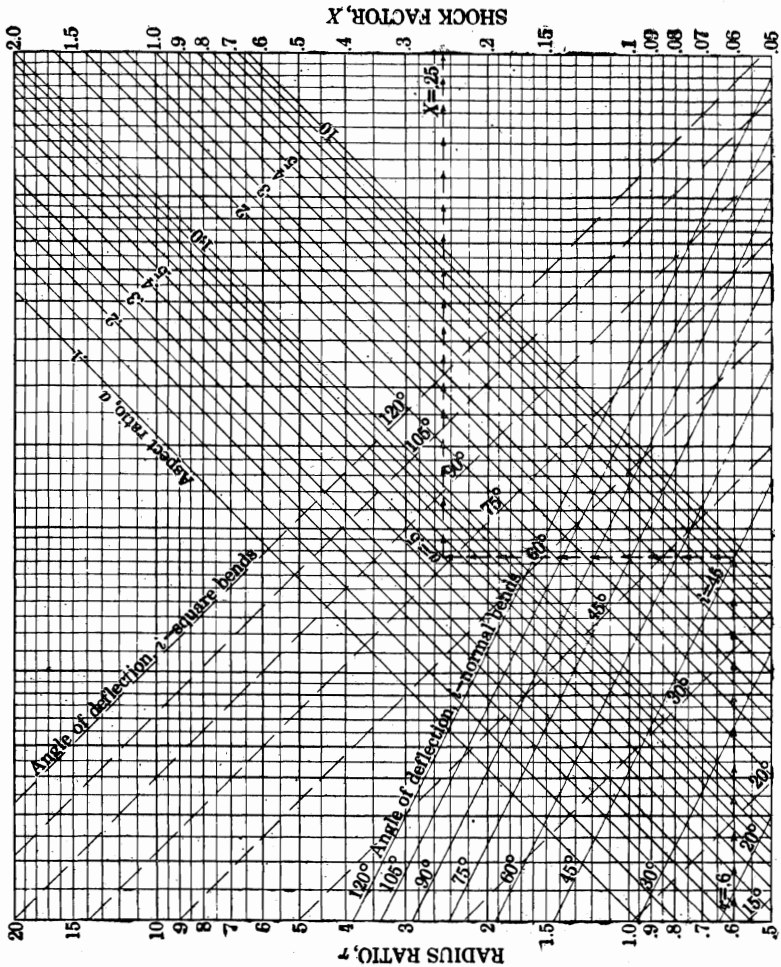


FIGURE 16.—Chart for determining shock factors, or shock pressure losses in terms of equivalent velocity pressures, for bends in airways of uniform area.

SHOCK FACTORS FOR BENDS IN AIRWAYS OF NONUNIFORM AREA

Except for a few special conditions, no factors have been directly determined for average conditions of area change at bends in mine airways, and no rational method of computing them has been developed. However, there are data on two conditions that indicate the effect of nonuniform areas—bends discharging directly to the atmosphere and closely spaced bends—and on intermittently occurring bends of small deflection in irregular airways.

BENDS DISCHARGING DIRECTLY TO THE ATMOSPHERE

If a bend discharges directly to the atmosphere, as at an elbow on the end of a pressure pipe line, the shock loss is not only much larger than the normal loss in an airway but also a constant percent larger, variously reported as 50 to 80 percent. With more consistent and more complete data possibly both the degree of contraction and the law for shock loss due to one-sided expansion can be developed from comparison of similar sets of data and applied for computing shock factors for average conditions of nonuniform areas.

CLOSELY SPACED BENDS

A double bend (fig. 15, *H*) has a lower pressure loss than two similar single bends, the amount depending on length of offset and type of bend involved. This low loss probably is due both to incomplete expansion following the first bend and higher velocities than normal at the outside of the second bend.

A reversed bend is reported to have a higher pressure loss than two similar single bends under the conditions tested, but the effect probably depends on the relative length of the offset. Incomplete expansion would reduce the loss at the first bend, but the resulting higher velocities at the inside corner would increase the loss for the second bend.

A compound bend probably would result in about the same pressure loss as two similar single bends, since the decrease in loss at the first bend would be largely offset by somewhat higher losses at the second bend due to higher velocities along one side; these, however, would affect both the inner and outer corners.

SINUOUS, CROOKED, AND CURVED AIRWAYS

Mine airways often include numerous small bends and curves involving deflections of 30° or less and are termed "sinuous", "crooked", or "curved", as the case may be. Drifts following a vein, alternately turning one way and then the other, may be sinuous if largely comprising short, curved sections or crooked if comprising short, straight sections. Where the deflections are approximately constant and in the same direction they are "continuous curves" or "turns" and are usually very irregular; the individual shock pressure losses are so small as to defy calculation, but a very rough approximation may be made of the combined shock losses as follows:

Slight degree: $X=0.2$ per 100 feet; a large-radius curve, a sharp bend of about 15° deflection, or a wall line close to the center line not oftener than once every 100 feet; or a small-radius curve, a sharp bend of about 30° deflection, or a wall line crossing the center line not oftener than once every 200 feet.

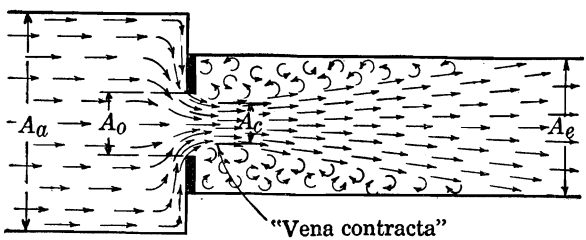
Moderate degree: $X=0.3$ per 100 feet; conditions intermediate between slight and high degree.

High degree: $X=0.5$ per 100 feet; a continuous large-radius curve, a continuous curve of repeated small deflections of 10° to 15° every 20 to 30 feet, bends of 20° to 30° deflection every 50 to 100 feet, or a wall line crossing the center line about every 50 feet.

The foregoing descriptions are rather meager for lack of comprehensive test data, but they are sufficient to permit of approximate computations that will satisfy the variable area conditions of mine airways.

FLOW CONDITIONS AT AREA CHANGES

Experimental determinations of pressure changes and pressure losses at changes of area of cross-section in mine airways, whether contraction or expansion, indicate that the excess pressure loss over the normal friction loss is a shock loss due to a faster stream expanding into a slower stream as determined by the actual areas occupied by the flow rather than the areas of the airway. No perceptible shock loss is due to the converging of the air stream itself where the flow is contracted, but the converging of the air stream causes the flow to contract beyond the edge of the constriction, giving rise to a condition known as the "vena contracta" (fig. 17) in which the area of cross-section of the air stream is the minimum immediately following the edge of the constriction. For contraction, therefore, the shock loss is that caused by expansion from this vena contracta to the full area following the contraction. Where expansion in area only is involved it may be considered as a special condition of general expansion following contrac-



$$\frac{A_c}{A_o} = c = \text{coefficient of contraction}$$

$$\frac{A_o}{A_a} = N_c = \text{ratio of contraction (areas)}$$

$$\frac{A_o}{A_e} = N_e = \text{ratio of expansion (areas)}$$

FIGURE 17.—Sketch showing general conditions of flow and defining characteristics of changes in area of airways.

tion, and all area changes may be considered on a single basis as variants of the general conditions illustrated by figure 17.

CHARACTERISTICS AND TYPES OF AREA CHANGES

Expansion after contraction involves four areas: A_a, the area preceding contraction; A_o, the area at the contraction; A_c, the area occupied by the air flow at the vena contracta; and A_e, the area after contraction, to which expansion occurs. Although the shock loss is determined by the relation of area A_c to A_e only, area A_c is largely determined by the relation of area A_o to A_a. Shock losses may be expressed in terms of the velocity pressure at any one of these four areas, the particular one selected being a matter of convenience in computations; both velocity pressures and corresponding shock factors are designated by the same subscripts as those used for the respective areas. The magnitude of shock losses expressed in velocity pressures is primarily a matter of three area ratios: (1) The ratio of the contracted area to the area preceding contraction, or A_o/A_a, which will be designated by N_c and termed the "ratio of contraction"; (2) the ratio of the area at

the vena contracta to the area at the contraction, or A_c/A_o, which will be designated by c and termed the "coefficient of contraction"; (3) and the ratio of the contracted area to the area following contraction, or A_o/A_e, which will be designated by N_e and termed the "ratio of expansion."

Since area A_c is not directly measurable the value of the coefficient of contraction is determined by calculation from experimental determinations of shock losses due to area changes. The fact that it has been found to be independent of the ratio of expansion greatly broadens the field of application of experimental data.

In determining shock losses due to area changes the degree of symmetry involved in the area change is the main consideration. Virtually all of the empirical data available apply only to symmetrical changes of area, that is, areas of the same shape centered as respects each other; for this condition both contraction and expansion of the flow affect the whole perimeter of the air stream uniformly. It is possible to develop formulas for both the coefficient of contraction and for shock factors, in terms of the ratios of contraction and expansion, for symmetrical area changes; but their application to unsymmetrical area changes is very uncertain and approximate, for although the effect of lack of symmetry on the coefficient of contraction may be estimated there is no way of estimating the effect of unsymmetrical expansion on the shock loss.

Conditions of expansion and contraction include abrupt expansion only, gradual expansion only, abrupt contraction followed by either abrupt or gradual expansion, and gradual contraction followed by either abrupt or gradual expansion.

SHOCK FACTORS FOR SYMMETRICAL AREA CHANGES

FORMULAS

The shock loss occasioned by a faster moving stream expanding symmetrically and abruptly to a slower moving stream has been found to be equivalent to the velocity pressure corresponding to the difference of the two velocities involved. This law has but a limited application in this form, but it can be developed to the following general forms for shock factors for abrupt symmetrical area changes in terms of area ratios:

$$X_o=\left(\frac{1}{c}-N_e\right)^2;\ X_e=\frac{\left(\frac{1}{c}-N_e\right)^2}{N_e^2};\ \text{and}\ X_a=\frac{\left(\frac{1}{c}-N_e\right)^2}{N_e^2}. \tag{21}$$

Where the expansion of the area occupied by the air stream is gradual and symmetrical it has been found that the shock loss is a constant ratio of that for abrupt symmetrical expansion; the required shock factors are

$$X'=yX, \tag{22}$$

where X' represents the shock factor for gradual expansion corresponding to the value of X (X_o, X_e, or X_a) for abrupt expansion and y is an empirical factor based on experiments. Values of y vary with the included angle or slope of the walls; values of y found by experiment are plotted in figure 18 against the included angle. "Fan Engineering" data indicate that angles greater than 60° are no better

than abrupt expansion but do not show that the normal angle of symmetrical expansion of air flow is about 7°, a fact indicated by Brigg's data and proved by a wealth of corroborating experimental data on air flow. Shock losses for gradual expansion depend on N_e as well as the experimentally determined values of y, and N_e is directly related to the slope of the sides and the length in diameters of the diverging section. The relations of slope to included angle and of N_e to length and included angle are also plotted for convenience in figure 18.

The author has found that a definite relation exists between the ratio of contraction and the coefficient of contraction, at least for symmetrical conditions of contraction, which may be expressed thus:

$$c=\sqrt{\frac{1}{Z-ZN_c^2+N_e^2}}, \tag{23}$$

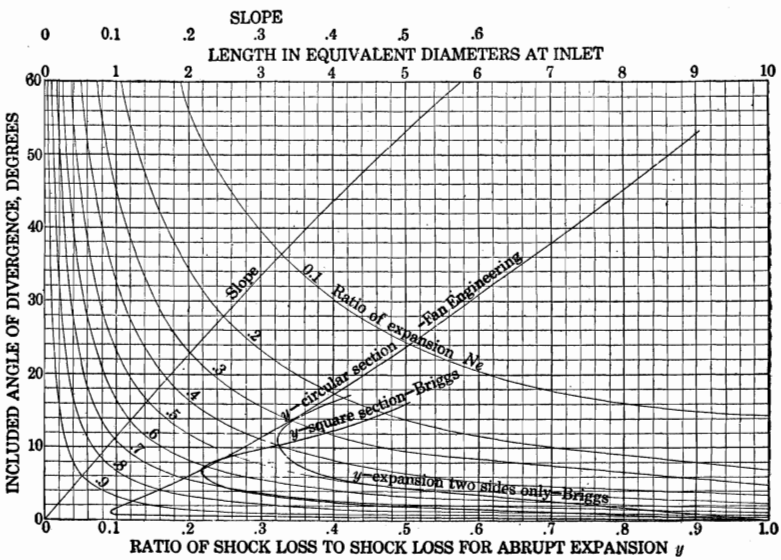


FIGURE 18.—Chart of data for gradual symmetrical expansion at diverging sections of airways for included angles of 0° to 60°.

where Z is an empirical factor, which will be designated as the "contraction factor." Values of Z are particularly affected by the edge condition at the constricted area, by the degree of abruptness of contraction, and by the form of the contraction. For free contraction to a sharp edge, as at the entrance to a plain open pipe, Z averages 3.80. For abrupt symmetrical contraction, that is, where a flat surface extends from a sharp edge at right angles to the direction of flow, Z is about 2.5 for ordinary degrees of sharpness as obtained with square-edge plates and square-edge timbers, although the very sharp edges of thin orifice plates used for air measurement give values of 2.7 to 2.8. If the edge is rounded the shock loss is greatly reduced, but since the edge condition is not susceptible to rigid specification this form is not applicable for air measurement; few data are therefore available for determining average Z values. For a well-rounded edge comparable to that presented by a round timber in a mine airway the

value of Z is estimated as about 1.5; although definite data are lacking it seems probable that the same approximate Z factor of 1.5 can be applied where the edge is beveled to about the same degree, that is, to the extent that the bevel would be circumscribed by a quadrant of a similar circular section. If the contraction in area is gradual instead of abrupt the degree of contraction of the flow at the vena contracta and likewise the shock loss are reduced. With a bell-mouth entrance or a "standard" orifice there is very little contraction of the flow, and an average value of Z for the best designs is about 1.05. This form can be considered as representing abrupt contraction to a definite edge condition and will be referred to hereafter as a "formed" edge.

For walls converging symmetrically on a uniform slope the only data available are for included angles ranging from 10° to 20°. Values of Z, plotted against included angles in figure 19, range from 1.57 for

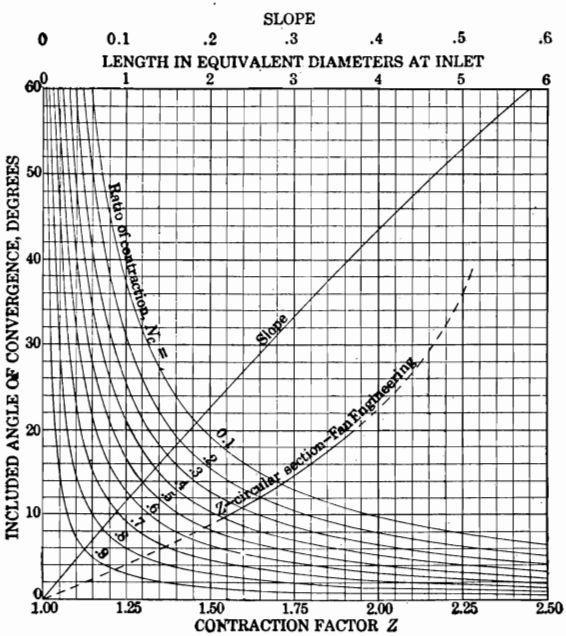


FIGURE 19.—Chart of data for gradual symmetrical contraction at converging sections of airways for included angles of 0° to 60°.

10° to 1.93 for 20°. At 0°, Z would be 1.0, and for abrupt contraction of 180° it would be about 2.5. The slope of the curve indicates that included angles of convergence greater than 60° are no better as regards shock loss than abrupt contraction.

The coefficient of contraction for gradual contraction depends not only on the experimentally determined value of Z but on the value of N_c also, and N_c is directly related to the slope of the sides and the length in diameters of the converging section. The relations of slope to included angle and of N_c to length and included angle are also plotted, for convenience, in figure 19.

INSTALLATION CONDITIONS

Installation conditions may be considered in three groups (fig. 20) for the purpose of presenting data on shock factors: (1) Entrance to an airway, (2) discharge from an airway to the atmosphere, and (3) within an airway.

However, a discussion based on the conditions of expansion and contraction is a more convenient method of presenting the various shock factors than this grouping.

Table 2 shows in condensed form the shock factors corresponding to the generally applicable contraction factors of 2.5 for a square edge, 1.5 for a rounded or beveled edge, and 1.05 for a formed edge, where

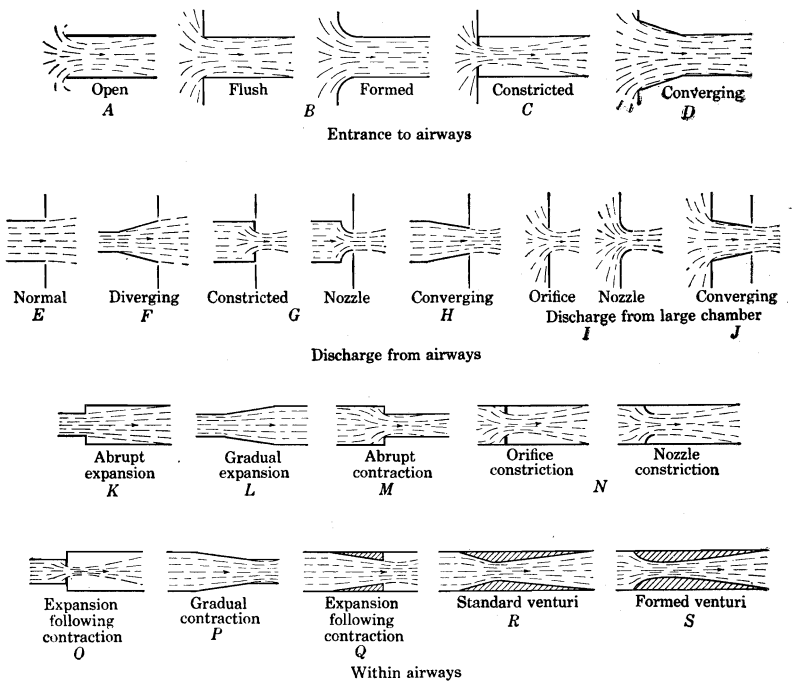


FIGURE 20.—Flow conditions at changes in area of airways.

N_c and N_e have fixed values of 1 or 0 or are equal to each other. The table also gives simplified forms of the general formulas applicable to these special conditions.

ABRUPT EXPANSION

Abrupt expansion that is not preceded by contraction of flow presents a special case of application of the general formulas which provide for such contraction, in which the value of N_c, Z, and C is 1.0. There are only two common conditions of installation: In the airway (fig. 20, K), where N_e may have any value between 0 and 1; and at discharge into the atmosphere (fig. 20, E), where N_e is 0 and X_a is 1.

TABLE 2.—*Shock factors or shock pressure losses, in terms of equivalent velocity pressures, for symmetrical area changes of airways for common conditions of abrupt*[1] *contraction and expansion in which the ratios of contraction (N_c) or expansion (N_e) are 1, 0, or equal to each other*

Constant ratio and special formulas	Variable ratio of contraction (N_c) or expansion (N_e)			1	0.9	0.8	0.7	0.6	0.5	0.4	0.3	0.2	0.1	0
	Condition of area change	Contraction factor, Z	Shock factor based on area before contraction (or expansion), X_a; on area at contraction, X_o; and on area following expansion (or contraction), X_e											
$N_c=1$ $X_a=X_o=(1-N_e)^2$ $X_e=\left(\frac{1}{N_e}-1\right)^2$	Abrupt expansion without preceding contraction.	(1.0)	$X_a=X_o$	0	0.010	0.040	0.090	0.160	0.250	0.360	0.490	0.640	0.810	1.0
			X_e	0	.012	.063	.184	.445	1.00	2.25	4.99	16.0	81.0	∞
$N_c=0$ $X_o=\left(\frac{1}{c}-N_e\right)^2$ $X_e=\frac{\left(\frac{1}{c}-N_e\right)^2}{N_e^2}$ $X_a=\infty$ $c=\sqrt{\frac{1}{Z}}$	Wall contraction to:													
	Square edge	2.50	X_o	.340	.462	.608	.774	.960	1.17	1.39	1.64	1.90	2.19	2.50
			X_e	.340	.569	.953	1.58	2.67	4.66	8.70	18.2	47.6	219	∞
	Round edge	[2] 1.50	X_o	.050	.105	.180	.275	.389	.524	.679	.854	1.05	1.26	1.50
			X_e	.050	.130	.280	.561	1.08	2.10	4.24	9.49	26.2	126	∞
	Formed edge	1.05	X_o	.001	.016	.051	.106	.181	2.76	.391	.526	.681	.856	1.05
$N_e=1$ $X_e=X_o=\left(\frac{1}{c}-1\right)^2$ $X_a=\frac{\left(\frac{1}{c}-1\right)^2}{N_c^2}$	Wall contraction to:													
	Square edge	2.50	X_a	0	.022	.091	.220	.446	.839	1.59	3.21	7.92	33.2	∞
			$X_o=X_e$	0	.018	.058	.108	.161	.210	.254	.289	.317	.332	.338
	Round edge	[2] 1.50	X_a	0	.003	.012	.030	.061	.120	.230	.471	1.18	4.96	∞
			$X_o=X_e$	0	.002	.007	.015	.022	.030	.037	.042	.047	.050	.050
	Formed edge	1.05	X_a	0	--------	--------	--------	.001	.001	.003	.005	.015	.062	∞
			$X_o=X_e$	0	--------	--------	--------	--------	--------	--------	--------	.001	.001	.001
	Obstruction with:													
	Square edge	[2] 5.00	X_a	0	.132	.495	1.13	2.19	4.00	7.41	14.8	36.0	151	∞
			$X_o=X_e$	0	.107	.317	.554	.787	1.00	1.19	1.33	1.44	1.51	1.53
	Round edge	[2] 3.00	X_a	0	.038	.152	.361	.723	1.35	2.54	5.14	12.5	52.7	∞
			$X_o=X_e$	0	.031	.097	.177	.260	.338	.406	.462	.501	.527	.536
	Formed edge	[2] 2.10	X_a	0	.012	.051	.127	.258	.503	.936	1.91	4.71	19.8	∞
			$X_o=X_e$	0	.010	.033	.062	.093	.126	.150	.171	.188	.198	.202
$N_e=0$ $X_o=\frac{1}{c^2}$ $X_a=\frac{1}{c^2N_c^2}$ $X_e=\infty$	Wall contraction to:													
	Square edge	2.50	X_a	1	1.59	2.41	3.60	5.45	8.50	14.1	26.3	61.1	249	∞
			X_o	1	1.29	1.54	1.77	1.96	2.13	2.26	2.36	2.44	2.49	2.50
	Round edge	[2] 1.50	X_a	1	1.35	1.84	2.56	3.67	5.50	8.88	16.2	37.0	150	∞
			X_o	1	1.10	1.18	1.26	1.32	1.38	1.42	1.45	1.48	1.50	1.50
	Formed edge	1.05	X_a	1	1.25	1.59	2.09	2.86	4.15	6.51	11.6	26.3	105	∞
			X_e	1	1.01	1.02	1.03	1.03	1.04	1.04	1.04	1.05	1.05	1.05

$N_e = N_c (= N)$

$X_o = \left(\frac{1}{c} - N\right)^2$

$X_a = X_e - \frac{\left(\frac{1}{c} - N\right)^2}{N^2}$

Wall contraction to:													
Square edge	2.50	$X_a = X_e$	0	.068	.304	.807	1.78	3.67	7.61	17.0	46.4	218	∞
		X_o	0	.055	.195	.395	.641	.918	1.21	1.53	1.86	2.18	2.50
Round edge	[2] 1.50	$X_a = X_e$	0	.027	.128	.361	.836	1.81	3.92	9.12	25.9	126	∞
		X_o	0	.021	.082	.177	.301	.453	.627	.820	1.03	1.26	1.50
Formed edge	1.05	$X_a = X_e$	0	.014	.068	.199	.480	1.08	2.41	5.79	17.0	85.6	∞
		X_o	0	.011	.044	.098	.173	.269	.385	.521	.679	.856	1.05
Obstruction with:													
Square edge	[2] 5.00	$X_a = X_e$	0	.224	.909	2.23	4.60	9.00	17.8	38.2	100	452	∞
		X_o	0	.182	.582	1.09	1.66	2.25	2.85	3.44	4.00	4.52	5.00
Round edge	[2] 3.00	$X_a = X_e$	0	.093	.409	1.06	2.30	4.68	9.57	21.1	56.9	264	∞
		X_o	0	.075	.262	.520	.828	1.17	1.53	1.90	2.27	2.64	3.00
Formed edge	[2] 2.10	$X_a = X_e$	0	.049	.227	.616	1.38	2.92	6.09	14.4	38.1	181	∞
		X_o	0	.040	.146	.302	4.79	.730	.974	1.30	1.52	1.81	2.10

[1] For gradual contraction interpolate in table for value of Z from fig. 19. For gradual expansion, with or without preceding contraction, multiply tabular values by y from fig. 18.
[2] Tentative approximations only

GRADUAL EXPANSION

Gradual expansion that is not preceded by contraction of flow also presents a special case of application of the general formulas which provide for such contraction, since the value of N_c, Z, and c is 1.0; but the shock factor, equation (22), is y times the factor for abrupt expansion, and y is derived from figure 18 or a similar source. The most common condition of installation is that within the airway (fig. 20, *L*). This form, the diverging connection or transformation, finds an important application in reducing pressure losses in mine airways by substituting gradual expansion for abrupt expansion. The same type of construction may be used at the discharge end of the airway (fig. 20, *F*); but gradual expansion is then followed by abrupt expansion, each with a separately determined factor, although the two factors can be combined into a single factor in terms of the normal area of the airway or of the area at discharge. This form is also of great practical importance in reducing the pressure losses at the discharge from mine-airway systems. As applied to a fan discharging to the atmosphere it is termed an "evasé" discharge, the main purpose of which is to reduce the velocity at the place where abrupt expansion (to zero velocity) occurs. For practical purposes it is rarely economical to use a length that will make the discharge area more than 4 times the area at the entrance to the diverging section, that is, to make N_e less than 0.25.

ABRUPT CONTRACTION

Abrupt contraction followed by abrupt expansion is a common cause of pressure loss in mine airways and embraces all three common conditions of installation. This fact is not readily apparent unless one realizes that the areas actually occupied by the flow rather than airway areas are the determining conditions. The common picture of abrupt contraction is that where a smaller section follows a larger section with abrupt transition in areas, as in figure 20, *M*, where $N_e=1$.

An unrestricted entrance presents a quite similar form (fig. 20, *A* and *B*) if the area preceding contraction is considered as expanded indefinitely. In addition to the special condition of $N_e=1$ is the special condition of $N_c=0$. Figure 20, *A*, illustrates the special condition of free contraction to an edge, as at the entrance to an exhaust pipe line, for which $Z=3.8$, rather than the common condition of contraction against or guided by a surface. The flush surface entrance (fig. 20, *B*) represents the common entrance form for mine airways, where $X_e=0.34$ for the ordinary square edge, $X_e=0.05$ for a rounded or beveled edge having a radius or bevel approximately equal to the wall thickness of a lined mine airway, and $X_e=0.0006$ for a formed edge.

A restricted entrance (fig. 20, *C*) has only the one special condition $N_c=0$; it follows that the coefficient of contraction c, equation (23), is 0.633 for a square edge ($Z=2.5$), 0.817 for a round edge ($Z=1.5$), and 0.976 for a formed edge ($Z=1.05$). Abrupt constrictions at discharge from a duct to the atmosphere (fig. 20, *G*) have only the special condition $N_e=0$. Both of these forms are frequently used in testing fans, usually with orifice plates whose edge condition can be closely approximated by $Z=2.5$ (square edge).

Flow through a small hole in the thin wall of a large chamber (fig. 20, *I*) may be considered a special condition of constricted dis-

charge. Both N_c and N_e are so small that they can be considered to be 0, and X_o=Z. The values of X_o for a square edge, round edge, and formed edge are, therefore, 2.5, 1.5, and 1.05, respectively, for mine-airway applications. The values usually given range from about 2.7 to 2.8 and are for orifice plates with very sharp edge conditions.

A constriction in an airway of uniform area (fig. 20, *N*) has only the special condition $N_c=N_e$ and represents the set-up for air measurement by orifice plates in ducts and constrictions in mine airways. However, the more usual condition of actual application to a constriction in mine airways, such as a regulator, involves unequal areas and presents the general condition of abrupt contraction followed by abrupt expansion (fig. 20, *O*). General formulas (21) and 23) are required for both, and tables of values or graphic charts are desirable to obviate computations.

GRADUAL CONTRACTION

Where gradual contraction is followed by abrupt expansion from the vena contracta to the area following contraction of the flow the shock factors are less than those for corresponding conditions of abrupt contraction because of the effect of the angle of convergence on the value of *Z*, which may be determined roughly for the common edge condition at the junction of surfaces from figure 19. The special condition of contraction at a uniform rate against a formed surface, pictured for particular conditions of installation in figure 20, *B*, *G*, *I*, and *N*, has a constant value of $Z=1.05$ and may therefore be considered abrupt contraction with a special edge condition, which is here termed the "formed edge."

The common form of gradual contraction is that in which a section of the airway converges in the direction of flow and the transition of areas is gradual. The common picture is that of a larger section converging on a uniform slope of sides to a smaller section, as in figure 20, *P*, for which N_e is 1 and *Z* is derived from figure 19 or a similar source. In mine airways a constriction that has been eased off on the approach side only might take the form shown in figure 20, *Q*, for which N_e is less than 1.

A converging discharge from an airway (fig. 20, *H*) has only the special condition $N_e=0$. A converging section used as the discharge from a chamber (fig. 20, *J*) or as the entrance to an airway (fig. 20, *D*) actually combines abrupt contraction with gradual contraction. The latter can be determined separately, but N_e is indeterminate for the abrupt-contraction condition at the entrance. The shock loss accompanying abrupt contraction at the entrance is reduced rapidly as the included angle of the converging section is increased, but separate graphs of shock factors are required for each combination of conditions of installation. None are given here, as these particular forms of installation of converging sections have little practical application to mine airways.

CONTRACTION FOLLOWED BY GRADUAL EXPANSION

Either abrupt or gradual contraction may be followed by gradual expansion, in which case the normal factors for abrupt expansion are multiplied by *y* (fig. 18) according to equation (22). Such combinations are seldom met in practice, except that of gradual contraction

followed by gradual expansion. This combination permits a large change of velocities and pressures with but a small shock loss and is therefore extremely useful in fluid-measurement methods based on pressure changes. The most common form is the standard venturi (fig. 20, *R*), although equally good results may be obtained with the special venturi (fig. 20, *S*) in which the converging section is replaced by the formed surface, for which Z may be taken as 1.05. Shock factors for the resulting shock loss, which primarily determines the cost of this form of air measurement for obtaining a continuous record, may be computed from general formulas (21), (22), and (23) by the use of appropriate values of y from figures 18 and, for the standard venturi type, of Z from figure 19 or similar sources.

CHART FOR DETERMINING SHOCK FACTORS FOR SYMMETRICAL AREA CHANGES

A chart for determining shock factors for the multitude of conditions of symmetrical contraction and expansion of the air stream that may be encountered in practice is presented in figure 21. It has 4 sections, marked *A*, *B*, *C*, and *D*, corresponding to the 4 possible steps in computations, all of which may or may not be required for a particular condition.

In *A* the intersection of the proper value of Z with the curve for N_c determines the value of c from equation (23). In *B* the intersection of the value of c with the curve for N_e determines the value of X_o from equation (21). Corresponding values of X_e and X_a are directly related to X_o through the relations of velocity pressures to area ratios; thus $X_e=\frac{X_o}{N_e^2}$, and $X_a=\frac{X_o}{N_c^2}$. Therefore in *C* the intersection of the proper value of X_o with the line for N_e determines the value of X_e; similarly, its intersection with the line for N_c determines the value of X_a. In *D* the intersection of the proper value of X_o, X_e, or X_a for abrupt expansion with the proper line for y (fig. 19) for the condition of gradual expansion determines the corresponding value of X_o', X_e', or X_a' for gradual expansion. Section *D* is not required, of course, for conditions of abrupt expansion; nor is section *A* required for conditions of expansion only, without preceding contraction, since $c=1$. In the latter case $X_a=X_o$, and X_o for $c=1$ is easily determined as the intercept of the N_e curve with the X_o scale in section *B*.

A continuous method of solving examples is indicated on the chart by the solution of a particular example; arrows show the direction of procedure. The example represents a condition of gradual expansion following abrupt contraction, where $Z=2.5$, $N_c=0.25$, $N_e=0.35$, and $y=0.3$. The solution shows that the shock factor corresponding to the area after expansion (X_e') is 3.5, and that corresponding to the area before contraction (X_a') is 6.9. Ordinarily, one would not be interested in X_o', but it could also be determined through section *D* if desired. The intermediate values c, X_o, X_e, and X_a are shown to be 0.645, 1.44, 11.8, and 23.1, respectively.

SHOCK FACTORS FOR UNSYMMETRICAL AREA CHANGES

Obstructions in mine airways, whether along the walls or out in the air stream, cause more or less abrupt contraction and expansion of the air stream, which are not symmetrical, and consequent shock losses

due to changes in the area occupied by the flow. Contraction may take place on one or more sides of rectangular openings or only along the edges of obstructions in the airway, and an obstruction may be moving relative to the air flow and in the same or opposite direction. The few data that have been presented on symmetrical contraction and expansion, mainly on the few types involved in air measurement

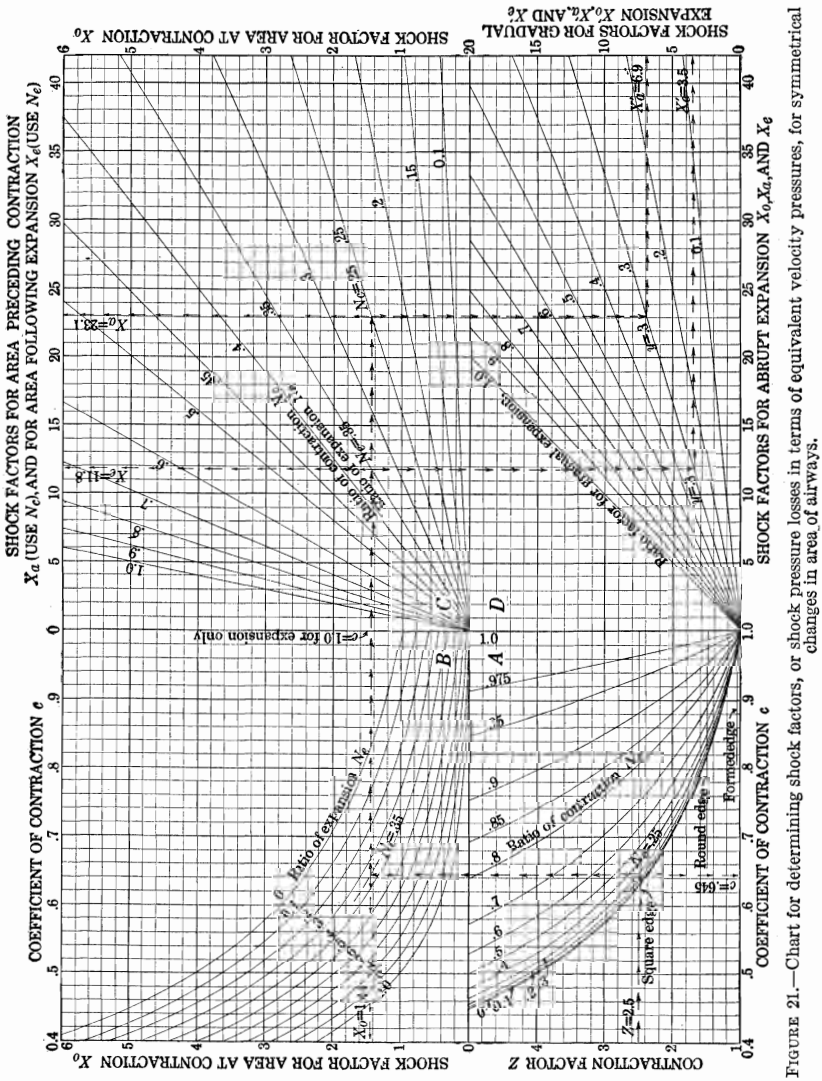


FIGURE 21.—Chart for determining shock factors, or shock pressure losses in terms of equivalent velocity pressures, for symmetrical changes in area of airways.

by pressure methods, do not begin to cover the field of shock losses in mine airways due to area changes and can serve only as a basis for very rough approximations until more data on the effect of unsymmetrical area changes on contraction factors and shock loss are available.

LOCAL ENLARGEMENTS OF AREA AND DEAD ENDS

In mine airways there are many local enlargements of one dimension of the airway over a limited length, such as dead-end stations off shafts, local widenings at sidetracks in drifts, and local increases in height, for which the shock loss is negligible because the expansion of the area occupied by the flow is small and over but a limited length. Where the enlarged section is relatively long shock losses result from expansion and contraction, but they are counteracted by the decreased friction loss occasioned by the lower velocity of flow in the intermediate section, with the net result that the change in pressure loss over that of an equal length of normal cross-sectional area is usually negligible. The total pressure losses for sections including such enlargements may therefore be taken as equivalent to the friction losses for equal lengths of normal area.

DOORFRAMES

Doorframes at open doors on mine airways are a typical form of abrupt nonsymmetrical contraction, with contraction and expansion on but three sides of an off-center rectangle. Until substantiated methods and values are available it is recommended that the formulas for symmetrical expansion be used with $Z=2.5$ for a square-timber frame only and $Z=2.0$ for the same frame with the door open.

REGULATORS

A common device used to control the distribution of air currents in mines is a small opening in a stopping that can be regulated, usually by a sliding shutter, to reduce the flow on one circuit in order to improve the flow on other circuits. It is simply a device for causing a desired loss of pressure and amounts to installing a square-edge orifice (fig. 20, *N*) in the airway. A value of 2.5 for Z is probably high enough for a symmetrically placed regulator opening, but the value of Z that should be used for different degrees of unsymmetrical placing, particularly where the opening is in a stopping parallel to the main flow, as at a split or junction, is unknown.

TIMBER OBSTRUCTIONS

A few Bureau tests indicate that shock factors for timbers across mine airways may be approximated roughly from the formulas for symmetrical expansion by doubling the contraction factor for the edge condition involved, that is, using $Z=5.0$ for a square edge. Lagging laid across the center of vertical shaft compartments gave an average Z value of 5.2, and center posts of roughly squared timber in a coal-mine entry gave an average Z value of 5.3. Contraction of the flow caused by obstructions of this type takes place only along the perimeter of the obstruction and not along the walls.

MINE CARS

A mine car or trip of cars causes a similar type of contraction of the air stream, but in this case the obstruction is so long that three pressure losses are involved: (1) Abrupt contraction, similar to the preceding case, except that the expansion following contraction takes place in the constricted area; (2) increase in friction pressure loss due to flow for a certain length of airway in the constricted section rather

than in the full area of airway; and (3) abrupt expansion from the constricted area to the full area of the airway.

The Bureau's data for pressure losses due to mine cars in a coal-mine entry indicate that shock factors for the pressure loss caused by the front of the car may be approximated roughly from the formulas for symmetrical area changes by doubling the normal value of Z for the edge condition involved, as for timber obstructions, that is, using a Z value of 5 for all square-edge obstructions in the airway and 3 for round-edge obstructions.

The friction loss due to high velocities through the constricted area may be large enough to justify precise calculations, but for ordinary purposes it can be taken as equivalent (within 5 percent for the average range of mine-airway conditions) to $\frac{2-N}{N^3}$ times the normal loss for the same length of unobstructed airway, where N is the ratio of constricted area to normal area. The shock loss due to abrupt expansion at the rear end of the car or trip of cars may be approximated by using shock factors for symmetrical expansion.

SHIELDING EFFECT OF OBSTRUCTIONS

Tests on mine cars have also revealed the effect of closely spaced obstructions to flow, which is commonly termed the "shielding effect." When the obstructions to flow are so closely spaced that the flow does not travel far enough downstream to expand fully before encountering another obstruction the shock loss is diminished to that corresponding to a higher value of N_e than the actual areas would indicate. Tests on cars in a coal-mine entry showed that the cars had to be spaced about 30 feet center to center (about 25 feet clear distance between cars) before the pressure loss due to a group of cars was equivalent to that for one car multiplied by the number of cars. Where the cars form a closely spaced train—the normal condition—there is but 1 front shock loss and 1 rear shock loss, and the intermediate cars contribute only the increased friction loss in the constricted airway proportional to the total length they occupy coupled together.

INTERMITTENT OBSTRUCTIONS

Mine airways often contain intermittent obstructions which change the area of cross-section and cause individual shock losses so small that they defy computation; it is then more convenient to determine directly the average total shock loss per unit of length, which may be approximated roughly as follows:

Slight degree: $X=0.1$ per 100 feet; trolley box, water box, large flanged pipe, occasional small falls of roof, occasional small crossbars, hangers, props, etc.

Moderate degree: $X=0.3$ per 100 feet; large fan pipes or tubings, occasional large crossbars or heavy hangers, frequent small falls, occasional constrictions, etc.

High degree: $X=1.0$ per 100 feet; combinations of obstructions given above, large falls, occasional storage piles of timber or pipe, closely set crossbars, props or constrictions, etc.

These values are for average areas determined by subtracting the area occupied by the obstruction where the latter is continuous and of appreciable size. For very high degrees of obstruction occasionally found in mine airways it is impossible to fix even approximate limits, as the normal characteristics become unimportant when compared with the characteristics of the obstructions; each case must therefore

be computed by analyzing its separate features and calculating separately the friction and shock losses involved.

SHOCK LOSSES AT SPLITS AND JUNCTIONS

Splits or junctions—air currents leaving or entering the main current—involve shock losses due to changes in both direction and area. The few test data available indicate that for splits or leaving currents the losses are comparable to bend losses, except that the velocity pressure of the partial current or current in the split should be used. Where minor air currents enter the main stream, however, both reason and test data indicate that the normal deflection shock losses for the entering stream are increased by its interference with the air stream of the main current. Exact quantitative data are lacking, but since the effects are know to be of considerable magnitude it seems prudent to use X factors for normal conditions, based on the velocity pressure of the entering stream, multiplied by at least 1.5, or an allowance of 50 percent.

Evidently shock losses at splits and junctions are largely determined by the proportional divisions of the total quantity of flow, which depend on the total friction and shock losses in the branch circuits as well as the shock losses at entrance and departure in each. Problems of calculating free distribution are therefore difficult. Approximate solutions require a reasonably close estimate of the proportional divisions of the flow, and accurate solutions are reached only through a process of trial and error.

Virtually all splits and junctions in mine airways present irregular conditions, such as nonuniform areas at bends and unsymmetrical area changes, for which no strictly rational methods of computation have been developed; although a few shock factors for special combinations have been determined directly, few data for these important types of pressure losses in mine airways are available.

EFFECT OF AIR-FLOW CONDITIONS ON CONSTANCY OF PRESSURE-LOSS FACTORS AND VELOCITY DISTRIBUTIONS

In the preceding discussion both friction and shock factors have been assumed to be constant; that is, pressure losses have been considered independent of flow conditions, except as represented by the formulas given, although reference has been made to the fact that slight variations in these factors were actually required to compensate for variations in flow conditions. Velocity distributions and other air-flow conditions have been found to vary in a similar manner, and the various sets of circumstances are apparently related. The variations are so small as to be a negligible factor in pressure calculations for such erratic and irregular conditions as those presented in mine airways, but they must be considered in accurate experimental work, in gaging the application of experimental data to conditions markedly different from the conditions of the experiment, and in their application under special flow conditions.

THEORY OF FLUID FLOW

These variations are adequately explained by a theory of fluid flow developed by English scientists, in which the variations are correlated with a factor termed the "Reynolds Criterion" of flow conditions;

good correlation has been obtained for both fluids and gases. This theory shows that the flow conditions of any fluid, including air, are the same for equal values of the Criterion for similar passages having the same relative roughness or same relative degree of disturbance of flow. The Criterion is the product of the diameter, velocity, and density divided by the viscosity.

FRICTION FACTORS

Many available friction-factor data for mine airways have been so correlated by the author in a previous paper [21] which shows that, although there are three distinct zones of fluid flow—viscous in which the factor decreases as the Criterion increases, unstable in which the factor increases rapidly as the Criterion increases, and turbulent in which the factor for low rates of pressure loss decreases slightly as the Criterion increases but in which it is virtually constant for high rates of pressure loss—fortunately mine-air flow falls in the turbulent zone and mainly involves high rates of loss.

For smooth-lined airways there is a perceptible change of factor in the mine air-flow range, but test work has demonstrated that minor changes in type of wall surfaces have such a large effect on friction factors that changes in flow conditions are relatively insignificant. For rough-lined airways there is no perceptible change of factor in this range.

In applying friction factors developed in small airways to relatively large airways with the same absolute conditions of roughness of surfaces the reduction in relative roughness that accompanies an increase of area must be taken into account, as well as the fact that friction factors may represent both friction and shock losses. For example, the author has found that with the same absolute roughness the friction factors for 400-square-foot railroad tunnels were about 25 percent less than for straight 40-square-foot metal-mine airways for both timbered and rock sections; and factors for smooth pipe were reduced 10 percent for a 4 to 1 area ratio. Similar and approximately equal ratios of friction factors have been reported for a range of sizes of rough-lined (corroded) pipe.

FLOW THROUGH CLOSELY PACKED MATERIAL

The critical velocity or velocity at which turbulent flow changes to unstable flow is about $22/D$ f. p. m., where D is the diameter in feet; such velocities are insignificant in mine airways. Velocities of flow which would bring the flow conditions into the viscous or the unstable zone practically occur only where air leaks through closely packed material such as finely broken ore and waste fill; here the pressure losses are so small and the distribution so complicated that mathematical analysis appears to be futile.

VELOCITY DISTRIBUTIONS

In normal flow through straight airways or ducts of uniform cross-sectional area the distribution of velocities is regular throughout the cross-section; the velocity at the center is much higher than the average and grades down to relatively low velocities close to the wall. The velocity for any position in the section, as averaged over an

[21] McElroy, G. E., Mine-Air Flow: Trans. Am. Inst. Min. and Met. Eng., vol. 74, 1926, pp. 297–311.

appreciable period of time, bears a constant ratio to the mean velocity of flow; the mean velocity can therefore be determined from the velocity at but one point in the stream. The ratio, however, has been found to vary with the Reynolds Criterion and correlates with the variations found in friction factors; that is, the rate of change is perceptible in smooth ducts but is practically nil in very rough airways. Since the other components of the Criterion are usually constant the change may be correlated against velocity only and, over the limited range of most test or experimental work, will plot as a straight line on logarithmic-coordinate paper. The ratio of mean velocity to center velocity is the position constant frequently used; for straight sections of uniform area with undisturbed flow in the turbulent zone it approaches a constant value of 0.81[22] at high values of the Criterion for smooth pipe but has this constant value of 0.81 for rough airways where the friction factors are constant. The positions of points of average velocity have been calculated on the bases that the distribution of velocities is a parabolic curve with the axis at the center of the airway and that the distribution curve is a logarithmic curve, a point still unsettled.

However, a perfectly straight duct with absolutely uniform area of cross-section is rarely met even in test work, and since position constants are affected by minor deviations from this condition they should be determined in situ. In test work on ducts and airways the author has determined center-position constants ranging from 0.75 to 1.0, although the general average is about 0.80. Widenings give low values, and constrictions give high values. Velocity distributions are also largely affected by conditions that disturb the flow and by temperature differences between the air and the walls. Warmer air gives higher velocities in the upper part of the section and colder air, higher velocities in the lower part of the section. With very small flows this effect may be so pronounced that velocity of flow is perceptible only along the roof or along the floor according to whether the air is warmer or colder than the rock.

SHOCK FACTORS

Shock factors have not been correlated by the Reynolds Criterion, but it seems probable that they can be so correlated, as the variations noted in test work agree closely with the variations in friction factors and in position constants of velocity distributions. For low rates of pressure loss where velocity of flow was practically the only variable a definite change of factor with velocity has been noted, whereas for high rates of pressure loss the change of factor has been practically nil. Similarly, appreciable decrease in the factor accompanied large increases in size of an elbow designed to have a low rate of pressure loss. Enough information on exact test conditions is not yet available to justify an attempt to correlate the data on shock factors, nor is the subject important as respects mine-airway pressure losses.

LOW-VELOCITY FLOW IN ROUGH AIRWAYS

In correlating friction factors by the Reynolds Criterion the curve for certain English tests on an 8- by 8-foot timbered airway showed a rapid increase in friction factor with decrease in the value of the

[22] Stanton, T. E., Friction: Longman's, Green & Co., 1923, p. 51.

Criterion, even though tests on a small-scale model of this airway showed an almost constant factor. F. Ernest Brackett suggested that the effect experienced was a velocity-cascade effect and that at the lower velocities of flow used in the tests, which extended as low as 15 f. p. m., the flow expanded into the space between timbers.[23] The author made a few observations with smoke in timbered airways to check this assumption and found that for velocities above about 250 f. p. m. the angle of expansion of the flow appeared to be constant at an included angle of about 7°, but as the average velocity of flow was decreased from 250 f. p. m. the angle of expansion increased to such an extent that at 50 f. p. m. the edge of the current struck the lagging behind 10-inch square timbers at about the midpoint between sets on 5-foot centers. A few friction-factor tests[24] made later in timbered and very rough rock airways confirmed this result—the friction-factor values increased rapidly as velocities were decreased below about 250 f. p. m. This result is significant but would rarely enter into pressure-loss calculations. It shows that the normal angle of expansion of air is not uniform but increases at velocities below a certain minimum. Both friction and shock factors are affected at these low velocities because, although shock loss only is directly involved, friction factors for rough surfaces, especially for timber lining, are based largely on included shock losses. These results also probably explain the "kink" in the curves obtained from tests[25] of pressure-volume relations at some English coal mines; the tests revealed a sudden change in the slope of the curve expressing the relation at low volumes of flow.

PRESSURE-QUANTITY RELATIONS

For fixed or constant conditions of an airway or system of airways the relation of pressure to quantity of flow is constant and can be simply expressed both as a mathematical aid in solving problems of combined pressure losses and for the direct determination and application of pressure-quantity relations without reference to such details as dimensions and other particulars of airway conditions. Many such expressions have been and are in use—"equivalent orifice", "mine temperament", "pressure potential", "inherence", "blast area", "Atkinson", "Murge", and "Guibal"—the values of the resulting factors depending both on the form of the expression and the units used.

RESISTANCE FACTORS

Confusion is obviated by using but one form for pressure-quantity relations; the most common form is $H=Rq^2$, that is, the pressure (H) equals a constant (R) times the square of the quantity (q) for constant conditions. This form was proposed by Atkinson about 1850 as a mathematical aid in solving problems of combined airways and by Shaw in England in 1890 and Rateau in France in 1891 for direct expression of the pressure-quantity relation for mines and sections of mines.

In this form R may be considered a unit of specific resistance comparable to the ohm of electrical resistance; its value will depend on the

[23] McElroy, G. E., Mine-Air Flow: Trans. Am. Inst. Min. and Met. Eng., vol. 74, 1926; discussion, p. 309.

[24] McElroy, G. E., Ventilation Experiments in the Moffat Tunnel: Eng. News-Record, Dec. 15, 1927.

[25] Briggs, H., Williamson, J. N., Penman, J. S., and Hyde, H., Experiments Concerning the Relationship between Pressure and Air Volumes in Mines: Trans. Inst. Min. Eng. (London), vol. 73, 1926–27, pp. 41–73.

units used for pressure, quantity, and density, although the latter term does not appear in the formula. Since pressures vary directly as the air density and the formula contains no density term, a standard density is implied. In mine-ventilation practice pressures are determined and expressed in inches of water and quantities in cubic feet per minute; 0.075 pound per cubic foot is the standard for air density. The use of cubic feet per minute as the unit of quantity would result in extremely small values of R for mine circuits; for convenience in computing and handling pressure-quantity relations for mine airways the author therefore uses 100,000 c. f. m. as the quantity unit. Values of R are then inconveniently large for fan-tubing calculations, but these are a minor phase of mine-ventilation calculations.

The standard formula for pressure-quantity relations used throughout the remaining part of this bulletin as an aid in discussing and evaluating airway conditions and mine and fan relations is:

$$H=RQ^2 \text{ or } R=\frac{H}{Q^2}, \tag{24}$$

where H is total pressure, inches of water, for an air density of 0.075 pound per cubic foot;
Q is quantity of flow, 100,000 c. f. m.; and
R is a factor of specific resistance, which will be referred to as "resistance factor."

R is equivalent to the pressure required for a flow of 100,000 cubic feet of air weighing 0.075 pound per cubic foot. The value of R is unity (1) when 1 inch of water pressure is required to pass 100,000 c. f. m. of standard air. With few exceptions all calculations can be carried out on a standard air-density basis. The complete formula is $H\times\frac{0.075}{d}=RQ^2$, but it is more convenient to make the calculations on a standard density basis and make the correction for density either at the beginning or end of the calculation, since many problems of distribution may be solved on a standard air-density basis without considering actual air densities.

Calculation of resistance factors.—Values of R may be determined by tests or calculated separately for friction and shock losses. Other designations for pressure-quantity relations may also be expressed in terms of equivalent values of R. The basic formula (10) for friction pressure loss may be written:

$$R_f=\frac{H_f}{q^2}=\frac{\frac{KPLq^2}{5.2A^3}\times\frac{0.075}{0.075}}{\left(\frac{q}{100{,}000}\right)^2}=\frac{k\ (100{,}000)^2\ PL}{5.2A^3}=\frac{KPL}{5.2A^3}, \tag{25}$$

where $K=k\ (100{,}000)^2=k\times10^{10}$.

The basic formula (11) for shock pressure loss may be transposed from a velocity base to a quantity base through equation (3) and the relation $V=\frac{q}{A}$:

$$R_s=\frac{H_s}{Q^2}=\frac{XH_v}{Q^2}=\frac{X0.07623V^2}{Q^2}=\frac{\frac{X0.07623q^2}{A^2}}{\left(\frac{q}{100{,}000}\right)^2}=\frac{623X}{A^2}. \tag{26}$$

For resistance in terms of equivalent orifice O, in square feet, from $O=\frac{0.000388q}{\sqrt{H}}$, $R=\frac{1,510.}{O^2}$ For resistance in terms of Atkinson's R_A, a similar unit in quite common use in England—that resistance which requires a pressure of 1 pound per square foot to pass a quantity of 1,000 cubic feet per second of 0.075 air—$R=0.535\ R_A$.

The large unit for quantity simplifies the calculation of friction losses, since k multiplied by $100{,}000^2$ is k times 10^{10}, a result that requires moving the decimal point 10 places to the right; for $k=0.0^820$, $k\times10^{10}$ becomes 20, and for $k=0.0^7100$, $k\times10^{10}$ becomes 100, figures that are more easily remembered and handled than actual values of k. The same holds for shock losses; 623 is more easily remembered and handled than 0.0^7623.

SERIES FLOW

The combined R for any individual section of the flow system is found by adding the individual R's for the particular friction and shock-loss conditions involved in the section. Similarly, where the total quantity of flow passes through a single circuit or system the resistance of the system (R) is simply the sum of the separate resistances (R_1, R_2, R_3 . . .) of its separate parts, and

$$R=R_1+R_2+R_3+\ldots \tag{27}$$

PARALLEL OR SPLIT FLOW

Where the total quantity flows through two or more airways in parallel whose individual resistance factors are R_1, R_2, R_3 . . . the equivalent series resistance (R) encountered by the total flow may be derived as follows: Total quantity (Q) is the sum of the separate quantities (Q_1, Q_2, Q_3 . . .), and the pressure difference (H) between the beginning and end of each split or path in parallel is the same; that is,

$$Q=\sqrt{\frac{H}{R}},$$

$$Q=Q_1+Q_2+Q_3+\ldots \text{ and } Q_1=\sqrt{\frac{H}{R_1}}, \text{ etc.}$$

Therefore

$$\left.\begin{aligned}\sqrt{\frac{H}{R}}&=\sqrt{\frac{H}{R_1}}+\sqrt{\frac{H}{R_2}}+\sqrt{\frac{H}{R_3}}+\ldots \text{ and}\\ \frac{1}{\sqrt{R}}&=\frac{1}{\sqrt{R_1}}+\frac{1}{\sqrt{R_2}}+\frac{1}{\sqrt{R_3}}+\ldots\end{aligned}\right\}. \tag{28}$$

The distribution of quantities will vary directly with the $\frac{1}{\sqrt{R}}$ value of the different splits; this value may be termed their "conductance factor" (C). Since $R=\frac{H}{Q^2}$, $C=\frac{Q}{\sqrt{H}}$.

CALCULATION OF RESISTANCE FACTORS FOR MINES

The resistance factor (R) for a mine or for a complete ventilating circuit is the equivalent series resistance for a system of airways arranged in series and parallel. For any large metal mine the airway system usually comprises long lengths, through which all the air flows, and an intricate network of cross-connected splits and branches complicated by leakage paths; exact mathematical solutions for R are impracticable, if not impossible. However, the main airways and main branches usually constitute 90 percent or more of the total resistance to flow, and approximate factors of sufficient accuracy can usually be obtained by analyzing the more important members of the network into a series of branching splits and adding the equivalent series resistance for the split flow to that for the series flow.

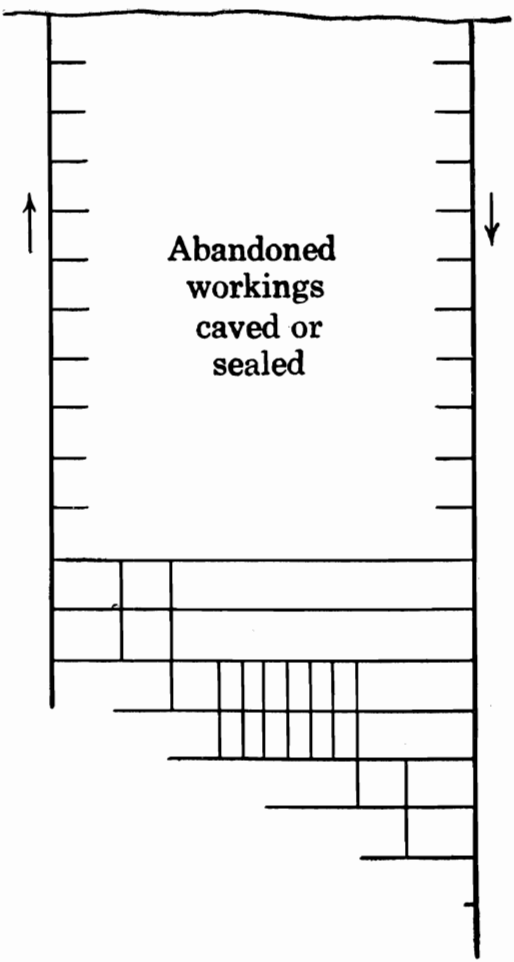


FIGURE 22.—Line diagram of the main airways of a typical metal mine.

The main purpose of such calculations, as regards metal mines, is to determine, in conjunction with whatever natural-draft components may be acting in series with it, the duty of a fan installation for mechanical ventilation; for most metal mines a sufficiently accurate figure for assuring fan operation close to maximum efficiency may be derived by considering only the main airways of the system and not the confused and intricate network of minor airways offered by a working zone of stopes or other openings.

An analytical sketch of the main airways of the usual metal mine will generally resemble a ladder with the upper rungs missing, as shown in figure 22. In this case there is only one place to start the

calculation—at the bottom split. The equivalent series resistance or resultant for this split is then added to the values of R for any airways in series with it, usually a section of a shaft and the resultant R for a number of raises; this combined R for the group of airways is then considered as one leg of the next split above. In this way the resultant for a larger and larger group of airways is obtained as leg after leg or rung after rung is considered, until finally a single value of R represents all flow on and below the first open level; this value of R added to the values of R for both upcast and downcast shaft sections above this level gives the R value for the system as a whole. Such a calculation requires many arbitrary decisions as to the effective length of airways to avoid the mathematical difficulties involved in handling networks, but a large latitude is allowed in the ordinary metal-mine airway system because of the generally small ratio of the series resultant for the split flow to the total resistance.

POWER-QUANTITY RELATIONS

The horsepower required to pass air through airways at a definite pressure is equal to the total pressure loss in pounds per square foot multiplied by the quantity in cubic feet per minute, which is the rate at which work is being done in foot-pounds per minute, divided by 33,000, the number of foot-pounds per minute equivalent to a horsepower. Since pressure in inches of water (at 60° F.) times 5.2 is equal to pressure in pounds per square foot,

$$\text{Horsepower (hp.)} = \frac{5.2\,Hq}{33{,}000} = \frac{Hq}{6{,}350}. \tag{29}$$

But $H = RQ^2$ and $q = 100{,}000Q$, so hp. $= \dfrac{RQ^2 \times 100{,}000Q}{6{,}350} = 15.75\,RQ^3$. (30)

This is the value for the air horsepower at standard air density. At other densities the pressure changes directly with the density ratio and the horsepower likewise. Since the process of applying and using power is never 100 percent efficient the actual power supplied to the prime mover of a ventilating appliance is always larger than the air horsepower, over-all efficiencies commonly ranging from 40 to 60 percent. The "brake horsepower" or actual power requirement is therefore $\dfrac{\text{air horsepower}}{\text{efficiency (as ratio)}}$.

CHARTS FOR DETERMINING RESISTANCE FACTORS AND PRESSURE AND POWER LOSSES

Tables for determining pressure losses in mine airways are available [26] but are not very satisfactory; since so many variables with different rates of variation are involved, concise tables cannot be readily interpolated, and tables complete enough to reduce the errors of interpolation to a reasonable limit would be very bulky and laborious to compute.

Figures 23, 24, and 25 were developed for the rapid determination of resistance factors, pressure losses, and air horsepower from dimen-

[26] Fitch, Thomas W., Jr., Mine Resistance: West Virginia Coal Min. Inst., June 1910. Reprinted by American Blower Co. as Pamphlet 297.
Weeks, W. S., Ventilation of Mines: New York, 1926, pp. 60–61.

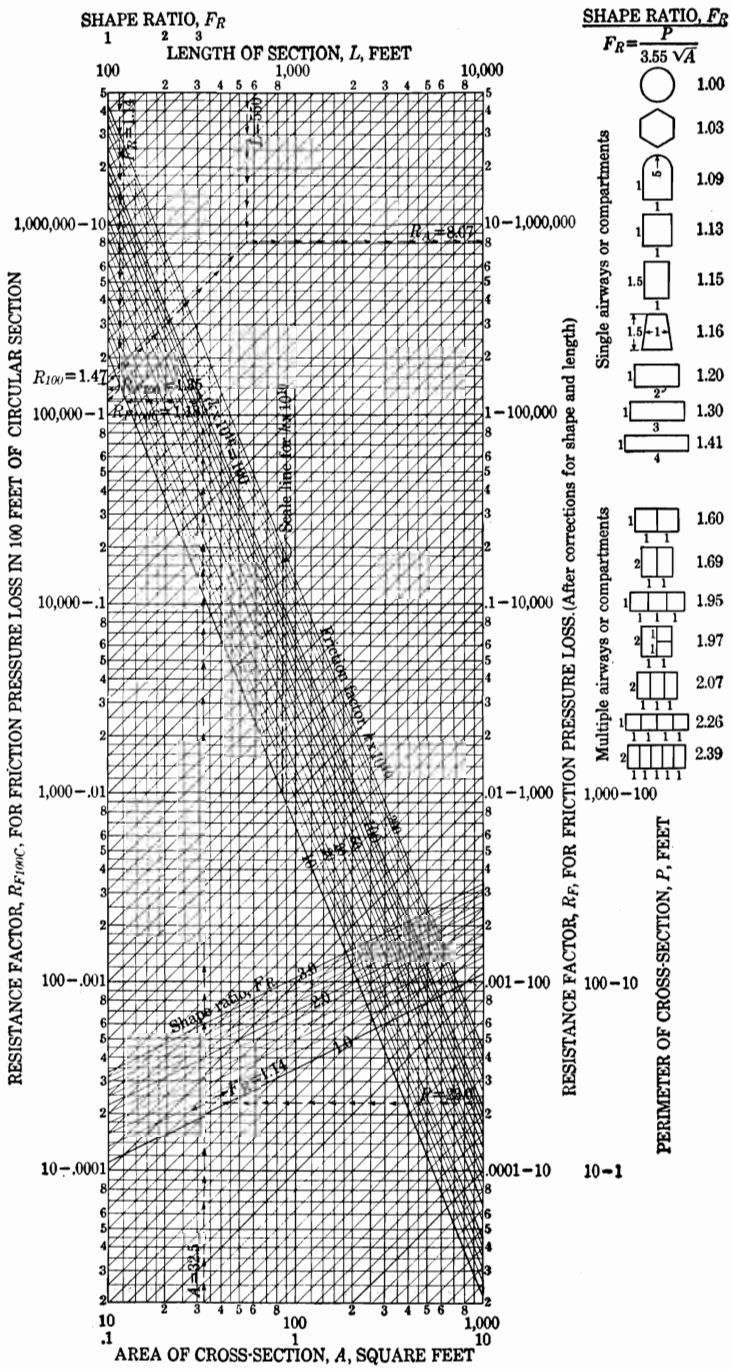


FIGURE 23.—Chart for determining resistance factors for friction pressure losses in airways.

NOTE.—Use corresponding scales of A and P and of A and R; $k \times 10^{10}$ may be scaled only along the line indicated.

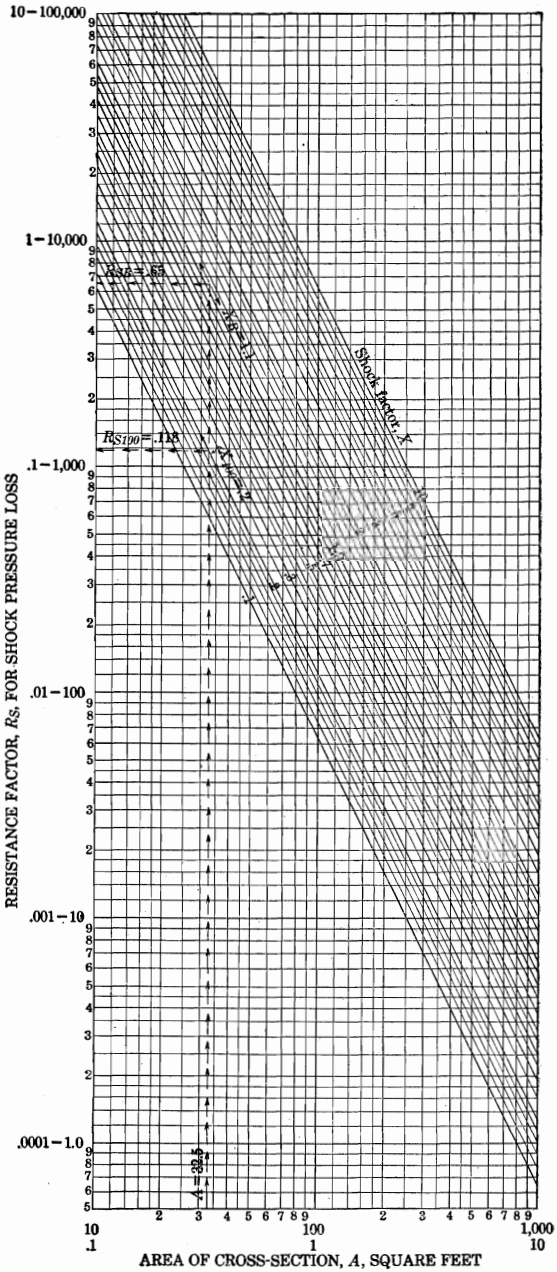


FIGURE 24.—Chart for determining resistance factors for shock pressure losses in airways.

NOTE.—Use corresponding scales of A and R; X may be scaled along any vertical "25" or "79" line. Moving the decimal point in X requires a similar movement of the decimal point in R.

sions, friction factors, and shock factors. They are based on a standard air density of 0.075 pound per cubic foot, but figure 25 includes a subsidiary chart for converting pressure and powers from this standard air density to actual air density, or vice versa. On account of the small scale used they serve only for very approximate solutions, but these are often all that the precision of the available data justifies. The charts also provide a rapid means of roughly checking more precise computations.

The general procedure in using the charts is to find the resistance factors for separate friction pressure losses from figure 23 and for separate shock losses from figure 24. These are then combined according to the principles of series or parallel flow, as the case may be, and the total resistance factor for the section of airway or system of airways, together with the known or assumed quantity of flow, are used to determine the pressure loss and air horsepower from figure 25. The latter chart can also be used to determine values of R corresponding to $\frac{1}{\sqrt{R}}$ and O (equivalent orifice). As a matter of fact, a large variety of air-flow problems may be solved by the use of these three charts.

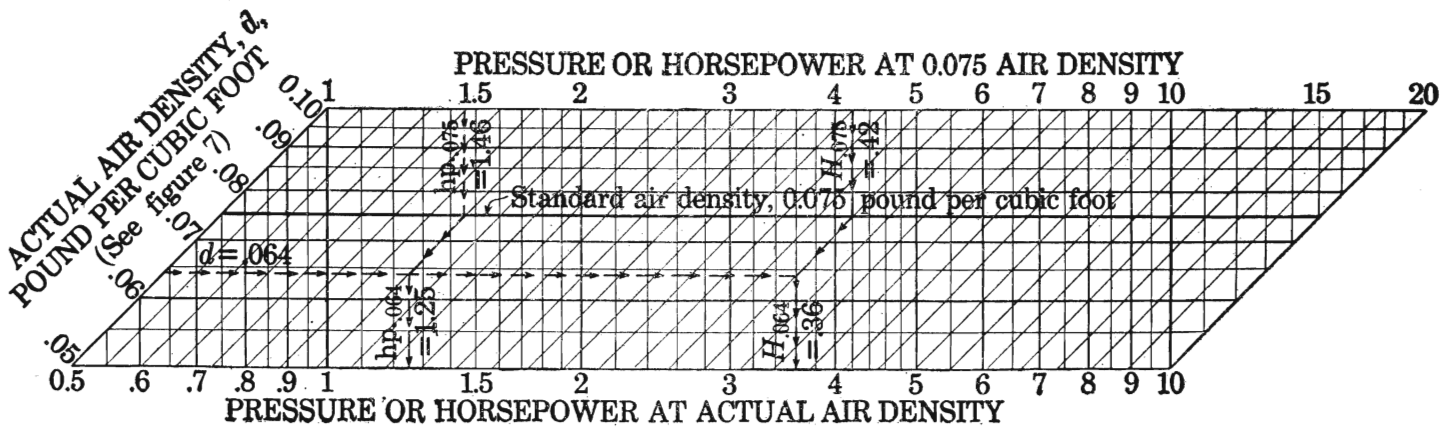
The friction chart (fig. 23) provides for the direct determination of the resistance factor (R_{F100C}) for the friction loss in a 100-foot length of circular section and multiplication of this value by both a shape ratio and a length ratio to give the resistance factor (R_F) for friction loss in a section of any shape or length. The length ratio is a simple proportion and requires no explanation. The shape ratio is also a question of simple proportion, as will be shown, but the proportion must be calculated, determined separately through the "shape-ratio" scale in the lower part of the chart, or estimated from the values given along the right margin.

The shock chart (fig. 24) is plotted directly from equation (26) and the pressure and power chart (fig. 25), from equations (24) and (30).

Derivation of shape ratio, F_R.—The influence of shape of section on friction pressure losses and resistance factors representing these losses is shown in general formulas (10) and (25) by $\frac{P}{A^3}$. Since both P (perimeter) and A (area) are determined by the same dimensions their ratio may be determined in terms of area only for different shapes. By expressing both perimeter and area in terms of the same dimensions it will be found that for any particular shape $\frac{P}{A^3}=\frac{\text{constant}}{A^{5/2}}$. If F represents the constant then

$$F=\frac{P}{\sqrt{A}}. \tag{31}$$

F may be termed the "shape factor" since it determines the relative value of different shapes with respect to resistance factors and thus to pressure and power requirements. For a circle F equals 3.55; for a square, 4.00; for a rectangle proportioned 1½ to 1, 4.08; for a 2 to 1 rectangle, 4.24; and for a 3 to 1 rectangle, 4.62. For irregular shapes F may be determined from direct measurements of perimeters and areas through equation (31). Where the flow is through similar sections, whether contiguous to each other or not, as in shafts divided into compartments or in airways in parallel, the shape factor for the combination will be found equal to $F\sqrt{N}$, where F is the shape factor



A

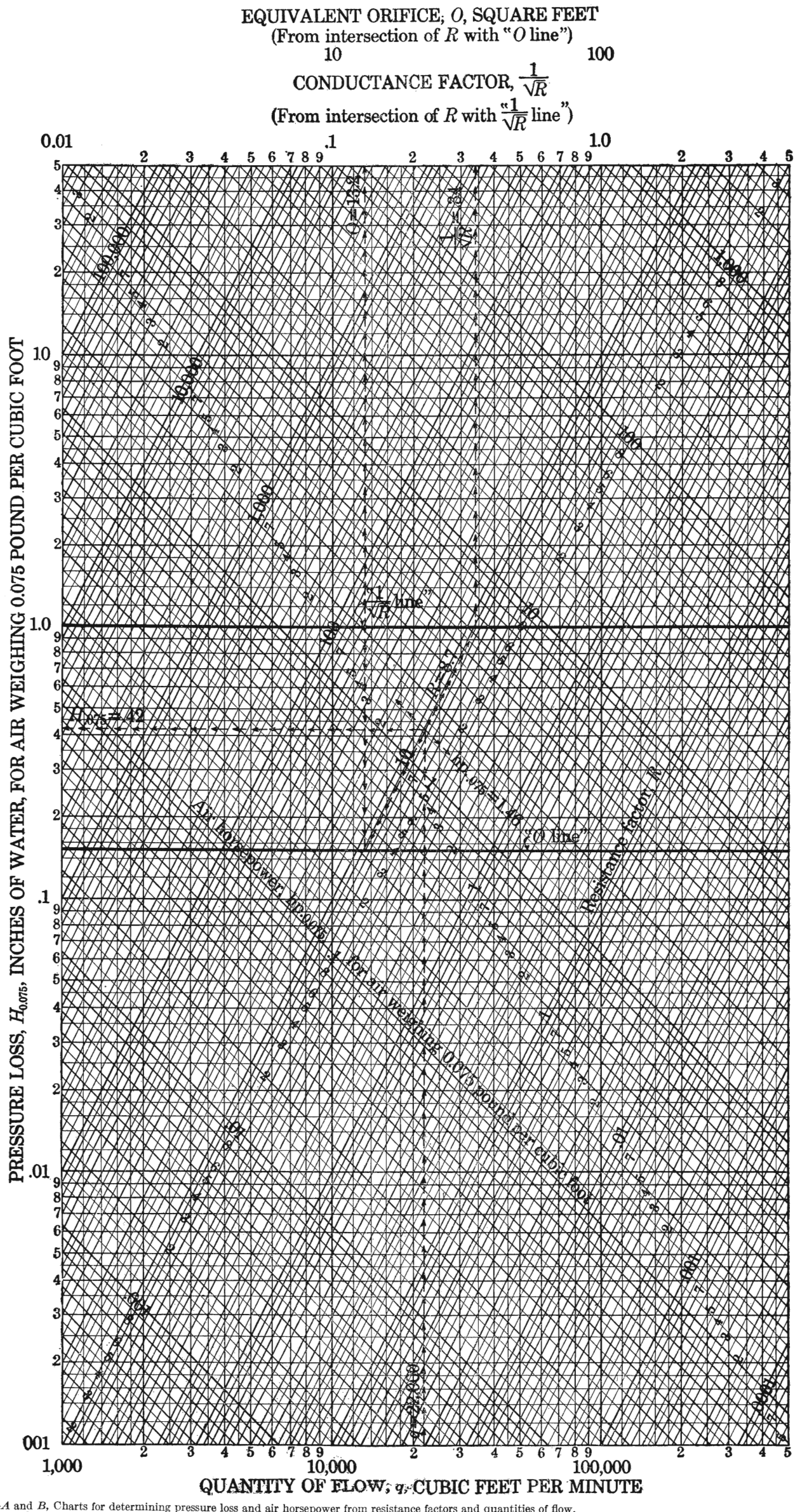


FIGURE 25.—*A* and *B*, Charts for determining pressure loss and air horsepower from resistance factors and quantities of flow.

NOTE.—*R* may be scaled along any vertical "635" line; hp. may be scaled along any vertical "1" line. Moving the decimal point 2 places to the right in *R* requires moving it 1 place to the left in $\frac{1}{\sqrt{R}}$ and in *O*.

for the single compartment or airway and N is the number of compartments or airways in parallel. If all conditions other than shape are constant resistance factors and pressure and power losses will vary directly as the shape factors. This expedient has been used in developing the friction chart (fig. 23), which is plotted for a circular shape, for which shape factor F is 3.55. Corrections are therefore required for other shapes according to the ratio of their shape factor to 3.55. If this shape ratio is designated by F_R then

$$F_R=\frac{F}{3.55}=\frac{P}{3.55\sqrt{A}}. \tag{32}$$

The transposed form of equation (25) used in plotting the friction chart is thus

$$\left.\begin{aligned} R_F=\frac{k\times10^{10}\times P\times L}{5.2A^3}=\frac{k\times10^{10}\times3.55\times100}{5.2\times A^{5/2}}\times F_R\times\frac{L}{100}= \\ \frac{k\times10^{10}\times68.2}{A^{5/2}}\times F_R\times\frac{L}{100}\end{aligned}\right\}. \tag{33}$$

Use of charts.—The methods of using the charts are best shown by the solution of a particular example, which is indicated by arrows on the three charts:

Example.—Find the pressure and power required to pass 22,000 c. f. m. of air weighing 0.064 pound per cubic foot through a 550-foot length of a 5- by 6.5-foot rectangular timbered airway, which has timbers in average alinement on 5-foot centers but is slightly sinuous and includes one right-angle bend.

Solution.—The area (A) of cross-section is 32.5 square feet and the perimeter (P) 23 feet. These determine the shape ratio (F_R) (lower part of fig. 23) as 1.14. For a straight airway the friction factor ($k\times10^{10}$) is estimated to be 100, which, for an area of 32.5 square feet, determines the resistance factor for 100 feet of similar circular section (R_{F100C}) as 1.18. To correct for shape proceed parallel to the 45° diagonal ruling from this value on the scale at the left to intersection with a vertical through $F_R=1.14$ from the top scale and then horizontally to the left scale. This procedure multiplies 1.18 by 1.14 and gives 1.35 as the resistance factor for 100 feet of straight section (R_{F100}).

Before correcting for actual length it is desirable to determine the resistance factor for the shock loss per 100 feet due to the sinuosity of the airway. For a slightly sinuous airway the shock factor for 100 feet (X_{100}) is taken as 0.2, and for an area of 32.5 square feet the corresponding resistance factor (R_{S100}) is 0.118 per 100 feet (fig. 24). The total resistance factor for friction and shock for 100 feet of sinuous airway (R_{100}) is then 1.35+0.12 or 1.47.

Enter figure 23 on the left with this value and proceed parallel to the 45° diagonal ruling to intersection with the vertical through $L=550$ and then horizontally to the scale on the right (or left). This procedure multiplies 1.47 by 5.5 and gives 8.1 as the resistance factor (R_A) for the sinuous airway.

If the shock factor for the particular bend conditions (X_B) is estimated as 1.1 the resistance factor for the bend (R_B) is 0.65 for an area of 32.5 square feet (fig. 24). The total resistance factor (R) for the airway is then 8.1+0.65 or 8.7.

In figure 25 the resistance factor of 8.7 determines the pressure ($H_{0.075}$) as 0.42 inch and the air horsepower ($hp._{0.075}$) as 1.46 for a flow (q) of 22,000 c. f. m. of air weighing 0.075 pound per cubic foot. Reduced to the actual weight of 0.064 pound per cubic foot through the subsidiary chart at the top of figure 25, the final results show that a pressure ($H_{0.064}$) of 0.36 inch of water would be required to pass 22,000 c. f. m. through this airway and that the air horsepower required ($hp._{0.064}$) would be 1.25 hp.

In this same figure the intersections of $R=8.7$ with the "$\frac{1}{\sqrt{R}}$ line" and the "O line" determine the condutance factor $\left(\frac{1}{\sqrt{R}}\right)$ as 0.34 and the equivalent orifice (O) as 13.2 on the upper scales. The $\frac{1}{\sqrt{R}}$ value would be used if this airway were considered as one of a number in parallel; it is actually a coefficient of conductance similar to O. Either may thus be used to determine the corresponding R for either single or multiple airways, provided they have been determined for a standard weight of air of 0.075 pound per cubic foot.

ECONOMICS OF AIR FLOW

The economics of air flow is mainly a question of balancing capital charges against power charges. Either can, of course, be lowered by increasing the other, but a proper balance between the two is the essential requirement for minimum total costs.

Power is determined primarily by quantity, since power varies as the cube of the quantity. If quantity is fixed the power varies directly as the resistance, or value of R for the circuit, which may be modified in many ways by changes in the physical characteristics of the system of airways involved. Increase in quantity may be obtained through increase in power consumption only, but the rate of increase is so rapid that large increases in quantity of flow without change of the resistance of the airway system often result in impracticable power increases; then the only practical solution is a reduction of the resistance through physical changes in the airways or in the way they are used.

EFFECT OF AIRWAY CONDITIONS ON POWER REQUIREMENTS

Since the power requirements vary directly with the resistance, which is represented by R for the airway or system of airways, the effect of the various physical conditions of the airways on the power requirements for constant quantities of flow can be readily examined through the formulas for R.

SIZE OF AIRWAYS

For airways of similar shape the value of R, and therefore the power requirements for friction loss for constant quantities, varies inversely as the five-halves power of the area and, since the change in area is twice the change in a similar dimension, as the fifth power of similar dimensions. For an area ratio of 2 to 1 the power ratio is 1 to 5.66; whereas for a dimension ratio of 2 to 1 the power ratio is 1 to 32.

On account of this preponderating effect of size of airways on power requirements due to friction pressure losses the provision of airways of ample size, whether single or multiple, is a primary consideration in obtaining a maximum ventilating effect in either naturally or mechanically ventilated mines and in realizing minimum power requirements in the latter. In practice this means large main airways, through which the total flow or large portions of it pass, and splitting of the flow in the workings so that the combined area through which the total flow passes is as large as is consistent with the general plan of the mine workings and full utilization of the velocity of the air current.

The preponderating effect of both area and quantity of flow on friction-loss power requirements and power costs is illustrated in table 3, which shows a ratio of power and power costs of about 21 to 1 as between a 4- by 6-foot and an 8- by 10-foot airway and of 64 to 1 as between flows of 25,000 and 100,000 c. f. m. Other assumptions for length, friction factor, over-all mechanical efficiency, and unit cost of power would, of course, alter the actual figures for power and power costs but would not change the ratios.

The effect of size is almost as pronounced for shock losses as for friction losses; the value of R in this case, however, varies as the square of the area or the fourth power of similar dimensions instead of the five-halves and fifth powers, respectively.

TABLE 3.—*Comparative power costs due to friction pressure losses for 1,000-foot lengths of straight timber-lined airways*

Size of airway (feet)	Area of airway (square feet)	Resistance factor (R) for K=0.071	Air horsepower for—			Yearly power cost at 60 percent over-all efficiency and at 1 cent per kilowatt-hour for—		
			25,000 c. f. m.	50,000 c. f. m.	100,000 c. f. m.	25,000 c. f. m.	50,000 c. f. m.	100,000 c. f. m.
4 by 6	24	27.82	6.85	54.8	438	$746	$5,965	$47,716
5 by 7	35	10.77	2.65	21.2	170	289	2,308	18,472
6 by 8	48	4.87	1.20	9.6	77	131	1,044	8,353
7 by 9	63	2.46	.61	4.8	39	66	528	4,219
8 by 10	80	1.35	.33	2.7	21	36	290	2,316

CHARACTER OF SURFACE OF AIRWAYS

The effect of roughness of the walls of the airway on resistance and power requirements is measured by direct comparison of the corresponding friction factors. As between the normal minimum and maximum conditions found in metal-mine airways, there is a range of about 1 to 10. If the pressure is constant the quantity of flow will vary as the square root of the ratio of the corresponding friction factors, a relation easily established by transposing the basic formula for friction pressure losses. The comparative sizes of different types of mine airways required to pass equal volumes of air with equal friction pressure losses are shown graphically in figure 26.

SMOOTH-LINING AIRWAYS

In most metal mines the major part of the total resistance to flow and thus the major consumption of power occurs in the main intake and return airways as these are commonly limited in size and carry

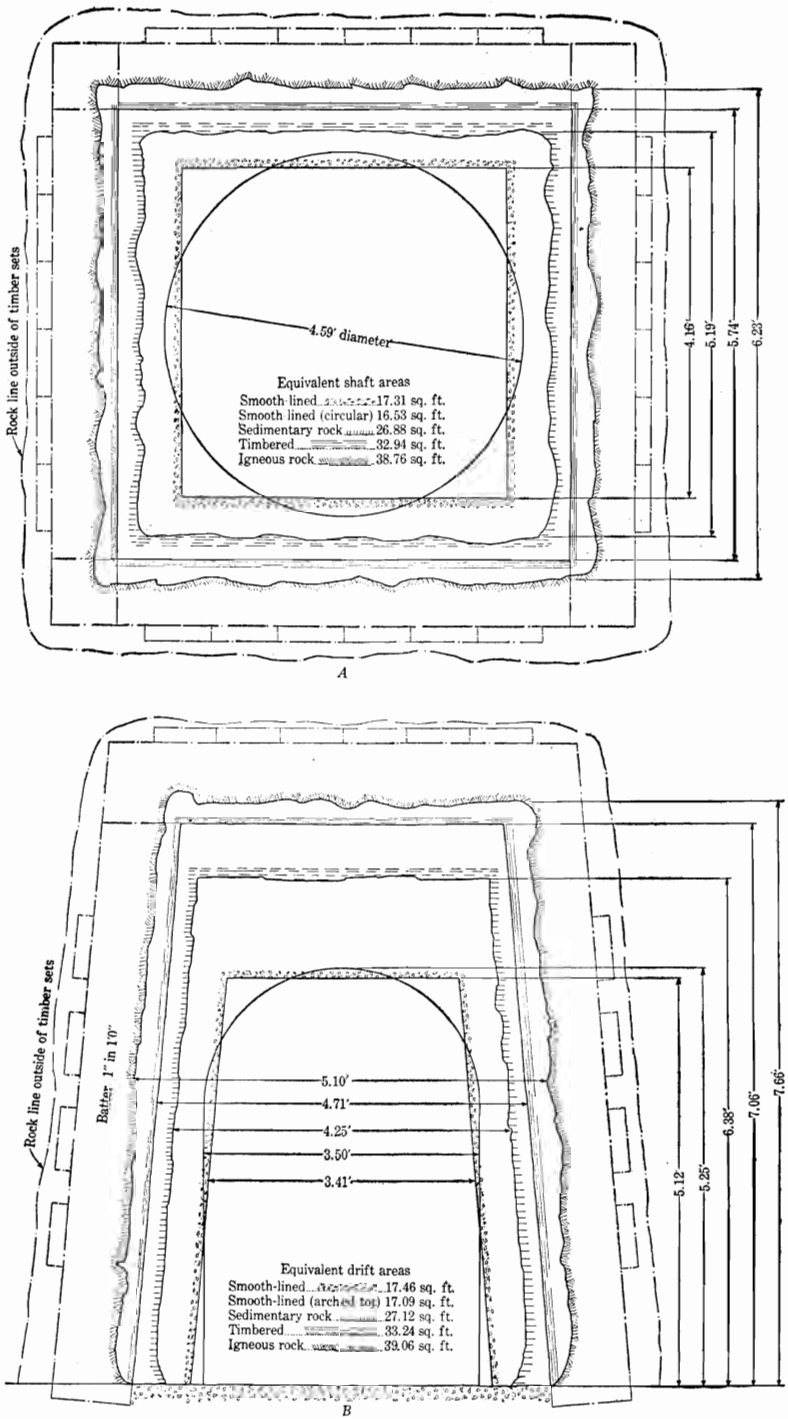


FIGURE 26.—Relative sizes of airways for equal pressure and power requirements: *A*, Square shaft sections: *B*, rectangular drift sections.

the total air current. Where these airways are types that have high factors of frictional resistance, such as timbered airways, the quantity of flow can be greatly increased by smooth-lining the airways. If all of the airways of a mine were changed from timber-lined to smooth-lined airways and the pressure remained constant the air flow would be increased about 2.2 times; a similar change confined to the main intakes and returns ordinarily would at least double the quantity of flow. In a naturally ventilated mine the increased power requirements would be supplied gratis. In a mechanically ventilated mine some of the increase in power requirements usually would be supplied gratis by the accompanying natural draft, and the speed of the fan could be reduced to bring the fan-power requirement back to its original value without material sacrifice of quantity or the quantity back to its original value with a large reduction in fan-power requirements.

SHAPE OF AIRWAYS

Compared to other factors shape of airway has a relatively slight influence on resistance and power requirements. Its effect on friction resistance is measured directly by shape factor F. If area and type of surface are fixed shape assumes some importance. The ratio of perimeter to area is the minimum for a circular shape, which is therefore the most advantageous shape as respects friction losses. A regular octagon has almost the same shape factor as a circle, and since this shape is better adapted to timber linings it is being used to an increasing extent in the metal mines of this country, particularly at Butte, Mont. If the relative resistance for a circular shape is designated as 1 then the values for other shapes are given by the shape ratios of figure 23. A range of approximately 1 to 2 for normal mine-airway shapes therefore offers some scope for obtaining minimum resistance to flow through attention to shape of airway, which is primarily a matter of original design rather than of alterations.

The resistance due to shock pressure losses is also affected to some extent by the shape of the airway, which affects the aspect ratio of bends and the degree of symmetry at area changes.

OTHER FACTORS

Resistance to flow and thus the power requirements are determined by both friction pressure losses and shock pressure losses, and any method of reducing either of these reduces the power requirements for constant quantities.

Friction pressure losses vary directly with the length; although length is usually a fixed factor it often can be adjusted to the minimum in planning a mine lay-out, or it can be reduced effectively by changing the existing lay-out of the distribution system.

Losses due to bends and changes of area can be reduced to the minimum by careful planning, proper alterations, and attention to mining details, such as removing obstructions from air courses. The exact procedure for any particular case is governed by the velocity of flow, since shock pressure loss is a function of the velocity of flow as determined by the quantity and area.

ALLOWABLE COSTS FOR REDUCING MINE RESISTANCE

Small main airways containing necessary bends and obstructions are economic conditions that must be met in metal-mine ventilation and offer a large field for improvements of a payable nature in all systematically ventilated mines. Most changes in metal-mine ventilation systems are made to increase the quantity of flow, and expense, although important, is secondary. Low-velocity flow involves so little power that costly changes in airway conditions or modifications of mining procedure cannot be economically justified. With high-velocity flow such matters as providing large-radius bends and removing or stream-lining obstructions require careful investigation although on account of the comparatively short time element involved in most metal-mining installations somewhat crude and relatively cheap installations are often more economical than elaborate and expensive types designed to reduce the pressure loss and power cost to the minimum. In judging the economic advantage of a change in airway conditions with respect to power requirements the saving in power costs should at least balance the capital charges—interest and amortization—of the investment required. In roughly estimating the advantage of a change it is convenient to remember that 100,000 c. f. m. at 1-inch pressure costs $1,000 per year at one half cent per kilowatt-hour with an over-all efficiency of the fan and drive of 51.5 percent. Costs for other quantities, pressures, and unit costs are then a matter of direct proportion. Velocity pressures at standard air conditions are approximately 1 inch at 4,000 f. p. m., one fourth inch at 2,000 f. p. m., and one sixteenth inch at 1,000 f. p. m. The amount that might economically be spent for each dollar of power saved is then primarily a question of the useful life of the installation, which is largely uncertain and therefore justifies approximate solutions only. Any increase in maintenance charges should, of course, be deducted from the power-cost saving; similarly, any decreases should be added.

ECONOMIC DESIGN OF MINE AIRWAYS

Although changes in existing airway and fan-installation conditions offer the most common opportunities for effecting economical operation of mine-ventilating systems the largest possibilities for obtaining this result lie in the original design of the main airways. Most of the openings so used are originally designed for other operating purposes. Natural conditions, such as heavy ground, often seriously limit the size of opening that can be maintained without excessive expense; and various operating factors, such as velocity limitations on traveling roads, often dictate size requirements in excess of those for economy in ventilation only.

Increasing the area of an airway increases the cost for excavation and lining almost proportionately but effects a very rapid decrease in power requirements. Balancing these two factors of expense to give the minimum total yearly operating cost for ventilation is mainly a matter of proper original design of the main airways of the mine, since the large volumes handled through these parts of the system involve large power costs and thus present opportunities for effecting large savings.

Total yearly costs for coursing air through mine airways are primarily a question of unit costs, quantity of flow, and type and size

of airway. All but the latter usually are fixed by the conditions obtaining, and design centers on the selection of a type and size of airway that will give minimum yearly costs for the existing conditions.

This calls for mathematical analyses involving a dozen or more variables that differ not only in range of values met in practice but also in rates of variation. The subject is too involved for adequate discussion here; it has been separately treated in a recent paper,[27] in which formulas and charts are presented for the determination of the elements of design where the conditions of service can be roughly approximated. The mathematical analyses made therein lead to the following conclusions:

1. The area for maximum economy is determined both by the absolute value of a dozen or more separate factors and by the relative value of the 2 groups—capital-cost factors and power-cost factors—into which they naturally fall; the relation between these 2 groups largely determines the area required for economy, which might vary as much as 4 to 1 for probable variations in practice.

2. Of the separate factors involved quantity of flow is the most important in that it affects the economic area at a rate three times that of all of the other separate variables. Range of variation in probable values of the other separate factors determines their importance as affecting economic area. Character of wall surfaces and ratio of unit cost of lining to unit cost of excavation are the two other separate factors of major importance. Unit cost of excavation, unit cost of power, shape, thickness of lining, service life, rates of interest, and mechanical efficiency of fans have but minor effects on economic area, decreasing in importance in the order given.

3. Where operating conditions limit the size of airways and require larger or smaller areas than the area for maximum economy the approximate cost of the limiting condition may be determined by the methods of mathematical analysis for economic area and cost.

4. When the area of either airway or fan-tubing line is that for maximum economy the yearly power cost is about 40 percent of the yearly capital cost if friction pressure losses only are involved. Assuming, as an average figure for metal mines, that the yearly capital charge is 20 percent of the original cost of construction, then the yearly power cost should be 8 to 10 percent of the original total cost, depending on the ratio of shock loss to friction loss.

The results of many calculations for assumed conditions lead to the following deductions:

1. Total costs per unit quantity of flow may vary as much as 20 to 1 for probable variations in the values of the variables involved, particularly unit costs.

2. Under conditions of comparable unit costs a probable range of variation of about 4 to 1 in total costs per unit quantity of flow, as affected by design, seems possible.

3. Shape of airway has only a minor effect on total costs and assumes importance only when the other major factors of design are fixed. On a constant-factor basis circular airways have a slight advantage over single-compartment rectangular airways, and the latter have a slight advantage over multicompartment airways.

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[27] McElroy, G. E., Economic Size of Metal-Mine Airways: Inf. Circ. 6585, Bureau of Mines, 1932, 21 pp.; Trans. Am. Inst. Min. and Met. Eng., vol. 102, 1932, pp. 59–85.

EFFECT OF DESIGN ON ECONOMIC AREA AND TOTAL YEARLY COSTS

In judging the economic value of different types of openings for use as airways the total yearly operating cost—power cost plus capital charge—is a proper criterion, provided the values assigned the various variables agree with average practice. The selection of a representative set of values is largely a matter of personal judgment and experience and is therefore subject to controversy. However, in order to determine approximately the effect of various variables on area and costs, the author has assumed sets of roughly corresponding values for the various variables and has calculated 3 groups of area and cost data for 100,000 c. f. m. through 1,000 feet of airway; the results are tabulated in table 4. In the first group various types of airways are compared for assumed average values of the variables. They show that, where feasible, unlined airways cost about half as much as lined airways; that circular airways have a small advantage over rectangular airways; and that single compartments have only a slight advantage over multicompartment airways. In determining the relative value of several types of lined airways, where the unit cost of excavation is constant, the relative unit costs assumed for lining practically control the results obtained. The figures therefore hold only for the relative values shown in the table and, as assumed, indicate slight advantages for thin-board lining inside open-timber framing over solid cribbing, for the latter over concrete lining, and for concrete lining over open-timber lining. Small changes in the assumed unit cost of lining in each instance, however, would change the results slightly, but the differences are so small that in any particular case they might easily be offset by safety, operating, or maintenance advantages or disadvantages.

TABLE 4.—*Design factor Z for common types of metal-mine airways for selected values* [1] *of the controlling factors, economic size for flow of 100,000 c. f. m., and yearly costs per 1,000 linear feet of airway*

Type of airway and lining (ratio of longer to shorter side of rectangles 1.25 in all cases) and relative value of factors	Friction factor, k	Unit cost of lining (per cubic foot)	Design factor, Z	For flow of 100,000 c. f. m. (q)					
				Free area, A_m (square feet)	Dimensions (feet)	Velocity (f. p. m.)	Annual expense per 1,000 feet of airway		
							Capital charge	Power cost	Total cost
Average value of factors:									
Unlined circular, in sedimentary rock.	$0.0^{8}6$	------	1.440	74.6	9.75 diameter----	1.341	$5,595	$2,238	$7,833
Unlined rectangular, in sedimentary rock.	$.0^{8}6$	------	1.493	77.3	7.87 by 9.83------	1.293	5,800	2,320	8,120
Unlined circular, in igneous rock.	$.0^{7}15$	------	1.871	96.9	11.11 diameter---	1.032	7,267	2,907	10,174
Unlined rectangular, in igneous rock.	$.0^{7}15$	------	1.940	100.5	8.97 by 11.21-----	.995	7,535	3,014	10,549
Rectangular, open-timber, smooth-lined.	$.0^{8}2$	$0.60	.768	39.8	5.64 by 7.05------	2.515	10,198	4,079	14,277
Rectangular, solid timber-crib lining.	$.0^{8}2$	.80	.742	38.4	5.54 by 6.93------	2.604	11,119	4,447	15,566
Circular, concrete lining--------	$.0^{8}2$	1.50	.670	34.7	6.65 diameter----	2.881	12,601	5,041	17,642
Rectangular, concrete lining------	$.0^{8}2$	1.50	.674	34.9	5.28 by 6.61------	2.866	14,131	5,653	19,784

[1] Low, average, and high values of factors not listed are as follows: Over-all mechanical efficiency, 55, 60, and 65 percent; lining thickness in percent of longer dimension or diameter, 15, 20, and 25; capital return (interest and amortization), 10, 15, and 25 percent; unit cost of power, $50, $100, and $150 per horsepower-year (0.77, 1.53, and 2.31 cents per kilowatt-hour); and unit cost of excavation, $0.25, $0.50, and $0.75 per cubic foot ($6.75, $13.50, and $20.25 per cubic yard).

TABLE 4.—*Design factor Z for common types of metal-mine airways for selected values of the controlling factors, economic size for flow of 100,000 c. f. m., and yearly costs per 1,000 linear feet of airway*—Continued

Type of airway and lining (ratio of longer to shorter side of rectangles 1.25 in all cases) and relative value of factors	Friction factor, k	Unit cost of lining (per cubic foot)	Design factor, Z	For flow of 100,000 c. f. m. (q)					
				Free area, A_m (square feet)	Dimensions (feet)	Velocity (f. p. m.)	Annual expense per 1,000 feet of airway		
							Capital charge	Power cost	Total cost
Average value of factors—Contd.									
3 rectangular compartments, concrete lining.	$0.0^{8}2$	1.50	0.831	43.1	3–3.4 by 4.2	2.322	14,468	5,787	20,255
Rectangular, open-timber lining	$.0^{8}9$	.50	1.203	62.3	7.06 by 8.82	1.606	14,945	5,978	20,923
3 rectangular compartments, open-timber lining.	$.0^{8}9$	.50	1.470	76.2	3–4.5 by 5.61	1.313	15,650	6,260	21,910
Unlined rectangular airway in igneous rock:									
Values of factors selected to give:									
Low capital costs—low power costs.	$.0^{8}9$	------	1.840	95.3	8.74 by 10.92	1.050	2,382	953	3,335
Low capital costs—high power costs.	$.0^{7}21$	------	3.365	174.3	11.81 by 14.76	.574	4,357	1,743	6,100
Average capital costs—average power costs.	$.0^{7}15$	------	1.940	100.5	8.97 by 11.21	.995	7,535	3,014	10,549
High capital costs—low power costs.	$.0^{8}9$	------	1.035	53.6	6.55 by 8.19	1.867	10,045	4,018	14,063
High capital costs—high power costs.	$.0^{7}21$	------	1.892	98.0	8.85 by 11.07	1.021	18,373	7,349	25,722
3 rectangular compartments with open-timber lining:					3 compartments each:				
Values of factors selected to give:									
Low capital costs—low power costs.	$.0^{8}8$	.25	1.648	85.4	4.77 by 5.97	1.171	4,824	1,930	6,754
Low capital costs—high power costs.	$.0^{7}10$	.25	2.522	130.7	5.90 by 7.38	.765	7,385	2,954	10,339
Average capital costs—average power costs.	$.0^{8}9$	.50	1.470	76.2	4.51 by 5.63	1.313	15,650	6,268	21,910
High capital costs—low power costs.	$.0^{8}8$	.75	.835	43.2	3.40 by 4.24	2.313	26,431	10,573	37,004
High capital costs—high power costs.	$.0^{7}10$	.75	1.277	66.2	4.20 by 5.25	1.511	40,442	16,177	56,619

EFFECT OF RELATION BETWEEN COST FACTORS

A much larger variation in possible total yearly costs is shown in the second and third groups of calculated data in table 4, where the effect of varying relations between the factors that influence power cost and those that affect capital charges is shown for 2 widely different types of airways under 5 different combinations of power- and capital-cost factors. Here the personal estimate of maximum, minimum, and average values of factors affects only the range of the results. Unit costs largely determine the total yearly costs, but the area for maximum economy is largely a question of balance between the two sets of factors. Unit costs for ventilation, although necessarily including many other items, cannot help but vary accordingly and therefore do not furnish, by themselves, any direct basis for judging the degree of economical planning and operation of the ventilating system.

ECONOMICAL VELOCITIES

Velocities for maximum economy are also shown in table 4. They are subject to the same variations as the areas for maximum economy, but in the reverse direction. The economical velocity therefore also depends on the particular conditions obtaining and cannot be judged offhand from a small set of sample values.

EFFECT OF QUANTITY ON ECONOMIC AREA

If the effect of all variables other than quantity are combined in a single factor, Z, which is here termed the "design factor", the relation

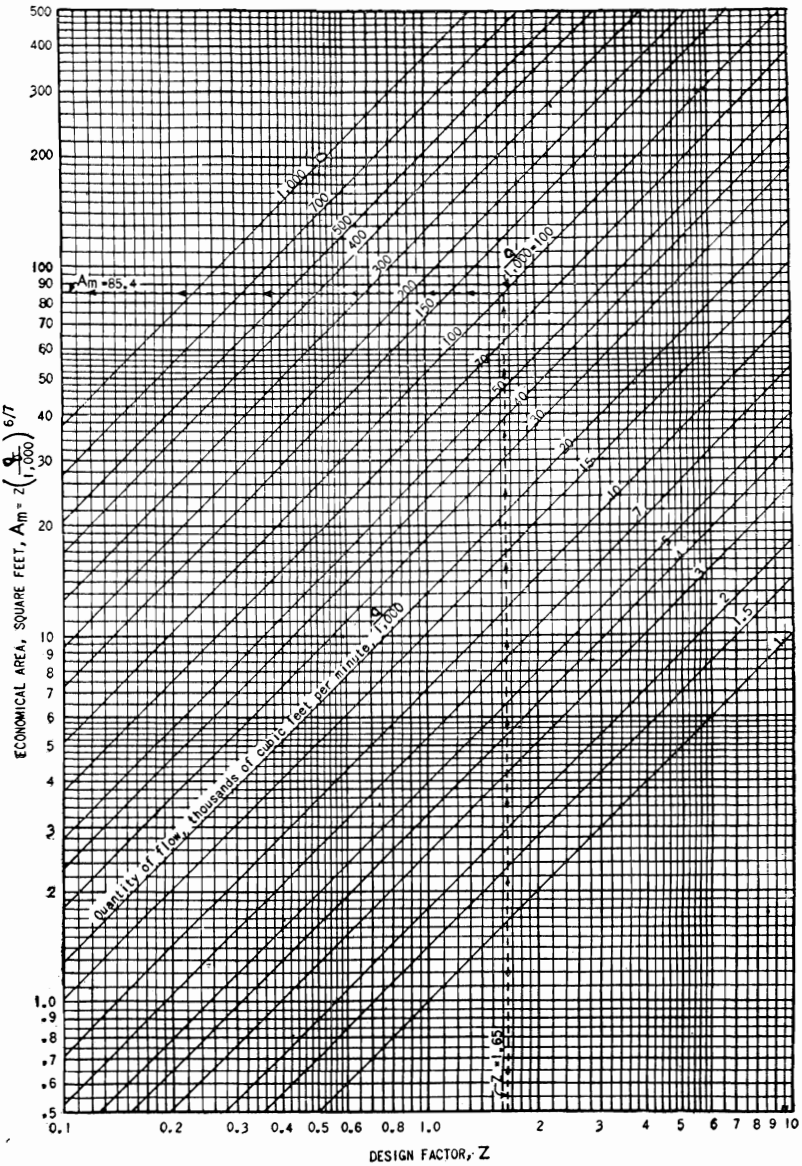


FIGURE 27.—Chart for determining economical area for mine airways from design factor and quantity of flow.

between economic area A_m in square feet and quantity of flow q in cubic feet per minute may be expressed by

$$A_m = Z\left(\frac{q}{1,000}\right)^{\frac{6}{7}}, \tag{34}$$

and the effect of quantity on area may be gaged by reference to figure 27, which is a chart expressing this relation. The solution of an example—A_m=85.4 square feet, when Z=1.65 and q=100,000 c. f. m.—is shown in dotted lines on the chart and indicates the method of procedure.

DESIGN FACTORS

Although design factors are determined by a large number of variables those listed in the first group of table 4 can be used as a rough approximation for general practice wherever there is a semblance of a balanced relation between power-cost factors and capital-cost factors. This is the normal condition; at small mines both sets of cost factors are likely to be high; at moderate-size mines, medium; and at large mines, relatively low.

NATURAL VENTILATION

The pressure differences required to cause flow—pressure gains in a flow circuit as opposed to pressure losses—may be generated by natural or mechanical forces. The most common natural force for generating pressure differences is that due to unequal densities of air columns in and adjacent to the mine openings. The flow resulting from this type of pressure generation is termed "natural ventilation" and the pressures generated, "natural-draft pressures." Other natural forces, such as surface winds and falling water, may be utilized to cause flow but are specially designated in each instance; the general terms refer, by common usage, to flow produced and pressures created by air-density differences. Pressures so generated also act to cause complete flow circuits in undivided dead-end openings, but such air currents are designated as "convection currents" and will be discussed later.

Most small metal mines and many large ones are ventilated entirely by natural means, and natural-draft pressures play an important part in the ventilation of mechanically ventilated mines, the degree of importance increasing with depth and the rate at which the virgin temperature of the rock mass increases with depth.

NATURAL-DRAFT FLOW CONDITIONS

PRESSURES

Natural-draft pressures are differences in total weights of air columns of unit cross-section for the same differences of elevation or vertical height. The differences in vapor contents or average absolute pressures of any two columns under consideration is small, so that the important factors in natural draft are temperature differences and vertical distances.

The columns concerned in natural draft extend into the atmosphere as high as temperature or pressure differences exist, but those for surface openings relatively close together are assumed to be (and usually are) balanced above the level of the uppermost surface opening.

Columns of equal height must always be considered, whether for the mine as a whole, a part of a mine, or a separate circuit under-

ground. Natural draft acts between all connected openings involving vertical differences; and each separate circuit, although it may overlap with others, develops its own natural-draft pressure, which is used up by the pressure losses accompanying flow. Part of the columns considered may be underground and part in surface air. Except for a few special cases, such as a mountainous region like the Coeur d'Alene district of Idaho, the surface components or those parts of the total pressure generated by the columns between the elevations of surface openings are seldom important because they usually act over insignificant vertical distances compared to the underground components.

TEMPERATURE CHANGES OF UNDERGROUND AIR

Although the surface components usually are small the temperature of the surface air is highly important, since it usually has a range

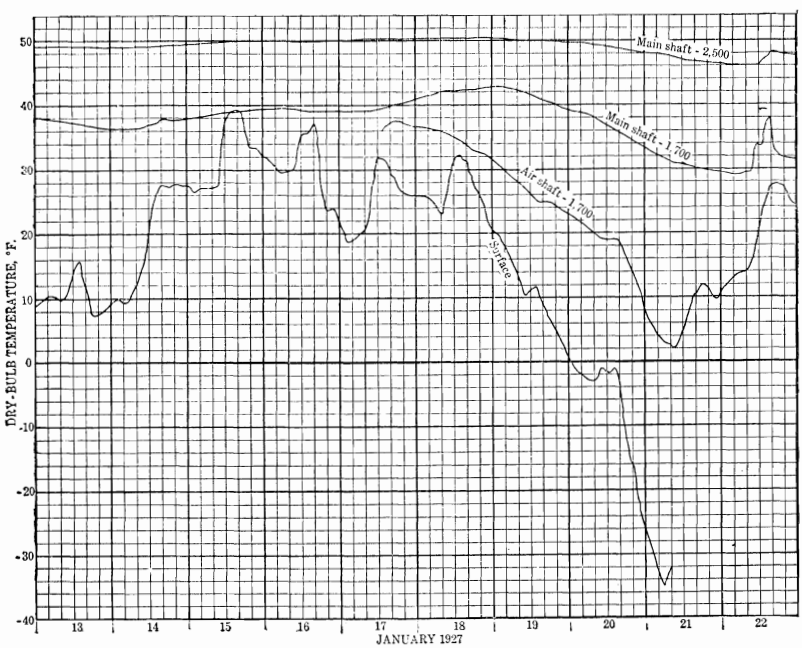


FIGURE 28.—Dry-bulb temperature records at surface and in two adjacent timber-lined downcast air shafts; average velocity, 750 to 850 f. p. m. in main shaft and 1,500 to 1,700 f. p. m. in air shaft.

throughout the year of at least 100° F. and determines the temperature at which air enters the underground openings. The temperature of air flowing underground is changed more or less rapidly by contact with the rock mass (see fig. 28), which acts as a huge heat interchanger. The effect of surface-air temperatures on underground-air temperatures is primarily a question of velocity of movement as governing the time of contact. With very low entrance velocities the effect of variations in surface-air temperatures may be imperceptible only a few hundred feet from the surface, whereas with high-velocity flow, common to mechanical systems, a yearly difference of

as much as one tenth of the surface variation may be noted as far as 3,000 feet below the surface.

DIRECTION OF FLOW

In general, air flows from the colder to the warmer column. In vertical or inclined openings the colder column sinks, while the warmer column rises; thus the opening containing the colder air is a downcast and that containing the warmer air an upcast.

Variations in surface-air temperature produce seasonal changes in the temperature of underground columns, confined largely to the upper parts of the downcast airways, which are large enough normally to reverse the flow seasonally wherever the surface components are large or the mines comparatively shallow. In deep mines surface-air changes usually are not sufficient to reverse the flow, except through cross-connections near the surface; the directions of the main flows remain constant the year round, although the seasonal effect is reflected in the variation in quantities of flow. As depth increases the seasonal effect is diminished because it is exerted over smaller proportional parts of the total length of the air columns involved.

Where conditions are such that reversal occurs, ventilation of the mine or parts of it may be sluggish or entirely lacking for hours, days, or weeks. There are two such periods—late spring and late fall. Often the flow will be reversed daily over a considerable period on account of the difference between day and night surface-air temperatures.

QUANTITIES OF FLOW

As the temperature differences usually are small and variable so also are the natural-draft pressures and the natural-draft flows, although where the openings are large and numerous the small pressures generated often produce ample flows even in large mines. Virgin mine-rock temperatures increase with depth and approximate the average yearly temperature of the air close to the surface. Mine-air temperatures, which are largely affected by rock temperatures, therefore average higher than surface-air temperatures, and larger natural drafts are produced when the surface temperature is below its average for the year than when it is above the average. In a naturally ventilated mine the circulation is better in the winter than in the summer, and in a mechanically ventilated mine the natural-draft pressure acting in series with the primary fan circulation is much larger in winter than in summer.

INTENSITY OF NATURAL-DRAFT PRESSURES

In a shallow mine the natural-draft pressure affecting the total flow usually is a few hundredths to a few tenths of an inch of water, whereas in a deep mechanically ventilated mine it may range from about an inch in the summer to as much as 3 inches of water in the winter.

In a tunnel the 2 columns above the 2 portals may have different temperatures up to the height of the highest intervening surface, and the natural-draft pressures generated, although extremely variable, may be quite large at times. Absolute-pressure measurements at the portal positions of the 6-mile Moffat Tunnel in Colorado showed,

over a period of a year, maximum differences of more than 5 inches of water in winter and 2 inches of water in the opposite direction during the summer season, accompanied by faster rates of variation than those prevailing in mines; both columns were affected by rapidly changing surface-air temperatures on opposite sides of a high mountain.

NATURAL-DRAFT MEASUREMENTS

Differences in total weights of columns of air between equal elevations by definition equal the natural-draft pressure. If a stopping is erected at any point in an airway circuit so that it stops the flow entirely the natural-draft pressure may be measured as the difference of pressure on the two sides of the stopping. If all the conditions that control the weights of the air columns are the same when the stopping is removed and flow resumed the pressure measured is the pressure causing that flow. In many instances the conditions obtaining are sufficiently exact to permit good approximations, since underground temperatures change very slowly, and the measurements may be made when surface temperatures are also changing slowly. It is often impracticable, however, to stop the flow of a mine-ventilating system, in which case the natural-draft pressure can be calculated from observations of temperatures and absolute pressures, usually with greater accuracy.

CALCULATIONS

The weight of 1 cubic foot of moist air may be calculated from formula (4) or determined from figure 7. Temperature conditions are so variable in mines that rigid mathematical solutions for average temperatures and absolute pressures are impracticable. Temperature and flow-pressure changes generally tend to make the density-change curve virtually a straight line, so for practical application it is accurate enough to determine the average density of each column from weighted averages of temperatures and absolute pressures for separate sections of the column, subtract the average densities to find the average difference in weight in pounds per foot of column, and multiply by the number of feet of column to obtain the natural-draft pressure. The result is pressure in pounds per square foot, which may be converted to inches of water by dividing by 5.2. Results by this method can be accurate to within 2 or 3 percent. It is important to note, however, that the average temperature of a variable-temperature column is not one half the sum of the top and bottom temperatures; a fair average for a deep-mine downcast column can be obtained from a few closely spaced observations near the surface, coupled with rather widely spaced observations below the point of maximum surface-temperature effect. As upcast temperatures usually change very slowly, a good average can be obtained from a few widely spaced observations in the absence of large leakage conditions. Absolute-pressure determinations at the same elevation just below the centers of the two columns under consideration are sufficiently close to the average for good approximations.

The effect of differences in absolute pressure and vapor contents of air columns on the natural-draft pressures varies, of course, with the conditions but is rarely more than 5 percent for each in a naturally ventilated mine or more than 10 percent for each in a mechanically ventilated, deep, hot mine. For rapid estimates of such a highly

variable quantity the effect of variations in average absolute pressures and vapor contents in the columns may be neglected and the following formula employed for calculating the approximate natural-draft pressure directly:

$$H_n = \frac{1.325\ BL}{5.2}\left(\frac{1}{T_1} - \frac{1}{T_2}\right) = 0.255\ BL\left(\frac{1}{T_1} - \frac{1}{T_2}\right). \tag{35}$$

where H_n is the natural-draft pressure, inches of water;
B is the average absolute pressure, inches of mercury, obtained approximately by direct measurement near the center of the columns or from a measurement at any elevation corrected for difference in elevation from the center at the rate of 1 inch change per 1,000 feet or more closely if the rate of change is known;
L is the vertical height of the air columns, feet; and
T_1 and T_2 are the average absolute temperatures of the columns, °F., obtained as weighted means according to lengths in the case of variable temperatures.

The intensity of natural-draft pressures may be roughly estimated as about 0.03 inch of water for each 10° F. difference in average temperatures for each 100 feet difference in vertical elevation at standard air density.

The fact that the small difference in elevation between the 2 ends of a connection between vertical or inclined openings forms part of 1 of the columns under consideration may be ignored for almost horizontal connections, which have an allowance for grade only, without perceptible error.

APPLICATION OF NATURAL-DRAFT MEASUREMENTS

Natural-draft measurements are particularly useful when changes in the ventilating system in either a naturally or a mechanically ventilated mine are desired, in which case a more or less close approximation of the resulting changes in quantities and distribution of the flow may be made, depending on the complexity of the distribution system. The pressure losses accompanying flow are the same whether the pressures causing the flow are generated naturally, mechanically, or by both agencies. The flow conditions, however, are determined by the positions of the pressure-generating sources in the flow system. The resistances of separate circuits and separate airways may be determined in any event, and the flow may then be calculated according to the pressure types and positions. In changing from natural to mechanical ventilation it is particularly desirable to determine the resistances [28] to flow by measurement of natural-draft pressures and flows as a check on calculated values; and in attempting to increase quantities of flow in a mechanically ventilated mine it is important to know the intensities of the natural drafts that may operate with or against the fan pressures in causing the primary flow, so that fan speeds and fan horsepowers may be calculated correctly for the new duties, as the application of the so-called "fan laws" only (see following section) is not a complete solution.

[28] McElroy, G. E., and Richardson, A. S., Determining the Air-Flow Resistance of a Small Shaft Mine by Natural Draft: Rept. of Investigations 2890, Bureau of Mines, September 1928, 15 pp.

SIMPLE FLOW CIRCUITS

For a tunnel through a hill, the simplest case, the columns to be considered, as shown in figure 29, *A*, extend from the lowest portal to the top of the intervening surface. Where the vertical distance is small the portal temperatures and the absolute pressure at either portal may be considered as the average temperatures and pressures of the columns. Here the draft is effected entirely by surface components, since both legs of the columns involved are almost wholly in surface air. Where the vertical distance is large and the portals at a distance from each other the data for density calculations are impracticable to obtain, and the best procedure is to measure the absolute pressures at both the portals, add the calculated weight of the column in the tunnel to the pressure at the upper portal, and subtract this result from the pressure reading at the lower portal. If the result is minus, the air will flow from the upper to the lower portal; if plus, from lower to upper. Quantities of flow can be computed readily if the resistances in series are known or can be estimated.

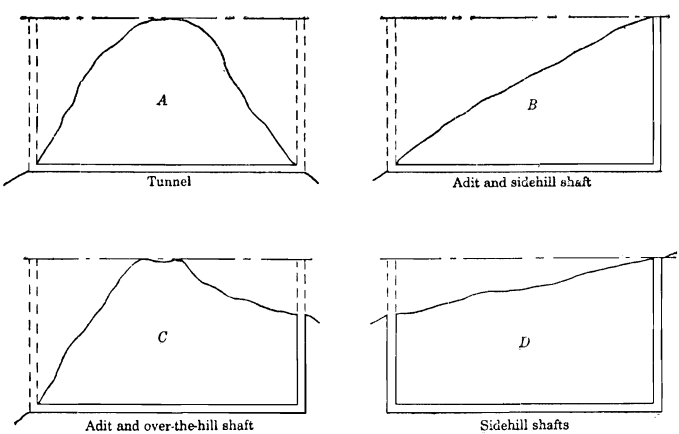


FIGURE 29.—Natural-draft columns for some simple flow circuits.

Without attempting to show small differences for grade, figure 29 shows the columns that act to create the natural-draft pressure differences in some simple flow circuits: *A*, for a tunnel; *B*, for an adit and sidehill shaft; *C*, for an adit on one side of a hill and a shaft on the opposite side; and *D*, two shafts on sloping ground. If the surface-air temperatures were always the same as the underground-air temperatures no flow would result. With the surface temperatures colder than the average mine-air temperatures flow would be in the adit and up the shaft for *B* and *C* and down the shorter shaft and up the longer shaft for *D*. With the surface-air temperature higher than the average mine-air temperature the flow would be just the reverse, or out the adits and shorter shaft.

COMBINED FLOW CIRCUITS

Figure 30, *A*, represents a simple type of combined circuit. The main flow through the adit and the upper part of the inclined shaft is produced by the difference between columns *a* and *b*, whereas the

flow through the inside loop is produced by columns c and d and is independent of the main flow. If columns a and b were balanced so that no flow entered air would still circulate around on the inside loop according to the pressure generated by columns c and d. With circulation on the outer circuit the proportion of intake air circulating on the inner circuit would depend on the conditions at the junction of the adit and shaft, particularly the temperatures of the air currents there. The usual result would be a rather large proportion of intake air circulating on the inside circuit. However, an inherent disadvantage of natural ventilation is that flow conditions are controlled only by the pressures generated, and recirculated air satisfies the conditions just as well as intake air. With a bypass for the air flow on the adit at the junction with the inclined shaft the pressure producing flow in the then single circuit would be the difference between columns $a+c$ and $b+d$, a lay-out that would generally result in much better ventilation.

COMBINED AND SPLIT-FLOW CIRCUITS

Most metal-mine lay-outs are a combination of combined circuits and overlapping or split-flow circuits. Calculations of distribution

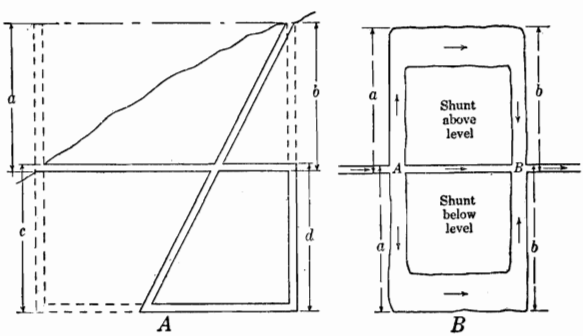


FIGURE 30.—*A*, Natural-draft columns for a simple type of combined flow circuit; *B*, natural-draft columns for shunt types of split-flow circuits.

then require the solving of simultaneous algebraic equations embracing as many equations as unknowns.

The most common type of metal-mine lay-out of airways is a multilevel system similar to that sketched roughly in figure 31, *A*. Here there are four overlapping circuits (*acb*, *adb*, *aeb*, and *afb*) through each of which the sum of the pressure losses occasioned by the necessarily varying quantities in each part of each circuit must balance the pressure generated by differences in weights of the columns (c_1 and c_2, d_1 and d_2, e_1 and e_2, and f_1 and f_2, respectively) above the bottom of each circuit. Calculation of the distribution for this mine, when the separate components of natural-draft pressures and resistance factors for the separate airway sections are known, would involve 5 unknowns—total quantity of flow and separate quantities across each of the 4 horizontal connections. Five algebraic equations would therefore be required; 1 can be obtained by equating quantities of flow—the total flow is the sum of the 4 cross-flows—and the other 4 can be obtained by equating the natural-draft pressure for each circuit to the sum of the pressure losses of the circuit. The solution

for quantities is, of course, tedious, as are all mathematical solutions of distribution for actual airway systems; in practice it is often necessary to omit much detail from consideration and confine the mathematics to the main airway system only.

DISTRIBUTION OF FLOW

With mechanically generated pressure acting at a single point in the airway system the solutions for distribution are comparatively simple, since combined resistance factors may be computed for the mine, or section of a mine, as a whole, and a direct relation for total quantity of flow and fan pressure may be obtained. However, in the naturally ventilated mine just discussed the distribution corre-

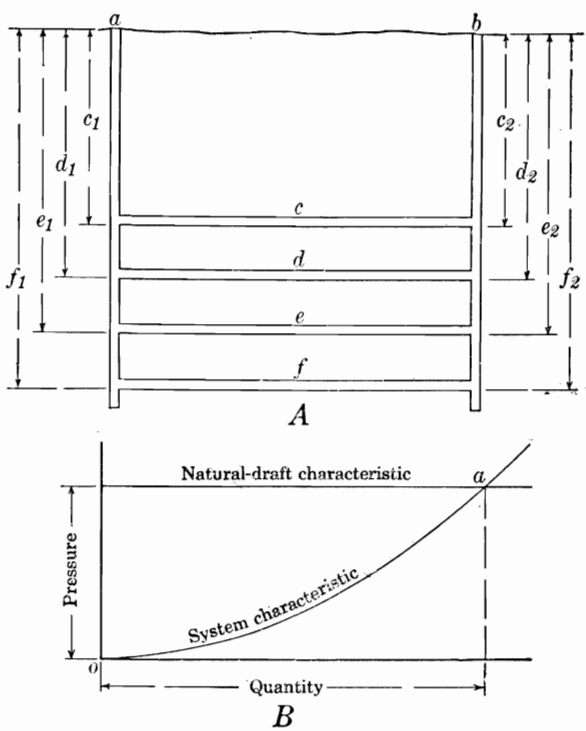


FIGURE 31.—*A*, Natural-draft columns for a multilevel type of split-flow circuit; *B*, pressure-quantity characteristics for natural-draft flow.

sponds to that effected by 4 fans acting in parallel, 1 on each cross-connection, each operating at a pressure equivalent to that generated by natural draft above that level; in general, natural-draft distribution must be considered as similar to that produced by a large number of fans, 1 on every connection between vertical or inclined openings. Since in all mechanically ventilated mines natural-draft pressures are also generated throughout the flow circuit calculation of the exact distribution would require the same type of solution as required for natural draft alone, with the fan pressure added to the circuits in which the fan pressure is generated; in this case the natural-draft pressures act as a group of fans operating in parallel with each other

but in series with the mine fan. In practice, detailed distributions are not calculated, but a procedure is followed that assures fan operation close to maximum efficiency and approximate equality of quantities and major distribution to that desired. The minor distribution is usually a matter of ever-changing detail and as such is solved as accurately as possible by a process of trial and error, guided by experience. The effect of natural-draft pressures on fan performances is discussed in a later section.

RECIRCULATION

The separate components of natural draft do not necessarily generate pressures that cause flows in the same direction; often the pressures oppose each other. Temperature changes are more in evidence near the surface than farther underground, and often the pressures generated above the upper cross-connection (see fig. 31, *A*) are in the opposite direction to those generated below that elevation, with the result that the air is recirculated at times from the return side to the intake side. This, of course, is a seasonal effect but often reaches large proportions in a naturally ventilated mine, even though the resulting temperature changes act automatically to reduce the magnitude of such effects. A more common result is that certain open connections between airways, particularly those in the upper part of a mine, carry large flows normally, yet contain stagnant air over long periods and are often filled with fog.

The intensity of flow or quantity in circulation depends on the relation of pressure gains and pressure losses; if the airways and temperature conditions of an underground circuit are such that a certain amount will circulate, that certain quantity will be circulated regardless of the source of the air. Where the openings to the surface are small compared to the underground openings, as in many metal mines, large quantities of air circulate underground, although only a mere dribble of air may be entering and leaving the mine.

SHUNT-FLOW CIRCUITS

A common condition in metal mines is parallel flow through 2 branches of a split, which has but 1 branch on a horizontal plane, with pressure generated by natural draft on the inclined or vertical branch. This might be termed a "shunt circuit", since one branch leaves and returns to a main airway and may be either above or below it. Temperature conditions in a mine are usually such that the location of the shunt circuit—whether above or below the level—is very important, since the temperature of the air in a main airway is usually lower than the rock in the vicinity, and feeble flows leaving a main flow tend to increase in temperature quite rapidly.

Consider the conditions represented by figure *30*, *B*. Flow of a certain intensity passes direct from *A* to *B* under the action of pressures generated naturally or mechanically on the main circuit. Under uniform temperature conditions, that is, without local generation of natural-draft pressure on the shunt branch, the flow through the branch would be that required to give the same pressure loss from *A* to *B*. But when flow occurs the general result is a warming of the air as it passes through the shunt circuit, which if above the

level generates natural pressure opposing the direction of flow, whereas if below the level generates pressure acting with the direction of flow. For this reason circuits below the level are relatively easy to ventilate, whereas those above the level are relatively hard to ventilate; for good ventilation in workings above a through-airway it is necessary to cut an opening to the level above, increase the resistance of the main airway between the two openings of the shunt circuit, or brattice off the main airway entirely, thereby making the shunt circuit part of the main-airway circuit. The first remedy decreases the total resistance to flow, whereas the latter two increase it and thereby reduce the total flow on the main airway. The particular remedy would be governed by local conditions and the flow required in the shunt. In general, the most satisfactory ventilation conditions for overhand stoping result from mining systems that include an opening to the level above.

Effect of gas-air mixtures.—Small percentages of gases in mine air do not affect natural-draft pressures appreciably through density differences, but the large percentages sometimes encountered locally in mines may affect them and thus the efficacy of the methods of dispersing local accumulations. Light gases, such as methane, act like heated air and are more difficult to remove from shunt circuits above a level than below it, whereas heavy gases, such as carbon dioxide, act like cold air and are more difficult to remove from shunt circuits below the level. Where one column is air and the other a methane-air mixture high natural-draft pressures may be generated by relatively short columns.

CHARACTERISTICS OF NATURAL-DRAFT FLOW

Although natural-draft pressures are not at all constant, particularly where large surface components are involved, they may be considered approximately constant with respect to changes in quantities of flow produced other than by changes in natural drafts, such as a change in fan speed. Such changes in quantities have but a minor effect in changing air temperatures over a short period. Under such conditions natural draft generally is considered a constant pressure-generating source; that is, the pressure remains constant while the quantity changes. An increase in quantity, with the pressure constant, results in an increase in power proportional to the increase of quantity; in a mechanically ventilated mine, therefore, the power supplied gratis by natural draft acting with the fan or that required by adverse natural draft changes approximately as the quantities of flow. The pressure-quantity characteristic for naturally generated pressures is therefore a straight line, whereas that for mine-airway pressure losses is a parabolic curve expressing the relation of pressure varying as the square of the quantity. If both are plotted to scale, as in figure 31, *B*, the resulting quantity of flow will be indicated by the crossing of the two curves, termed "characteristics", as at point *a*.

CONTROL OF NATURAL VENTILATION

The main difficulty with natural ventilation is the relative lack of control over direction of flow, particularly during a mine fire, which

may alter the natural drafts and directions of flow in a very short time. Even in mechanically ventilated mines a fire in a shaft creates such powerful natural drafts as often to reverse the flow against the normal action of the fan or fans.

Natural ventilation is not subject to the same degree of control as mechanical ventilation and is so little understood by the average operator of a naturally ventilated mine that few attempts are ever made to alter existing conditions unless some particularly disagreeable or inconvenient operating trouble results. Poor ventilation often is accepted as a matter of course when a few simple changes in airway conditions or their grouping would effect radical improvements. Where the openings are large and numerous or where the surface components of the natural-draft columns are large ventilation is usually so good that control is unnecessary, except at the time of a mine fire. Where there is timber decay reversal of the direction of flow and recirculated flow often carry the spores of decay all over the mine and largely increase costs of support.

Airway circuits can be changed by opening or closing connections, and air flows thus diverted change the temperature of the air and affect rock temperatures. Such changes in temperature may result from direct mingling of air currents or from velocity changes as affecting the time of contact with the rock. In certain instances the air temperatures may be changed by adding heat locally. Operating shafts are generally natural upcasts because of the little extra heat normally generated in them.

Where there is no extraneous heat production to control the direction of flow the airway in a vertical or inclined circuit that has the smaller cross-section usually will be found to be the upcast, because of the smaller temperature changes that accompany the higher-velocity flow. Once any shaft becomes an upcast or downcast, it tends to maintain that direction of flow since the downcast air averages colder than the rock mass and the upcast air warmer. For this reason the direction of flow sometimes may be mechanically changed in a shaft long enough to change the natural direction of flow permanently, particularly where the surface component is small and the summer heating effect on the intake air can be confined to a short vertical distance from the surface through low-velocity flow. However, surface-air temperature changes often reverse the direction of flow. If the surface columns are large such reversals cannot be prevented, but if they are small the tendency toward reversal usually can be checked by proper manipulation of the air currents so as to change the existing distribution of temperatures and natural-draft generation.

If the surface components are large and the flow is reversed seasonally attempts sometimes are made to determine the temperature of the surface air at which reversal will take place. Where natural draft is due almost wholly to surface components and the flow circuits underground are quite stable a close determination (within 1° or 2°) may possibly be made; however, where large underground components are also involved reversal is the effect of average temperatures over a period of time rather than of instantaneous surface temperatures, and variations of as much as 10° in the surface-air temperature at the time of reversal may be noted.

CONTROL OF NATURAL-DRAFT PRESSURES IN MECHANICALLY VENTILATED MINES

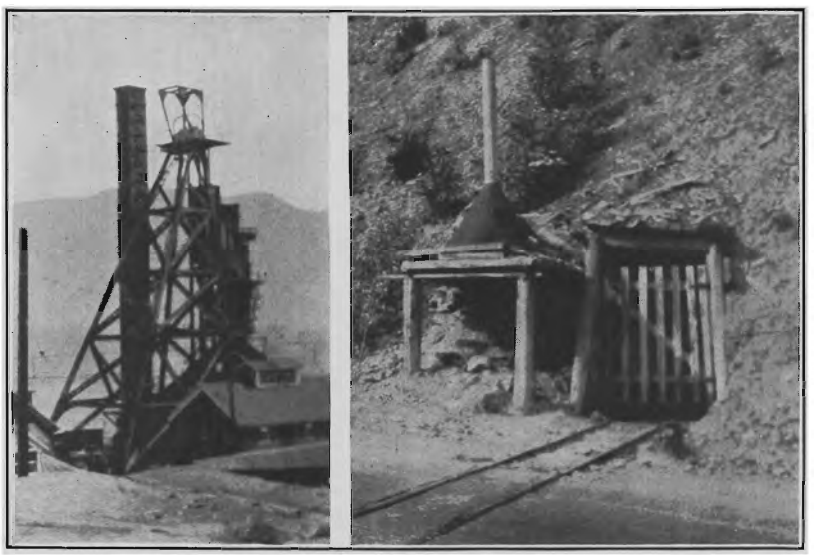
In mechanically ventilated mines also many opportunities—which are often neglected—are presented for improving natural-draft pressures, such as preventing leakage of warm return air into a cold intake and deflecting warm currents from local heating sources direct to return airways. In high-velocity downcasts the air usually is colder at depth than the surrounding rock but often is warmed considerably by circulating at low velocities in dead-end openings off the shaft. In a deep, mechanically ventilated mine the natural-draft pressure always operates with the fan pressure; conserving the temperature differences between upcast and downcast not only increases the horsepower supplied gratis by natural draft but also improves circulations in the lower part of the mine by increasing the natural pressures generated there.

In mechanically ventilated mines many sections that are off what might be termed the "fan-pressure loop" or main airway circuit are ventilated mainly by natural draft, because the natural-draft pressures are much larger than the acting fan-pressure components; and seasonal reversals are common in parts of fan-ventilated mines. Many of the somewhat perplexing effects noted in the distribution of air in mechanically ventilated mines are purely natural-draft effects, but as the average metal-mine operator considers natural draft merely a question of differences of surface elevations he is at a loss to account for them. A few of the more common conditions encountered are: The extension of a multicompartment downcast shaft for several hundred feet below the lowest connection is usually well-ventilated by natural draft, and provisions for auxiliary fan-pipe ventilation are generally discarded; the compartment in a downcast shaft that contains warm-pipe columns often has little or no velocity of flow and sometimes has upcast flow even though the flow in the adjoining compartments, to which there are many openings, is downcast at high velocities; shafts between the main-upcast and main-downcast shafts often carry but little flow, and a constant direction of flow is maintained in many instances only by occasionally applying emergency measures which may fail to prevent reversal of flow; and that part of a downcast shaft that extends above an intake adit may reverse with the seasons.

ARTIFICIAL AIDS TO NATURAL DRAFT

STACKS

A number of methods are available for controlling or increasing natural-draft circulation and providing circulation in special instances when natural or mechanical circulation is absent. One of the simplest ways of increasing the natural-draft pressure is to increase the length of the acting columns by an actual increase in length of one of the pair, as by adding a stack or chimney to the return-air shaft. Several examples of these, similar to that shown in figure 32, *A*, may still be seen in the metal-mining districts of this country. Their efficacy depends on the relative increase in length of the acting columns; as installed in deep mines, they rarely increase the flows perceptibly and are practically useless. Attached to a line of tubing extending



A *B*

FIGURE 32.—*A*, Stack on metal-mine shaft; *B*, stove used with tubing to ventilate dead-end prospect adit.

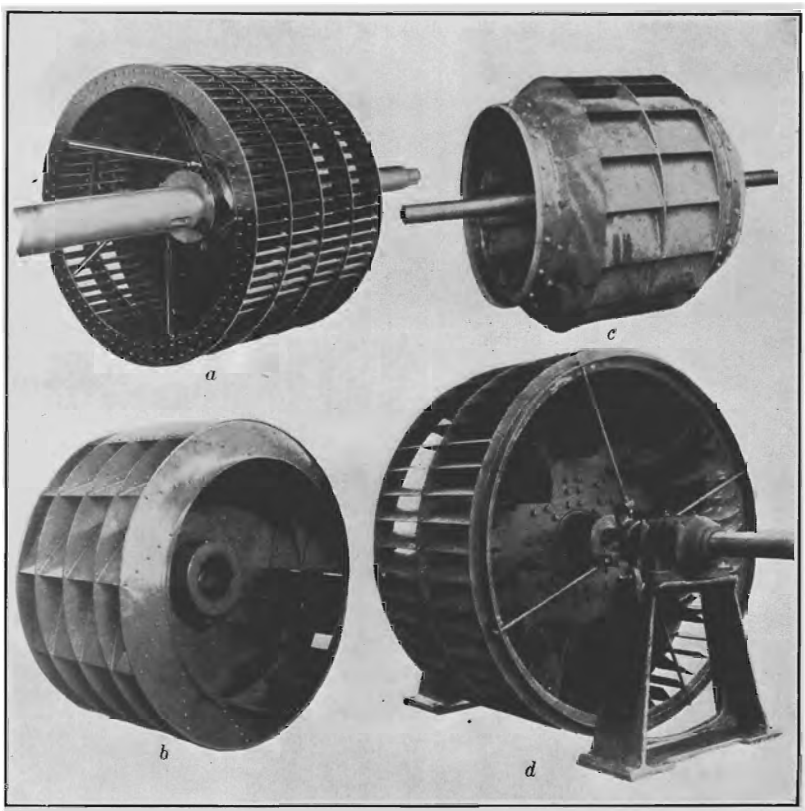


FIGURE 33.—Multiblade rotors of centrifugal fans of types commonly used for ventilating western metal mines: *a*, American Sirocco (forward-curved blades), American Blower Co.; *b*, American High Speed (backward-curved blades), American Blower Co.; *c*, Safeload (backward-curved blades), Jeffrey Manufacturing Co.; *d*, Stepped Multiblade (stepped forward-curved blades), Jeffrey Manufacturing Co.

to the face of a dead-end opening, they might at times give quite effective circulation, but if in surface air there would be seasonal reversals and quiescent periods. Some stacks are attached to fan-tubing pressure units, used to ventilate moderately deep prospect shaft openings affected by strata gas, as an emergency aid to natural draft should power or fan fail; with stacks about 20 feet high and shafts 500 to 1,000 feet deep it is doubtful if the stack is of any material assistance once the tubing becomes upcast, as it practically always will in time, but in winter the stack probably reduces to some extent the time required to reverse the flow in the tubing.

FURNACES

Before the introduction of fans, with their relatively high efficiency in the use of energy, furnaces were often used to increase the temperature of the upcast artificially where fuel was cheap, as at coal mines, and stacks were sometimes added to the top of the upcast air shaft to increase the pressure generated. The furnace was usually placed at or near the bottom of the upcast shaft. The method has many elements of danger and has been discarded in favor of fan ventilation.

A variation of this method is occasionally used to ventilate dead-end prospect openings in metal-mine districts not easily accessible or supplied with power. This consists of a small stove, usually of the Selby or cone type, with a short stack (fig. 32, *B*), from which a line of tubing extends to the faces of underground openings 500 to 1,000 feet away. Wood is burned in the stove after a face is blasted underground, and enough draft is produced by the short stack to clear the workings of blasting gases in a few hours; operations that would be confined to 1 shift with diffusion alone acting to disperse the blasting fumes are thus able to work 2 shifts and enjoy air less contaminated with powder fumes than would otherwise be the case.

AIR, WATER, AND STEAM JETS AND FALLING WATER

Air, water, and steam jets often are employed to provide circulation of air and may be used under certain circumstances to control natural-draft circulation. They act naturally by changing air temperatures and thus natural drafts; they also act mechanically in that the mechanically produced velocity of discharge is equivalent to generation of pressure. They usually are pointed in the direction of flow in order to act with, rather than against, the other pressures causing flow and are occasionally useful in an emergency in mine ventilation, particularly in the control of direction of natural-draft flow, as at times a very small additional force will prevent reversal of flow.

Falling water may act in several ways to help or hinder flow. The temperature of the water may alter the temperature of the air and thus change the natural drafts. The velocity of fall creates energy which, through the resistance of the air to its passage, generates pressure in the direction of fall; if the water falls in a pipe the same generation of pressure may serve to entrain air mechanically and carry it to lower levels, as in the once-used but now discarded method of the trompe. All such ventilation methods were important means of ventilating dead-end prospect openings and even mines before the

introduction of mechanical ventilation, which is so much more efficient that there is no comparison. They are now useful as emergency measures and for the ventilation of out-of-the-way prospect openings; as such, their importance and possible uses are so limited that no attempt will be made to present mathematical treatment.

EFFECT OF WIND

Wind is energy in the form of velocity and represents pressure exerted in the direction of flow. The actual effect of surface winds on the ventilation of mine openings depends largely on how the directions of movement coincide. Direction of surface wind with respect to direction of mine openings not only determines the component of the wind pressure causing flow but also affects the pressure losses during flow. The effects are so variable and usually so small compared to natural draft and mechanical draft that they are important only in the absence of these sources of pressure generation. Cross-winds at air outlets create pressure losses that affect the flow by adding a deflection of the air stream at the point of discharge. Deflectors may be used on discharges to direct the discharge in the direction of the wind and make use of the suction effect of the latter, as illustrated by many types of roof ventilators applied to small buildings. Winds denote a lack of equality of absolute pressures on the surface and often are given full credit for ventilation of tunnels through hills on which their effect may be negligible. Where surface winds are steady and strong, deflectors might be added to mine-air inlets or outlets to advantage to obtain the maximum useful effect of wind pressure, but such devices are cumbersome when applied to large openings and, unless properly made with a large-radius turn, would add a constant pressure loss greater than the intermittent loss they were designed to obviate.

CONVECTION CURRENTS

Natural-draft currents induced by air-density differences due to temperature differences are termed "convection currents" when the air circulates in a dead-end opening without separate passages for intake and return. Such currents are responsible for the distribution of warm air from a warm-air outlet or from heated surfaces to all open parts of a room; they are also responsible for the good ventilation of relatively short-end openings in a mine.

The temperature of the air flow passing the end of a horizontal dead-end opening usually is at least a few degrees different from that of the rock surfaces that control the air temperature in the dead end, and the small natural pressure generated by this temperature difference acting over the height of the airway must be used up as pressure losses occurring during flow. The result is a circulation without definite boundaries; the warmer air circulates along the roof and the colder air along the floor. If the temperature differences between the two ends of a dead-end opening are large, as where a cold intake current passes the open end of a development drift in hot rock, the flows along roof and floor sometimes reach velocities of 100 to 200 f. p. m. at the open end but gradually diminish toward the face, partly on account of temperature changes but largely on account of the change in elevation due to grade, since it has been observed that the bottom of the upper flow is a level surface. This latter condition,

evidenced by the level lines on the walls marking the extent of the wetted surfaces where the return current along the roof cools and deposits condensed water on the walls, apparently limits the distance to which such currents can extend as the point at which the roof has the same elevation as the floor at the open end. The temperature of the air passing the open end, not its quantity, is the controlling factor in such circulation. As an extreme example of ventilation of this type, the temperature of the air at the face of a 5- by 7-foot crosscut in 115° F. rock, 120 feet from an airway carrying air saturated at 66°, was 90° saturated, and the velocities of the opposing flows along the roof and the floor, although not measured, were very perceptible. In another mine, where a cold intake current at 42° passed the open end of a 1,500-foot development crosscut in 85° rock, the lower level of the return current along the roof was marked by a level line intersecting the floor line at a point about 1,000 feet in.

Dead-end vertical or inclined openings show convection currents less often, but they are usually present to some degree, particularly where water runs down the walls or where only part of the cross-section at the open end is in contact with a markedly different air temperature than the remaining part of the section.

MECHANICAL VENTILATION

As natural ventilation of mines is generally inadequate and unreliable and artificial aids to natural drafts inefficient and not always applicable, miners have been interested since early times in the development of machines for creating the pressure differences required for adequate and controllable flow of air through mine openings; most of the machines now in use for this purpose are based on designs originally developed for mine ventilation. All of the more ancient machines were of the positive displacement type—bellows, rotating paddle wheels, and reciprocating compressors. They had small capacities and were used in conjunction with box conduits extending from the surface to the working places. Their application to circulating air through mine openings resulted in such huge, cumbrous forms that other types of mechanical ventilators were sought. These generally took the form of a rotating drum on the rim of which different forms of plates were mounted; this type developed into the form now known as the centrifugal fan. Although other types of ventilators, such as air screws and rotary displacement blowers, are used to a limited extent in mine ventilation, almost all of the large machines as well as the great majority of the smaller ones in use are centrifugal fans. Mechanical ventilation therefore refers almost always to the production of air currents in mines by pressure differences produced at particular points in the air circuits by centrifugal fans, acting in conjunction with natural-draft pressures generated throughout the air circuits. Most American coal mines and approximately half of the metal mines are ventilated mechanically. The performance of centrifugal fans should therefore be a subject of major interest to mine operating officials, yet the essentials are known to comparatively few of them, misconceptions are prevalent owing to a rather confused literature, and sales representatives are not always familiar enough with basic facts to combat arguments. Arguments between operators and manufacturers regarding fan performance are common, due more

to the operator's misunderstanding of fan performance than to misrepresentation by the manufacturer.

CENTRIFUGAL FANS

The centrifugal fan is a low-pressure large-quantity air pump. It acts by producing a difference in absolute total pressure between its inlet and outlet. This pressure difference is generated by blades set in a drum-shaped wheel rotating in a scroll-shaped housing. Rotation of the blades, according to present conceptions, generates static pressure as tension is generated in a string by which an object is whirled through the air, and velocity pressure develops at the blade tips as a result of the centrifugal and rotative velocities of the air passing through the fan; and part of the velocity pressure is converted concurrently to static pressure in the scroll-shaped housing. The full application of knowledge of air flow resulting from recent aeronautical studies probably will modify the concept of pressure development by centrifugal fans. The mine operator or mine-ventilating engineer is not concerned with how the pressure is developed; his major concern, under present-day conditions, is in its proper utilization, a subject complex enough to engage his best efforts. When fans were constructed on the job to rule-of-thumb design each operator necessarily interested himself in design, but with efficient designs manufactured by specialists available his primary interest should be in correctly ascertaining the duty of the fan when applied to his particular conditions, so that efficient service will result.

MODERN TYPES

Virtually all modern designs of centrifugal fans are of the multiblade type; that is, the rotors or wheels have a large number of comparatively shallow blades and differ mainly with respect to shape and position of the blades. Although the development of types having radial, forward-curved, backward-curved, and composite-curved blades has advanced simultaneously in recent years the forward-curved multiblade type, on account of its much greater capacity for equal size and speed, has been the favorite for both coal- and metal-mine ventilation. All of the fans installed for primary ventilation of metal mines in this country, except a few very recent installations, have been of this type. Recent developments in centrifugal fans have been governed mainly by commercial rather than mine application; most standard designs are therefore made for direct attachment to ducts, with no arrangement for gradual expansion in area except that provided in the scroll-shaped housing. For fans made to discharge direct to the atmosphere a discharge section of gradually enlarging area, termed an "evasé", is usually attached to the standard design, although this feature is incorporated in some designs.

Within the last 5 years certain advantages of the backward-curved multiblade type for mine ventilation have come to be more generally recognized, with the result that there has been a spurt in the development of designs of this type. A few of these are now in service for primary ventilation of coal mines, a few others have been considered and possibly installed for primary ventilation of metal mines, and a large number of the smaller sizes are in use for secondary and auxiliary service in both.

Most of the mechanically ventilated metal mines in this country are widely separated in the West, and there is no great incentive for the manufacturers of the East to compete for their fan business, except where several mines are grouped. As a result, the number of types represented in main-fan service is virtually confined to three makes, with rotors of the general designs shown in figure 33. A few other makes and types of fans are represented by those of smaller capacity used in secondary service and still more in the small units used for auxiliary service.

SERVICE DESIGNATIONS

The three common designations of type of service required of a fan installation relate to the common division of the air currents of the distribution system into main, secondary, and auxiliary circuits. Thus, a fan installed to handle the entire flow of the mine or of one of the major circuits is a "main" fan, one installed to aid distribution in a section of the workings only is termed a "secondary" fan, and one used with tubing to ventilate dead-end openings is an "auxiliary" fan. Metal-mine distribution systems are sometimes so complex that the distinction between primary and secondary fans is difficult when the fans are operating in series-parallel combinations on separate intakes or returns, and as secondary fans usually operate in the "booster" position they are referred to more commonly as "booster" fans.

POSITION DESIGNATIONS

The position of the fan with reference to the particular point in the airway circuit at which it generates a pressure difference determines the familiar trade designations—considered by many to include differences in design or at least in performance—such as "pressure fan" or "blower" at the intake end, "exhaust fan" or "suction fan" at the discharge end, and "booster fan" for intermediate positions. The position of the fan in a circuit determines the difference in pressure between any particular section of the circuit and the adjacent atmosphere but, within the limits of accuracy involved in the application of fans, does not affect the pressure generated[29] by the fan or, provided the direction of flow is the same, the pressure losses due to flow. These pressure relations are discussed in a later section.

FAN PERFORMANCE

As shown, the pressure-quantity relation for pressure loss in an airway or a system of airways may be expressed by the simple equation $H=RQ^2$ accurately enough for practical application. When R is known plotting the characteristic of the airway or the curve showing this relation of pressure loss to quantity of flow is simple; but in the case of the fan a variable amount of the pressure actually generated is used as pressure loss due to flow through the fan casing, and there is no simple mathematical relation between the net pressure generated and the quantity of flow. The pressure-quantity relation for a fan must therefore be determined by actual tests, and the results of the tests should be plotted in the form of a curve. This curve is termed a "characteristic" curve of the fan.

[29] McElroy, G. E., and Richardson, A. S., Experiments on Mine-Fan Performance: Tech. Paper 447, Bureau of Mines, 1929, 61 pp.

CHARACTERISTIC CURVES

Although the relation of total pressure difference generated to quantity of flow is the basic relation that determines what will happen when the fan is applied to the airway system and is thus the basic characteristic of the fan a number of other conditions that relate to the particular performance of the fan are interesting—changes in mechanical efficiency, power requirements, and relative proportions of pressure components, static and velocity, that accompany changes in this basic relation. These are also determined experimentally and, when plotted to correspond to the pressure-quantity characteristic, likewise specify the performance of the fan. Since these are also termed "characteristic" curves the separate curves are commonly designated according to their origin, and the basic characteristic is termed the "fan" or, more commonly, the "total-pressure" characteristic. Such a simple set, specifying the performance of a particular

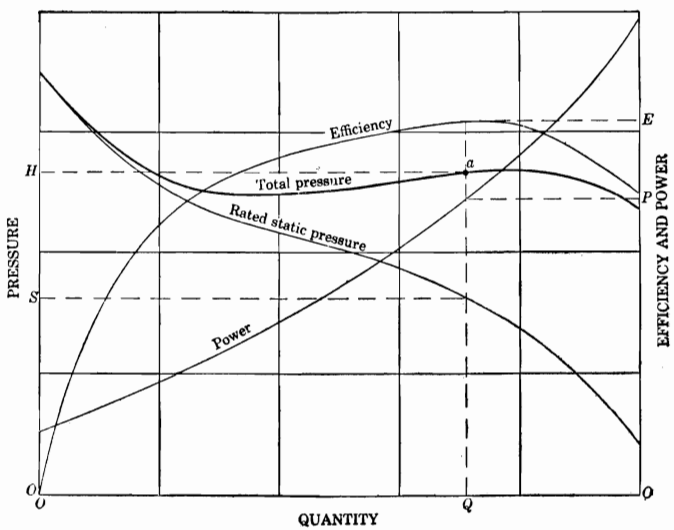


FIGURE 34.—Characteristic curves of fan performance at constant speed and constant air density.

fan at a certain constant speed, is shown in figure 34. The particular point, such as *a* on the total-pressure curve, at which the fan will operate is determined by the similar total-pressure curve for the system on which it operates, and this point in turn determines the other conditions of performance of the fan. Point of operation *a* (fig. 34) determines the scale values of the quantity as *Q* and the total pressure as *H*. Since characteristic curves for other conditions of performance are plotted to the same quantity base, their scale values are determined by the intersection of a vertical line through *a* with their respective curves, which determines the efficiency as *E*, the power as *P*, and the rated static pressure as *S*.

TESTS

Characteristic performance curves for fans usually are determined by laboratory fan tests conducted according to an exact procedure

such as that laid down in the Standard Test Code of the National Association of Fan Manufacturers and the American Society of Heating and Ventilating Engineers.[30] Five sets of measurements are required—speed, density, quantity, pressure, and power. To cover the full range of performance these measurements must be repeated, with the speed practically constant, for eight or more different conditions of resistance to the ventilating system. In the laboratory tests a short duct having the same dimensions as the discharge or inlet connection of the fan is added to the fan, and the resistance to flow is varied from the fully closed to the wide-open condition by varying the amount of resistance added at the discharge or inlet end of the test duct. Characteristic curves or other performance data are not always available for the fans in use at a mine, and it is often desirable to test fans in place to determine the approximate characteristics or, more commonly, to obtain data on the operating condition and on the mine resistance.

LAWS

It would obviously be impracticable for manufacturers to determine the characteristic performance curves of all types of fans for all sizes and for all possible speeds by tests. Fortunately, this procedure is not required, since the changes in performance have been found by experiment to obey certain laws with enough accuracy to permit their general use over a large range of variations in conditions of flow.

For a fan operating alone on a given ventilation system the following rules govern the performance of the fan and are useful in correcting test results to constant-speed, standard-density conditions:

1. The quantity of flow varies directly as the fan speed and is independent of the air density.
2. The pressures produced by the fan vary directly as the square of the fan speed and directly as the air density.
3. The power input to a fan varies directly as the cube of the fan speed and directly as the air density.
4. The mechanical efficiency of a fan is independent of the fan speed and of the air density.

The performance of similar fans, that is, fans made to the same design with all dimensions proportional, is based on the fact that within a limited range of sizes fans operated to have the same linear velocity at the tips of the blades, referred to ordinarily as the "tip speed", will produce equal pressures when the quantities of flow vary as the square of the outside diameters of the fan wheels or rotors.

Fan-performance characteristics for other speeds and sizes can therefore be computed from one set of constant-speed tests against variable resistance conditions. Actually, a very slight increase in efficiency accompanies increase in speed—so small that it is ignored. Also, fan performance differs slightly, depending on whether the fan inlet takes air at zero velocity, as in the blower position, or whether it takes air having a definite velocity, as in the exhaust or booster positions, since the shock pressure loss in the fan entrance is larger for the first condition. The difference is so small, however, that it can be neglected with coned or rounded inlets. A slight increase in efficiency also accompanies increase in size of fan; this is sufficiently

[30] Journal of American Society of Heating and Ventilating Engineers, Standard Code for the Testing of Centrifugal and Disk Fans: Vol. 29, 1923, pp. 371-380 and 438.

marked for the smaller sizes as to require separate characteristics for separate size groups. The larger sizes used for main-fan service usually can be considered as in the same size group without appreciable error.

PRESSURES AND RATINGS

The subject of fan pressures and fan ratings is confusing because the pressure differences measured directly against atmospheric pressure do not correspond directly to the total-pressure difference generated by the fan. In an attempt to avoid confusion most manufacturers rate their fans on a static-pressure basis, which is equivalent to rating the fan for a particular condition of installation seldom encountered in mine-ventilation practice except in auxiliary service—fan in blower position discharging into a duct of uniform area the same as that of the fan-discharge connection. When the air density at the fan inlet is standard the static pressure in such a duct at the fan discharge is the pressure rating given to the fan. The velocity pressure at the discharge of the fan under these conditions is equivalent to the shock pressure loss at the discharge end of a duct of uniform area and is therefore disregarded, as it is not available for overcoming the pressure losses in the duct. Present concepts of fan pressures and fan ratings are really based on this concept of a simple duct system; to take care of the widely different conditions met in practice various artificial concepts are introduced so that accurate results may be obtained by those conversant with their origin and relation to basic facts, while the average run-of-mine operator is left in a fog. Fan performance has therefore come to be based almost entirely on static-pressure measurements, which have no direct relation to fan performances but which, in general, approximate them closely enough to obviate serious errors in the great majority of cases, inasmuch as the velocity pressures involved in the usual mine-fan installation are small. Attempts to obtain accurate results by artificial concepts are more difficult, in the author's opinion, than a procedure based on the facts of pressure gains and pressure losses (p. 35); the latter course is therefore recommended and the only one outlined in any detail throughout this bulletin.

A static-pressure rating of a fan refers to a definite area and operating position. Ordinarily, it is based on the area of the discharge connection provided and the blower position. Fans for use only in the exhaust position are sometimes given static-pressure ratings based on the area of the inlet connection; in this case, although the ratings may refer to a fan with free discharge, they usually refer to a fan equipped with an evasé discharge section as an integral part of the standard design.

PRESSURE GRAPHS

An easy way to avoid confusion in specific cases, particularly in correlating field test results to fan ratings, is to make a graph of the pressure changes. Working from atmospheric pressure as a base but remembering that all pressure changes are changes in absolute pressures, one must consider five general facts to avoid confusion: (1) The difference between the average total pressure at the inlet and discharge of a fan is the only constant value for all operating positions and the value to be used in determining the rated static pressure, which customarily is one discharge velocity pressure less than this value; (2) the

average total pressure at any cross-section of the ventilating system is the algebraic sum of two mutually convertible components, static pressure and average velocity pressure, whose relative proportions depend only on the area of the cross-section; (3) average total pressures and static pressures may be positive or negative referred to the atmospheric pressure as a base, but velocity pressure must always be considered a positive pressure; (4) the absolute value of the average total pressure always decreases in the direction of flow; and (5) a fan always increases the absolute value of the average total pressure in the direction of flow.

The relations of fan and ventilating-system pressures to atmospheric pressure are shown graphically in figure 35, *A*, *B*, and *C*, for three different positions of the fan in the same ventilating system, assuming uniform resistance conditions and velocities throughout the system. In each position the total pressure developed by the fan is the same, and the fan static-pressure rating is equivalent to the fan total pressure minus the velocity pressure at the discharge; that is, it

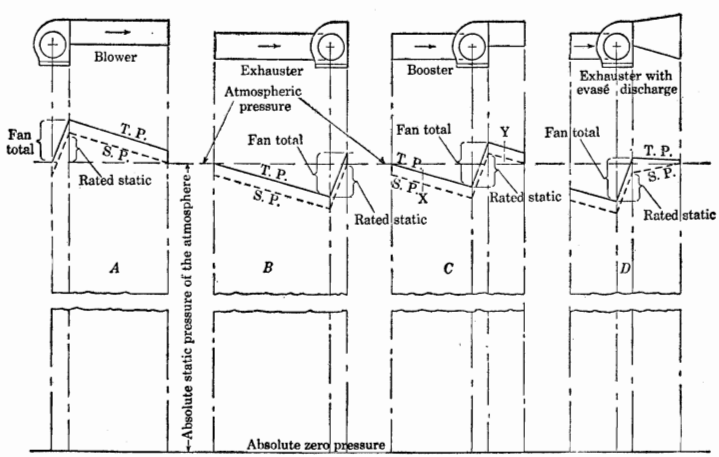


FIGURE 35.—Graphs of pressure changes in ventilating systems as affected by position of fan with respect to system.

is based on the discharge area of the fan operating in the blower position. For any point in the ventilating system static pressure (S. P.) is equal to the algebraic difference between the total pressure (T. P.) and velocity pressure (V. P.).

As shown in figure 35, *B*, the negative total pressure at the inlet for the exhaust position is equal to the static-pressure rating (blower position) and, as long as the fan discharges to the atmosphere, will remain constant regardless of changes in area of the duct at the inlet. It is therefore advocated and used as an alternative rating for a fan in the exhaust position. One-point total-pressure measurements can be made, but the average total pressure over the section is that involved and can be obtained directly only by a traverse and indirectly from the measured static pressure and the calculated mean velocity pressure. However, some investigators consider one-point total-pressure measurements correct, whereas others are concerned with determining a position in the duct that will give the average total pressure. A position that will give a good approximation of the

average total pressure can be determined for conditions of uniform area but can rarely be applied at mine-fan installations. Although the actual values of the pressures are the same the use of two such totally different pressures as rating pressures for different positions leads to confusion, which is increased by the fact that still another method of determining the rating must be used for a fan in the booster position. Facts are therefore more reliable: The position of operation is immaterial; the true performance is the total pressure difference generated; the rated static pressure is an arbitrarily determined figure that has a direct relation to the pressure generated; and fan performance is not necessarily directly related to the pressures measured in the ducts adjacent to the fan but must be determined from measurements made according to position of fan, duct areas, and, if they are involved, corrections for pressure losses between points of measurement and the inlet and discharge of the fan.

EVASÉ DISCHARGE

The effect of an evasé, a section of gradually expanding area added to the discharge of the fan proper, on a fan operating in the exhaust position is shown graphically in figure 35, *D*. The evasé reduces the total resistance of the ventilating system by reducing the shock pressure loss of the flow system on discharge to the atmosphere. If the evasé is considered a part of the fan then the true total pressure rating of the fan is reduced by the amount of pressure lost in the evasé and the calculated efficiency of the fan likewise, although the pressure produced by it is thus used more efficiently. The evasé discharge section is sometimes a component part of the fan but usually is attached to standard fans by either the manufacturer or the mine operator, and performance must be correlated with the ratings for standard fans without evasé sections. For this reason the evasé section is more properly considered a part of the ventilating system. The burden of using the velocity component efficiently is thus thrown on the operator, who can best provide the proper airway and duct condition. Special types of fans, however, may have such a normally high velocity of discharge that efficient use of the velocity component is a practical necessity. The evasé is then made a component part of the fan design, and the fan is always so rated.

PRESSURES UNDERGROUND

Fans operating underground are always in the booster position, with resistance to flow on both inlet and discharge sides. Auxiliary fans connected to tubing have only a negligible resistance to flow on the side open to the mine airways, so this resistance is not considered; the position is considered as determined by the resistance to flow in the tubing only, and the adjacent atmosphere is used as the pressure base. Where the flow for a fan in the booster position underground is through mine airways, rather than separate ducts or tubing, the graph of the pressure changes is the same as that shown in figure 35, *C*, but the atmospheric pressure is not available as a base or datum pressure. The conditions are the same as for a fan on the surface, but the pressures must be measured entirely within the ducts of the ventilating system. With the atmospheric pressure line omitted the graph is actually a graph of the absolute-pressure changes, and the basic measurement is the difference in static pressure (D. S.

P.) between a point close to the inlet, such as X, and a point close to the discharge, such as Y. If the quantity of flow and the areas at X, Y, fan inlet and fan outlet are known, velocity pressures can be computed and the graph followed to determine the fan total pressure and the rated static pressure.

MECHANICAL EFFICIENCY

All of the energy supplied to a fan does not result in useful work. A certain proportion, which varies with the resistance against which the fan operates, is required for flow through the fan itself; the rest represents useful work, the horsepower in the air. The useful work per unit of time divided by power input for the same unit of time is the mechanical efficiency of the fan. The useful work is obtained from the total pressure of the fan and the quantity of flow. The term "mechanical efficiency" in data on fan performance refers to efficiency based on the total pressure, even though the pressure rating of the fan is given on a static-pressure basis. Occasionally, for convenience of application with rated static pressures, an efficiency termed the "static efficiency", or efficiency calculated from the rated static pressure, is given. This bears the same ratio to efficiency that rated static pressure does to total pressure and can thus be used directly with rated static pressures to calculate power requirements.

APPLICATION OF FANS TO VENTILATING SYSTEMS

Fans may act singly or in combination with other pressure-generating sources such as natural draft or other fans. In a metal-mine ventilating system there are natural-draft pressures acting in series with or against the fan pressure in most instances, and often there are two or more fans acting in series, parallel, or series-parallel combinations with each other and natural-draft pressures.

When a fan is the only pressure source acting on a ventilation system the duty required of it is constant and is controlled entirely by the resistance of the system; but when a fan operates in combination with other pressure-generating sources all combine to overcome the resistances of the system, and the duty of a fan operating at a particular point in the system is determined by the position and intensity of the other pressure sources acting with it to produce the flow. Under such circumstances the duty of the fan is not constant but variable, as determined by the proportion it contributes to the total pressure generated throughout the system which varies with fan speeds and natural-draft pressures.

FAN AS ONLY SOURCE OF PRESSURE

A fan running at constant speed can produce only the combinations of pressure and quantity of flow indicated on its characteristic curve for that speed as determined directly by tests or indirectly by application of fan laws to test results. The pressure losses in the ventilating system vary with the quantity of flow, and the combinations that may exist are indicated by the characteristic curve of the system. Since the pressure generated must equal the pressure losses only one combination of pressure and quantity will satisfy both fan and system characteristics; this combination is indicated graphically by

the intersection of fan and system characteristics at a in figure 36, A, which determines the pressure as H and the quantity as Q.

FAN IN SERIES WITH NATURAL-DRAFT PRESSURE

Except in auxiliary service, metal-mine fans rarely are the only source of pressure. Components of natural-draft pressures always act either with or against fan pressures. If of small intensity they may be neglected in determining the fan duty, but in a deep mine they are an important factor in determining the quantity of flow and the duty of the fan. If all workings are on one level, or practically so, there is but one component of natural draft—that developed between the highest surface opening and the level of the workings—and this pressure acts in series with a fan handling the total flow.

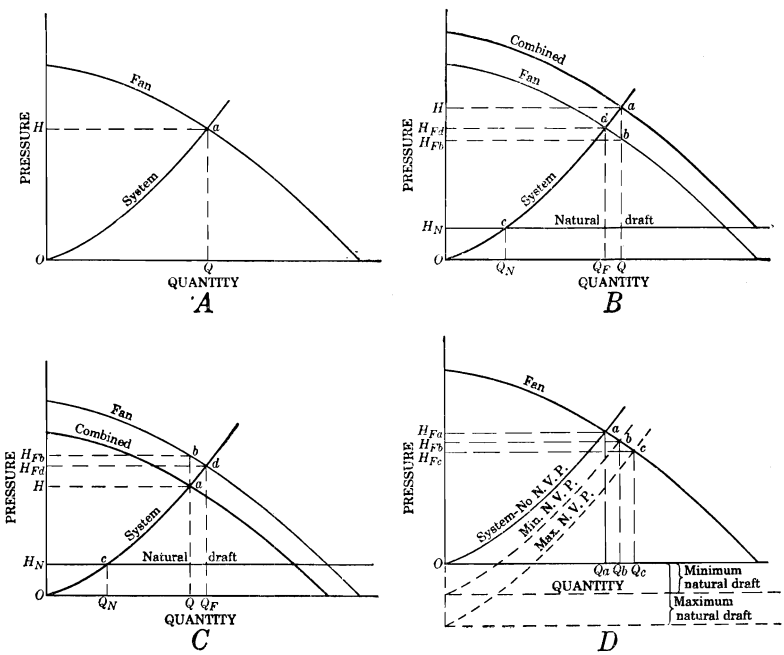


FIGURE 36.—A, Fan acting as only pressure-generating source; B, natural draft acting with fan; C, natura draft acting against fan; D, fan characteristic used directly as combined characteristic for natural draft acting with fan.

In a multilevel mine, however, all components of natural-draft pressures do not act in series with fan pressures; rather, they act as a group of constant-pressure (pressure constant while quantity changes) fans operating in parallel with each other, the resultant of the group operating in series with fans on the main flow. Although problems of distribution require a different treatment, as previously discussed in the section on natural draft, the resultant of the components acting in series with a main fan may be considered as approximately that developed between the highest surface opening and the uppermost cross-connection that carries an appreciable part of the total flow. Natural-draft pressures developed below the uppermost main cross-connection have little effect on the duty of a fan on the total-flow

part of the circuit but act mainly to change the distribution below this level from what it would be with a fan acting alone on the same system.

The conditions for natural draft acting in series with a fan are shown in figure 36, *B*. The flow is determined by the combined pressure characteristic of both fan and natural draft; this is obtained by adding the pressures for equal quantities and plotting the results against the respective quantities. The flow conditions are indicated by the intersection of the combined pressure characteristic and the system characteristic at *a*, which determines the total pressure as H and the quantity as Q. The latter determines the operating position of the fan for this condition, as at *b* on its characteristic, and the fan pressure as H_{Fb}.

With natural-draft pressure H_N acting alone on the system the intersection of the natural-draft and the system characteristics at *c* would indicate the resulting quantity of flow as Q_N. With the fan acting alone on the system the intersection of its characteristic with that of the system at *d* would indicate its operating position and determine the pressure as H_{Fd} and quantity of flow as Q_F. As respects the duty of the fan, the practical result of natural draft acting with the fan is in effect a decrease in the resistance of the system and an increase in quantity of flow over what would be produced by the fan acting alone.

The conditions for natural draft acting against the fan are shown in figure 36, *C*. The combined characteristic obtained by subtracting the natural-draft pressure from the fan pressure for the same quantities of flow crosses the system characteristic at *a* and determines the pressure and quantity of flow as H and Q, respectively. The latter determines the operating position on the fan characteristic at *b* and the fan pressure as H_{Fb}.

With natural draft H_N acting alone the characteristics cross at *c* and indicate that the quantity would be Q_N and, although not so indicated by the graph, the flow would be in the opposite direction. With the fan acting alone the characteristics cross at *d* and indicate that the flow would be Q_F and the fan pressure H_{Fd}. The duty of the fan is changed, through the action of natural draft, from *d* to *b*. As respects the duty of the fan, the practical result of natural draft acting against the fan is in effect an increase in the resistance of the system and a consequent reduction in quantity of flow.

Since the combined pressure characteristic for fan and natural draft changes with every change in natural draft, even though the fan is run at constant speed, it follows that its intersection with the constant mine characteristic will change positions and a difference in duty of the fan will result. Fan duty is therefore determined not only by the mine resistance but also by the varying intensities of natural-draft pressures. If the limits of the variations in natural-draft pressures acting in the same or opposite directions are known combined characteristics can be laid off for each condition and the range of variation in fan duty determined by the quantity intercepts on the fan characteristic, the quantities being determined by the intersections of the combined pressure characteristics with the system characteristic.

A more convenient method is to plot the same system characteristics directly on the fan-characteristic-curve sheet to separate pressure

bases determined by laying off the natural-draft pressures to the same scale. The corresponding conditions of fan performance are then determined directly by the intercepts of the separate system curves and the fan curve, since the latter, if considered with reference to the proper pressure base, is equivalent to the combined pressure curve of figure 36, *B* or *C*. The 3 points of operation of the same constant-speed fan on the same mine system, which result under 3 conditions—fan operating alone, minimum natural draft acting with the fan, and maximum natural draft acting with the fan—are indicated in figure 36, *D*, by the 3 intersections *a*, *b*, and *c* on the fan characteristic.

EQUIVALENT RESISTANCE

It is important to realize that when pressures are acting in combination on all or part of the flow in a system of airways the characteristics of a fan operating at a particular point are governed by the fan laws, but the operating point on the characteristic determined for a particular set of fan-operating conditions is determined not only by the pressure-quantity relation or resistance of the system, but also by the other pressures acting on the flow. Since the major point of interest in a mine-ventilation system is the pressure-quantity relation required at the main fan this relation often is referred to incorrectly as the mine resistance, whereas there is no common designation for this particular relation. It has been termed the "equivalent resistance" against which the fan must operate, which seems to define it well enough; this designation will therefore be used.

Since the equivalent resistance or pressure-quantity relation for a particular pressure-generating source in a mine-airway system acted upon by pressures in combination is determined partly by pressure losses which follow a definite law and partly by pressure gains which do not follow a definite law it can be determined only by direct test or by graphic methods. When the characteristics of the pressure gains and pressure losses are known graphic methods of solution may be used. When these are unknown the pressure-quantity relations at a fan run at different speeds may be obtained and plotted as the characteristic of the equivalent resistance. Then the particular point of operation of the same or a different fan at the same point for the same conditions may be obtained by using this equivalent-resistance characteristic as a system characteristic and solving graphically for the point of operation as though the fan were the only pressure source.

FANS IN SERIES

Although generation of all the mechanically applied pressure of a mine-ventilating system or a duct system at one point in the system is generally cheaper, a more efficient distribution and better operating conditions may sometimes be obtained by generating pressure at more than one point, particularly where the resistance to flow is abnormally high or certain leakage circuits are unavoidable. Some type of air pump can be purchased for any pressure-generating duty; but centrifugal fans, the common type used in mine ventilation, have a limiting tip speed beyond which they cannot be operated safely. Theoretically, the pressure and volume can be increased

indefinitely by increasing the speed, but practically the speed, as well as pressures and quantities, is limited.

Fans operating in series are those operating on the same flow circuit, each handling the total flow of the circuit and each generating part of the total pressure required to balance the pressure losses of the flow circuit. Any number of fans can be so used. Solutions for only two fans in series will be illustrated; but the same methods are, of course, applicable to any number or to combinations of fans in series with natural-draft pressures.

Since the pressure-quantity relations for fan performance are empirical graphic methods are required. As in the case of fans operating in combination with natural-draft pressures, the primary condition is that the sum of the pressures generated by the fans must equal the pressure losses for the particular quantity of flow. The flow conditions are determined by the system characteristic and the combined pressure characteristic of the fans. The latter is obtained by plotting the sum of the pressures for the same quantities. Figure 37, *A*, shows the separate and combined characteristics for two fans (*A* and *B*) acting in series on the system whose characteristic is shown. The resulting flow conditions are determined by the intersection of the system and combined fan characteristics at *c*, and the indicated system pressure and quantity of flow are H and Q, respectively. Since each fan must pass this quantity the operating position for each fan at the speed for which its characteristic has been drawn is indicated by the intersection of the vertical through this quantity line with its characteristic—at *d* for fan *A* and at *e* for fan *B*—which determines the separate fan pressures as H_A and H_B.

If the mine characteristic does not intersect the combined characteristic of the two fans then the larger-capacity fan will handle more air alone and the flow conditions require more air to pass through the smaller-capacity fan than it will handle alone when operating wide open or against no external resistance. In this case the pressure losses in the smaller fan are greater than the pressure it generates, and instead of representing a pressure gain in the circuit the fan becomes a pressure loss even though it actually continues to generate pressure. The condition has no practical importance, but experiments [31] indicate that it can be solved by roughly extending the characteristic of the smaller fan for increased quantities and negative pressures and using this extrapolated section to determine the combined pressure characteristic for quantities in excess of the capacity of the smaller fan. Practically, a change of fan speeds is indicated if both fans are to produce a useful effect, and the major point to be determined would be whether the speeds could be adjusted so that each fan would operate at close to maximum efficiency.

Unless the two fans are properly selected for the work to be done they will not operate together at maximum efficiency. Where one fan, however, is operating inefficiently because designed for a lower resistance than that encountered, whether due to changes in resistance or misapplication, a second fan may be installed to operate in series with it at any point on the total-flow part of the system so selected that both fans will operate at or near maximum efficiency.

[31] McElroy, G. E., and Richardson, A. S., Experiments on Mine-Fan Performance: Tech. Paper 447, Bureau of Mines, 1929, p. 19.

Whether or not this arrangement is advisable depends on the economic factors involved—whether the gain through increased efficiency is greater than the cost of the second installation and whether the same increase in efficiency might be better attained by replacing the single fan with another designed to operate at maximum efficiency under existing conditions.

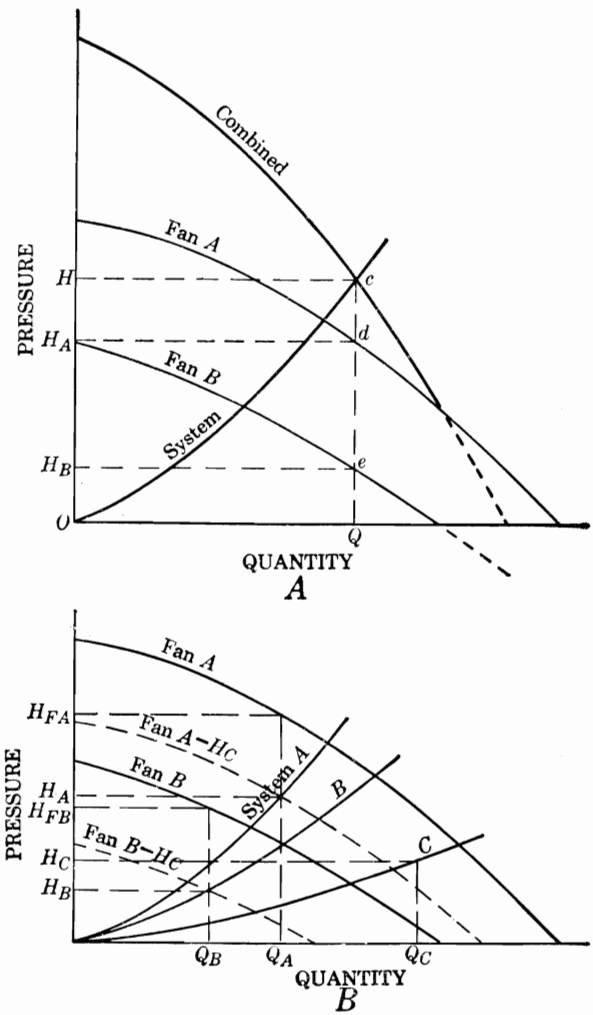


FIGURE 37.—*A*, Two fans acting in series; *B*, solution by trial and error for flow conditions for fans acting in parallel on separate airways.

FANS IN PARALLEL

To take full advantage of the lay-out of airways available for a mine-ventilating system, it is sometimes desirable to operate fans in parallel, that is, to have a number of fans together handle the total flow. Fans may also be used in parallel where space is limited, or to provide for efficient operation within certain limits of change

in resistance, or to increase the efficiency of a fan that by itself would operate against a resistance much lower than that for which it was designed.

Fans on same airway.—Fans may be placed together so that they operate on the same airway, or they may be on separate airways. In the first case they act together directly on the total flow; and a simple graphic solution, similar to that for fans in series, is available. The primary condition is that the pressure generated by each must equal the pressure loss sustained by the total quantity they produce at that pressure. The combined pressure characteristic of fans acting in parallel at the same point in the system is obtained by plotting the sum of the quantities handled at the same pressure against the pressures. The intersection of the system and combined fan characteristic determines the pressure; and, since the pressure must be the same for both fans, the pressure determines the relative quantities that each will contribute. In figure 38 the combined characteristic for the two fans acting together in parallel, each with the constant-speed characteristic shown, intersects the system characteristic at point c, which determines the pressure of both the flow and the fans as H and the total quantity of flow as Q. The constant-pressure line

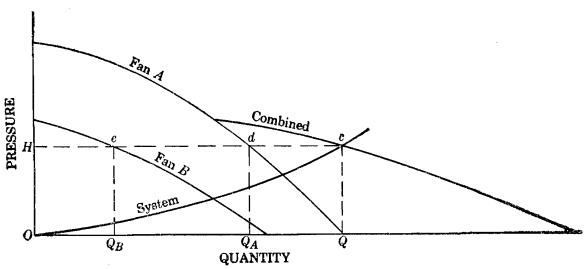


FIGURE 38.—Two fans acting together in parallel.

H intersects the individual fan characteristics at d and e and determines the separate quantities handled by each as Q_A and Q_B.

If the mine characteristic does not intersect the combined characteristic then the higher-pressure fan will handle more air alone than the two fans together, and if an attempt is made to operate them in parallel the higher-pressure fan will blow air back through the lower-pressure fan. Intersection at a point that indicates a higher pressure than the lower-pressure fan attains at any point on its characteristic indicates the same condition of reversed flow through the smaller-pressure fan. Fans that have steeply sloping pressure characteristics act together well in parallel; those that have characteristics combining a comparatively flat part and a steeply sloping part must be operated on the steeply sloping part, or performance range, at a sacrifice of efficiency to prevent sudden changes in the equivalent resistance of the system, causing the higher-pressure fan to take all of the load with the possible result of burning up a motor. With forward-curved-blade fans so installed various precautions are taken to guard against operating trouble, such as the use of fans exactly alike operated at the same speed or from a common drive shaft, arrangements for speed regulation on one or both fans, and/or an "equalizing" tube connection between the fan inlets or the fan discharges.

Fans on separate airways.—The more common installation of fans operating in parallel is that in which the fans are installed in separate airways. Where the resistances of the separate airways on which the fans are installed are comparatively large very little operating trouble is experienced, but it is difficult to determine the equivalent resistance against which fans so installed must operate for a fixed speed. The characteristics of the separate airways must be considered, as well as that of the system beyond the point of junction of the separate flows. Flow conditions and fan performances must be solved graphically by a process of trial and error.

Where fans in parallel are the only sources of pressure on the flow circuit the distribution may be indicated by fans discharging through separate branches into a common duct, as shown in figure 39, *A*, for two fans, although the actual conditions represented might be quite different. If the actual flow conditions were as sketched in figure 39, *B*, the series resistance $b+c+d$ would be equivalent to *A*, that for *e* equivalent to *B*, and that for $a+f$ equivalent to *C*. The primary condition is that the quantities of flow in the two branches must be such as to make the differences between the pressure generated by each fan and that used in flow through its branch equal to each other and equal to that required to pass the total flow through the rest of the system. Both fan characteristics and the three mine-section characteristics are plotted on the same sheet to scale, as shown in figure 37, *B*, and the flow conditions are solved graphically by successive trials. A total quantity of flow, Q_C, is assumed; the intersection of this quantity line with system characteristic *C* determines the pressure (H_C) that would be required in the total-flow or *C* section of the system. The balance of each fan pressure is available for flow through its respective branch; the intersections of curves similar to the fan characteristics, but with the fan pressures reduced by the constant amount H_C, with system characteristics *A* and *B* of the branches determine the quantities of flow through the branches as Q_A and Q_B and the pressures required as H_A and H_B, under the assumed condition of total quantity. If the combined quantities total less than the total quantity assumed a smaller total quantity would be assumed in the succeeding trial; if they add up to more than the total assumed a larger total quantity would be assumed in the next trial. When the quantities check, the intersections of the quantities through branches Q_A and Q_B with the characteristics of their respective fans indicate the equivalent-resistance conditions against which the fans must operate and the corresponding pressures as H_{FA} and H_{FB}.

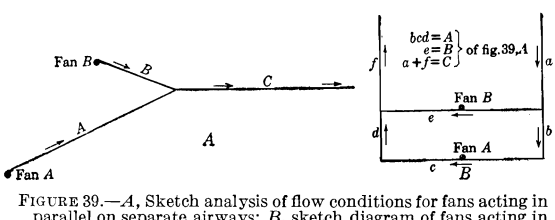


FIGURE 39.—*A*, Sketch analysis of flow conditions for fans acting in parallel on separate airways; *B*, sketch diagram of fans acting in parallel on separate airways.

COMPLEX COMBINATIONS OF PRESSURE SOURCES

In large metal mines fans may operate at various points in the system in conjunction with natural-draft pressures. In some mines the

lay-outs may be resolved into equivalent simple flow systems, but in many the various fan pressures act on but part of the flow in series, parallel, and series-parallel combinations with each other and natural drafts; many leakage circuits are involved with resultant changes in quantities of flow in addition to those caused by density changes and the addition of compressed air underground. Solutions are therefore complex and necessarily approximate. The important condition is that the pressure losses on any complete circuit, regardless of changes in quantity, are equal to the pressures generated on that circuit, whether by natural draft or by fans. Where the pressure-generating source may be considered to have a constant-pressure characteristic, as for natural draft and for certain limited ranges of operation of forward-curved-blade centrifugal fans, solutions may be aided by mathematics.

EFFICIENCY OF FANS OPERATING IN METAL MINES

If the conditions under which a fan operates agree in all respects with those for which it was designed the fan will operate at maximum efficiency; if the operating conditions do not meet these requirements the fan will operate at less than maximum efficiency. Comparison of the actual mechanical efficiency of a fan during operation with the maximum efficiency possible with that fan gives the relative efficiency of the installation.

Regardless of the degree of complexity of flow systems, most of the main fans used in western metal mines operate within 5 percent of maximum efficiency, largely owing to the fact that centrifugal fans operate against a wide range of resistance conditions with but little decrease from maximum efficiency. In many instances it is partly due to the fact that manufacturers' representatives assisted in determining the duty for the original installation and that the change in mine resistance, on account of the preponderating influence of the resistance of the main airways, has been small. Fan speeds, and even the fans themselves, are changed occasionally, but in the great majority of instances the fans have been kept at relatively high efficiency by occasional extensive changes in the airways, a method often employed when the operator's guess as to the required duty of the fan falls wide of the mark.

Secondary or "booster" installations in the same group of mines generally operate at average efficiencies 5 to 10 percent under maximum efficiencies, partly because conditions change rapidly but mainly because the operating or equivalent-resistance condition is more difficult to determine than for main-fan service. Auxiliary fan installations present the same difficulties but to a higher degree; they operate at average efficiencies ranging from 10 to 20 percent below maximum, although for this and booster service results are more important than mechanical efficiency.

SELECTION OF FANS

As shown in the preceding section, the performance of a fan in a ventilating system is determined by its characteristic (total-pressure) curve and the mine characteristic, if acting alone on the system, and by its characteristic curve and the characteristic of the equivalent

resistance, if acting in combination with other pressure sources. Characteristic curves of fans are a matter of design and are controlled by the manufacturer. The characteristic curve of a mine is a matter of the lay-out of the ventilating system and is controlled by the mine operator.

SIZE

Manufacturers do not make special fans to fit conditions; rather, they make graduated series of fans of increasing size, with all dimensions in proportion, of a limited number of standard designs or types. Special fans—those whose proportional dimensions are not the same as the standard—are occasionally made to suit speed limitations or fixed speed conditions and usually involve merely a change in the proportion of width of wheel and casing to diameter of wheel. For use in the exhaust position on mine shafts an evasé discharge section usually is attached to the standard design; some standard designs are altered slightly to include an evasé discharge, and a few types developed particularly for use in the exhaust position include it as standard design.

The relative performance conditions for different-size fans of the same design practically amount to this—any one fan will operate at maximum efficiency against only one resistance condition, but a size can be selected that will operate at or close to maximum efficiency against any particular condition of resistance. The primary condition for satisfactory and efficient operation is that the size of the fan must fit the mine or its position in the mine. The lower the resistance the larger the fan required; the higher the resistance the smaller the fan required. Resistance to flow, which practically means size of airways, and not quantity of flow determines the size of fan. Quantity of flow for an existing installation can be increased only by increasing the fan speed or reducing the resistance to flow. The fan speed is limited by mechanical stress to a safe maximum tip speed for the particular design. Since, with a fixed resistance, the power requirements of fans increase as the cube of the fan speed and quantity, the quantity usually can be increased more cheaply by reducing the resistance than by increasing the fan speed. With the same fan speed the increase in power requirements is then closely proportional to the increase in quantity, but the power may be made normal by reducing the fan speed at a sacrifice of part of the quantity increase. Notwithstanding these facts, which have been given wide publicity, the average mine operator usually attempts to increase the quantity by replacing a small fan with a larger one. With fans of the same design the resulting difference in quantity of flow depends largely on the relative tip speeds of the fan wheels; so the new quantity may be more or less than the old. The usual result is disappointing, and eventually the increase in quantity desired is obtained by using more power or by changing the airways.

PERFORMANCE GUARANTEES

Determination of the range in duties or range in resistance conditions against which the fan must work is the primary concern of the mine operator. Having determined the requirements, he submits them to one or more manufacturers who quote proposals covering a

limited number of sizes and types from which to make an economic selection on the basis of satisfactory performance over the life of the installation at a minimum total cost for installation and operation. The proposals of the manufacturers can be held to offer only fans guaranteed to operate, within the limit of the safe tip speeds, at certain efficiencies over a given range of resistance conditions without developing mechanical defects. If the actual resistance conditions differ from those specified or if the resistance conditions change after the fan is installed, the manufacturer is not responsible for the resulting lower efficiency of operation.

CHARACTERISTIC-RATIO CHARTS

From the results of constant-speed tests against variable resistances with a single size of fan the manufacturer has enough data to determine the operating characteristics of a group of similar fans for any speed. Such separate characteristics for constant speed and definite size (see fig. 34) or for the same size of fan at a number of speeds answer the common requirements of application, but for general purposes of fan selection to meet particular requirements a more general method of representing performance is required. The basic method used is that of plotting the separate characteristics as ratios and percentages rather than actual values; these may take a variety of forms. In almost any form they present a concise conception of the relative changes in performance for the particular design, but their practical use for fan selection involves considerable mathematical ability, some degree of experience in using the particular type involved, and often some basic starting data not marked on the chart but usually given as supplementary data. Some types of ratio charts are simpler and require fewer computations than others, but no really simple and complete type has been developed to the author's knowledge. What is needed is a chart from which, starting with quantity and pressure data, one could determine fan size, speed, power, and efficiency.

Figure 40 shows four types of characteristic-ratio charts, of which type *B* represents an approach to these requirements. The ratio-of-opening type (*A*), now generally discarded, is based[32] on the pressure losses at orifice plates used in tests; since these may have a wide range of variations the base is variable rather than constant as between different manufacturers using orifice plates of different types for their tests.

TABLES OF PERFORMANCE

Since the more concise method of presenting fan-performance data in the form of ratio charts is practically limited to the use of specialists and manufacturers' representatives the more bulky method of presenting these data in tables is used for catalog and general use. The tables, in general, are presented in three different forms. In one type only the rated performances—performances for combinations of pressure and quantity that permit the fan to operate at maximum efficiency—are given. A single table usually represents a complete line of sizes of the same type and design. In the second and third types a separate table shows the performance for each size over a limited

[32] McElroy, G. E., Ratio of Opening of Fan Performance in Terms of Direct Pressure-Quantity Relations: Jour. Am. Soc. Heat. and Vent. Eng., vol. 34, 1928, pp. 783–793.

range of high-efficiency operation: In one type of table the rated performances are indicated by bold-face type; in the other they must be deduced mathematically. Efficiencies are not given directly but may be calculated from the power, pressure, and quantity data and the rated performance thus obtained. Data for performances intermediate between those given may be determined approximately by interpolation and accurately by calculations based on the fan laws. A sufficiently simple type of graphic-computation chart, could one be developed, would be a great improvement over catalog tables for general use.

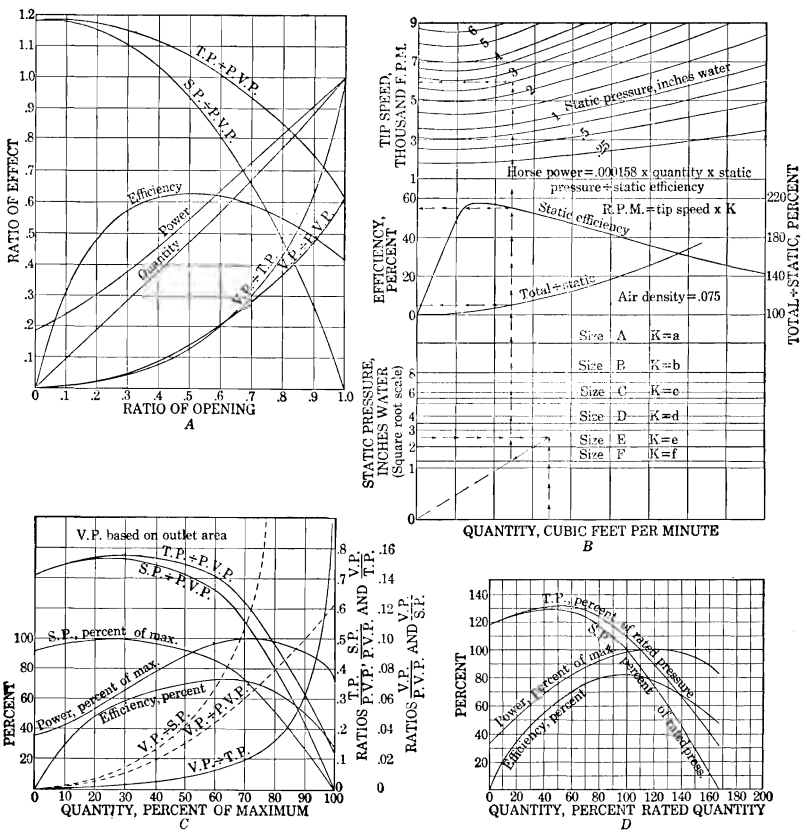


FIGURE 40.—Four types of characteristic-ratio charts of centrifugal-fan performance.

TYPES AND DESIGNATIONS

Centrifugal-fan designs differ mainly in the design of blade of the wheel or rotor, particularly its shape with respect to the direction of rotation. The shape of the blades practically determines the relative performance characteristics but is seldom mentioned in catalog data and cannot always be inferred from trade designations. Blade shapes fall into four general classes—radial, forward-curved, backward-curved, and combination. The type characteristics differ widely; those for the latter class are similar to those of the basic shape it most nearly approaches in design. Most manufacturers

offer designs embodying the three basic shapes as suitable for various types of service, but for many years the most popular type of fan for metal-mine ventilation has been the multiblade, forward-curved-blade centrifugal. Radial-blade types, often termed "paddle-wheel" or "steel-plate" (an early trade name) types, usually have a few deep blades and are now applied only to auxiliary service in American metal mines. Most forward-curved and backward-curved types have a large number of closely spaced blades of shallow depth, but a few designs have in addition a small number of deep blades.

Although the blade shape usually determines both the operating characteristics of the design and its trade designation a large number of common designations also are used to indicate variations of basic designs and methods of installation, which will be discussed later.

EFFICIENCIES

The comparative mechanical efficiency of the various types is fairly well established by consensus of opinion. Virtually all designs can be made to give equally high efficiencies if cost is disregarded, but the designs now offered to meet competitive commercial conditions have the following comparative ranges of maximum efficiency: Radial blade, 50 to 60 percent; forward-curved blade and straight-side backward-curved blade, 60 to 70 percent; and backward-curved blade with coned sides (now apparently limited as to maximum diameter by mechanical design), 70 to 80 percent. These are actual or total-pressure efficiencies. Efficencies based on static pressures average about 10 percent lower. In general, the lower efficiencies apply to smaller sizes and the higher efficiencies to larger sizes. The maximum variation for designs of the same type apparently is limited to about 5 percent. Maximum efficiencies are being gradually increased by refinements in design, and improvements are rapidly incorporated in most competitive designs.

EFFECT OF BLADE SHAPE ON PERFORMANCE CHARACTERISTICS

The general performance characteristics of centrifugal fans attributed to blade shape are illustrated by the five sets of characteristic-ratio curves of figures 41 and 42. These are for purposes of illustration only and represent types rather than particular designs. A form different from those in common use is employed to bring out more definitely the relation of performance characteristics to flow-resistance characteristics, since these are the two factors that must be coordinated in application.

POWER CHARACTERISTICS

Five sets of characteristics are shown—power, total pressure, rated static pressure, quantity, and efficiency. The dominating characteristic for mine application is that of power. Since the power requirements of the backward-curved type are practically constant at or lower than the rated power it is said to have a "nonoverloading" power characteristic and is therefore suitable where the resistance conditions fluctuate over a large range. Both radial-blade and forward-curved-blade types have rising power characteristics; they are therefore subject to large overloads if the resistance is greatly reduced, as by short circuits due to doors left open or to tubing disconnected

from an auxiliary fan. These types require motors of excess capacity, up to 50 percent, to guard against possible overloads from a fluctuating resistance.

PRESSURE CHARACTERISTICS

Pressure characteristics of fans probably have received the most attention, as they are related more directly to blade shape. They determine the suitability of the type for operation in parallel and for special application where constant pressure or constant quantity rather than constant efficiency is required. The backward-curved-blade

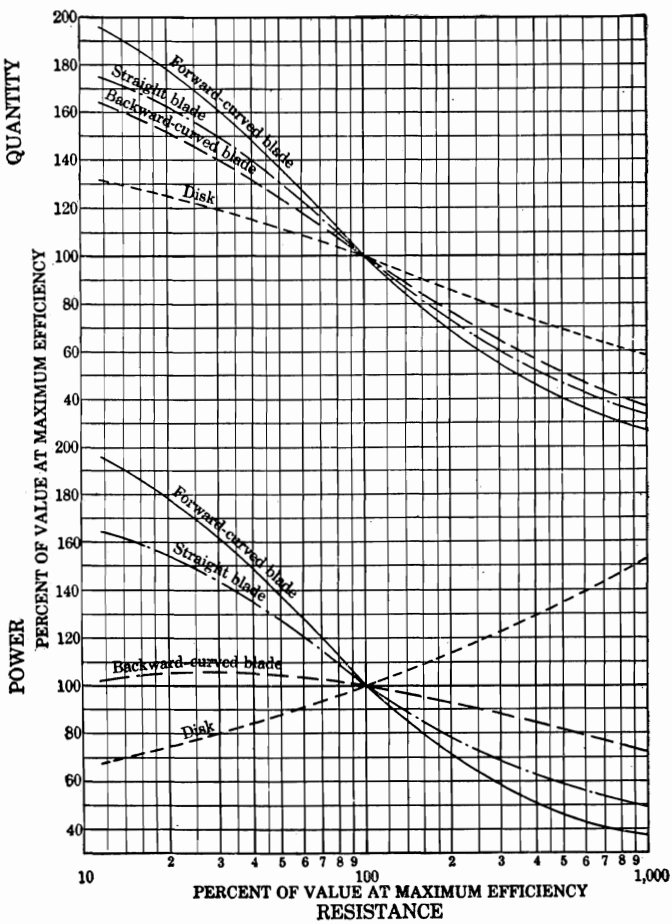


FIGURE 41.—Comparative quantity and power characteristics of mine fans.

type has a very steep total-pressure characteristic and is therefore suitable for operation in parallel at maximum efficiency. The forward-curved-blade type has a very flat total-pressure characteristic near maximum efficiency and requires a large sacrifice in efficiency for operation of the sloping part of the characteristic, as required for successful operation in parallel. The radial-blade type has a gently sloping characteristic and requires but a small sacrifice in efficiency for parallel operation. Pressure characteristics may be largely modified

by the design of the housing, as instanced by the design of certain forward-curved-blade types developed for auxiliary service which have a steeply sloping pressure characteristic similar to that indicated here for the backward-curved-blade type. Forward-curved-blade types are good for constant-pressure requirements and backward-curved-blade types for approximately constant-quantity requirements.

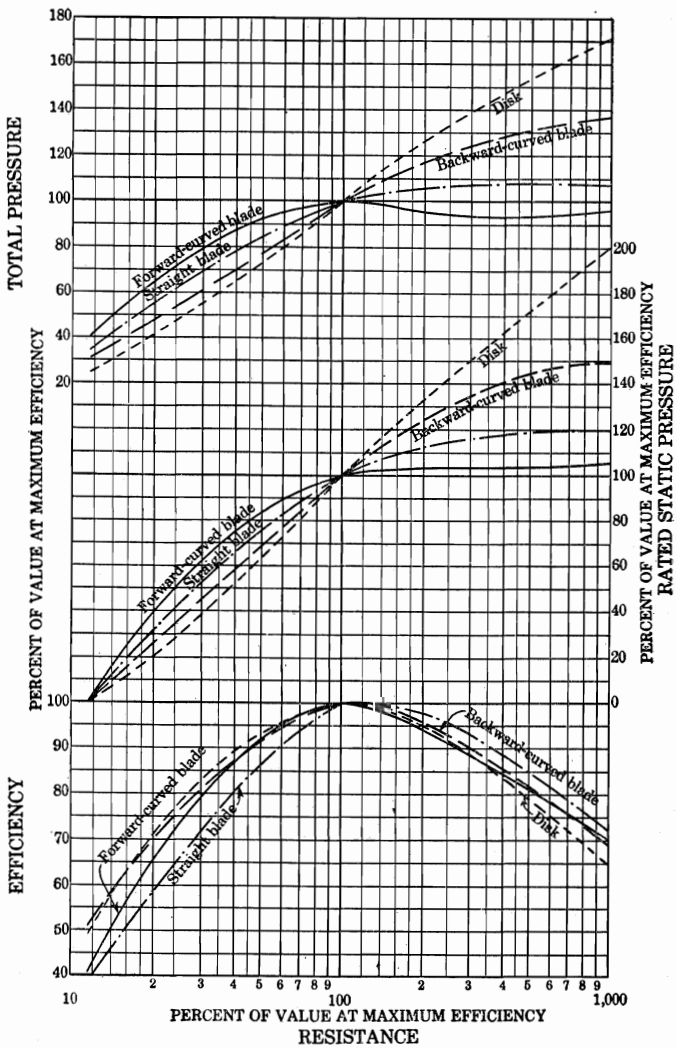


FIGURE 42.—Comparative pressure and efficiency characteristics of mine fans.

The question, so frequently discussed in coal-mining periodicals, of inferring changes in mine resistance produced by changes in airway conditions from observed or recorded fan pressures is easily solved by inspection of the pressure characteristics. The "water gage" for the fan in the blower position follows the rated static-pressure characteristic, and increases or decreases in water gage generally indicate

similar changes in resistance without necessarily indicating the degree of change. The water gage of a fan in the exhaust position, the more common position of mine fans, follows the total-pressure characteristic; the change in pressure generally indicates a similar change in mine-airway and resistance conditions, except for the forward-curved-blade type; the characteristic for this is so flat over the usual operating range that no inferences as to airway changes can be deduced from pressure changes. The large majority of coal- and metal-mine fans are forward-curved-blade types operating in the exhaust position on the flat part of the pressure characteristic where the water-gage records serve no useful purpose other than to gage the constancy of the fan speed. The most useful single record of fan performance is that of quantity of flow; unfortunately, it is more difficult and costly to obtain than a pressure record.

EFFICIENCY CHARACTERISTICS

Economy of operation is not only a question of maximum efficiency but also of range of resistance conditions that can be accommodated at high efficiency. The flatter the efficiency characteristic in the vicinity of the maximum the more suitable the type for operation against changing resistance. The efficiency characteristics shown indicate that, in general, the backward-curved-blade and radial-blade types have a slight advantage over the forward-curved-blade type.

OTHER OPERATING CHARACTERISTICS

The characteristic charts shown do not reveal the speed and other characteristics directly related to mechanical performance. For the same resistance and flow conditions the speed of rotation of the backward-curved-blade type is almost double that of the forward-curved-blade type, with the speed of the radial-blade type intermediate. The speed of the ordinary disk fan, seldom used for metal-mine ventilation, is about twice that required of the backward-curved-blade centrifugal. High speed is a definite advantage in direct-connected motor-driven units, since high-speed motors cost much less than low-speed motors.

Quietness of operation, although an important factor in some commercial applications, ordinarily is relatively unimportant in mine ventilation. Although the conversion of velocity pressure to static pressure in the housing is mainly responsible for noise, speed of operation and details of design of both fan and connecting ducts also contribute. In general, the forward-curved-blade types are the noisiest and the backward-curved-blade types the quietest.

The uniformity of the intensity of the noise indicates the constancy of performance. Cases of unstable performance, such as would require multiple characteristics rather than single characteristics to indicate the performance, are very rare and, to the author's knowledge, confined to particular installations of the forward-curved-blade types only, probably because of the high degree of velocity-pressure conversion in the housing of this type. The inference is that such rare cases of unstable performance are caused by unstable turbulence effects at the inlet due to "wild" turbulence or excessive shock pressure losses in the inlet passages.

VARIATIONS IN DESIGN TO ACCOMMODATE CONDITIONS OF INSTALLATION

Basic designs usually are offered in a variety of forms differing in number of inlets, width, degree of completeness of housing supplied by manufacturer, direction of rotation, and direction of discharge.

The basic design of standard single-width fan may have single or double inlets; that is, the air can enter from one or both sides. A double-width fan always has 2 inlets and is virtually 2 single-width fans joined with the rotors on a common shaft. Double-width fans are popular as main fans for mines, particularly for underground installations, because both fan and total installation costs generally are lower for large-capacity units than single-width installations. The latter generally give lower total costs for small capacities and for sidehill surface locations. The double-width fan handles just about twice as much air at the same pressure as the standard single-width fan of the same design, and in the absence of separate characteristics those for the single-width fan may be used with the quantity doubled. Fans of special width occasionally are used for special conditions of operation not conveniently handled by standard single- and double-width designs.

The standard fan calls for a complete metal housing and is then termed "full-housed." The operator constructs the foundations according to plans supplied with the fan; since a building to house the drive and ducts to connect the fan with the mine airways must also be erected he sometimes prefers to build part of the fan housing of the same or more permanent materials. The fan manufacturer will supply fans and necessary plans for completing the housing accordingly; the fan is "half-housed" if the operator constructs the lower half or "three-quarter" housed if the operator constructs the lower quarter. Various advantages, including reduction of leakage at fan connections, thus accrue to the operator. Where the casing extends below the ground level it is particularly important that means for draining the pit thus formed be provided.

REVERSING ARRANGEMENTS

Centrifugal fans cannot be reversed by reversing the direction of rotation; special provision must be made in the fan housing and the connecting ducts to reverse the direction of flow through the ventilating system while the direction of flow through the fan remains constant. The necessary ducts must be provided and equipped with suitable doors, if the ducts are large, or dampers, if tubing is used, so that in one case the mine air is directed to the inlet of the fan which discharges to the atmosphere, whereas in the other the atmosphere is connected to the inlet and the discharge to the mine. The general arrangement for tubing and dampers is shown in figure 43. A common design for a single-inlet fan is shown in figure 44, *A*, and for a double-inlet fan in figure 44, *B*. Reversing in these instances alters the direction of discharge (consequently, the fan characteristic) and the resistance to flow through the ducts that connect the mine with the fan. Designs are therefore selected for the better-performance conditions for both fan and duct connections and designated as "primarily blowing" or "primarily exhausting" reversible. In general, provision of the reversible feature usually involves larger leakages, higher resistances, and higher costs than for nonreversible

installations. The extent of these differences can, of course, be largely controlled by proper design. The cost of the reversing feature for the average main-fan surface installation is usually 10 to 20 percent of the total cost of the installation, but when the reversing feature is included in an underground installation the total cost may easily be double that of a nonreversible lay-out, for which reason the reversing feature has been applied to but one underground fan in the West, to the author's knowledge—the main fan of the United Verde mine in Arizona. The Central Eureka mine of the Mother Lode in

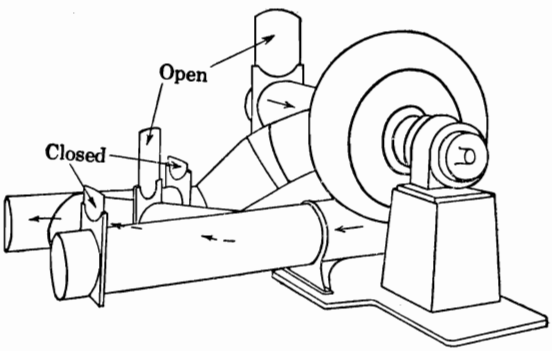


FIGURE 43.—Four-gate arrangement for reversing air flow at auxiliary fan-tubing installation.

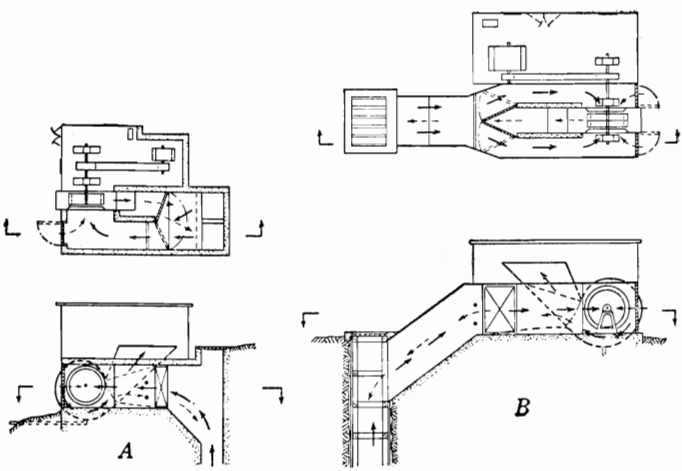


FIGURE 44.—*A*, Arrangements for reversing air flow at single-inlet, primarily exhausting fan installed on air shaft; *B*, arrangements for reversing air flow at double-inlet, primarily exhausting fan installed on air shaft.

California, however, accomplishes the same result with an installation of two separate fans, operated one at a time and automatically interconnected to reverse the air currents by remote control.

DIRECTION OF ROTATION AND DISCHARGE

In fitting the fan to conditions at the point of installation it is usually desirable to fix the position and direction of discharge with respect to the airways or duct connections. This merely involves the

rotation of the casing about the fan shaft and does not affect the performance of the fan but only the details of supporting the wheel and the housing and applying the drive to the shaft. The direction of discharge is designated by the relation of the center line of the discharge to the fan shaft when the fan is viewed from the drive side which, in the case of a single-inlet fan, is always taken to be the side opposite the inlet, regardless of the actual position of the drive. If in this position the normal direction of rotation of the wheel is clockwise the discharge is clockwise and if counterclockwise the discharge is counterclockwise. Clockwise is right-hand and counterclockwise left-hand rotation, but these designations are not used in this connection. If the line of discharge is horizontal and above the shaft the discharge is top horizontal and if below the shaft, bottom horizontal. If the line of discharge is vertical it is either upblast or downblast according to whether above or below the shaft, respectively. If the discharge is in any other direction it is angular discharge and specified in degrees as top or bottom angular, up or down discharge, as the case may be.

DRIVES

A drive mechanism of almost any type may be used to drive the fan shaft. In the smaller sizes the shaft bearings usually are supported on the housing, but in the larger sizes, particularly those used as main fans, the shaft and wheel are supported on pedestal bearings independent of the housing. When foundations settle, as they often do around mine workings, the wheel and casing are thrown out of alinement and the performance is seriously affected.

Constant-speed alternating-current motors generally are used for driving mine fans either through direct or belt connection. With belt connection the speed can be changed by changing the pulleys. Variable-speed motors are advantageous for certain installations but are seldom used on account of the higher cost. Direct-connected units are used almost exclusively for auxiliary service and occasionally for secondary or booster service. The speeds thus obtainable are limited to the few that are available as determined by the phase and cycles of the electric current. Three-phase, 60-cycle current predominates in western metal mines, but 3-phase, 25-cycle current is used in a few mines. On account of the saving in space of installation short-center V-belt drives are gaining favor over the common belt drive in almost universal use, particularly as the latter often gives considerable operating trouble in underground service. Squirrel-cage induction motors are the common type in use, although synchronous motors are sometimes used on large units for power-factor correction and other types of motors for multiple-speed control.

For auxiliary service fans of several types are available for direct drive by compressed air through a turbine attached to the fan wheel, and small compressed-air motors are also obtainable. These drives are less efficient than motor drives but are useful on failure of power supply, as at the time of a mine fire, and for handling temporary conditions at a distance from the power supply, since compressed air is generally available throughout a metal mine.

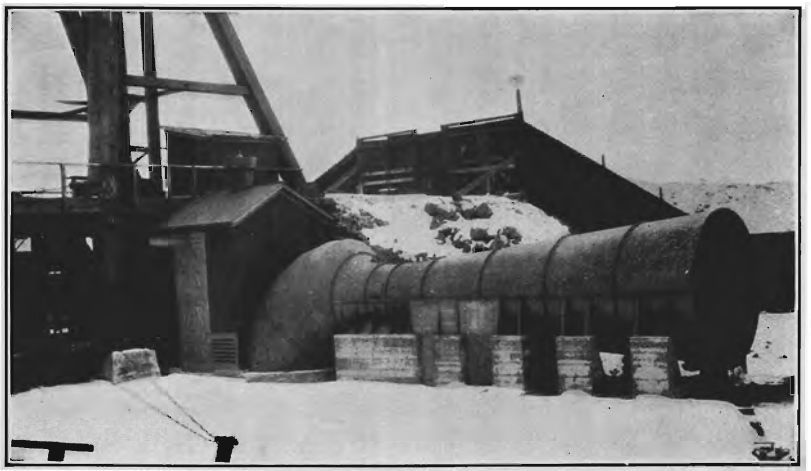
ECONOMIC SELECTION OF MINE FANS

The selection of a fan for maximum efficiency in the use of power is primarily a matter of fan size—selection of a size that most closely fits the resistance encountered. The primary object of a fan installation, however, is to obtain the required flow at a minimum total cost per year of operation, and total cost is not only a question of power but also of capital charges based on a fair return on, and replacement of, the money invested. The total cost of a simple main-fan installation usually is at least 3 and may be 10 or more times that of the fan alone. Although the cost of the fan increases with size—at the approximate rate of $500 per foot increase in diameter of wheel for single widths and $1,000 per foot increase for double widths for main-fan sizes—the increase in total costs and capital charges is not in proportion to the increase in size. To make an economical selection from the fans offered for a particular duty, with different possible arrangements and necessary connecting ducts and foundations, total costs comprising both power and capital charges must be computed. Details of differences in mechanical construction may indicate probable differences in maintenance charges, which must also be considered. Generally, a fan for large capacity must be selected to fit the resistance conditions as closely as the sizes available permit or may even demand a special width at a slightly higher first cost. But where the capacity required is moderate to small a size smaller than that demanded for maximum efficiency of use of power often gives a smaller over-all cost even though it operates at less than maximum efficiency on the resistance conditions existing. Proper provision for the future operation of a fan on an increasing or decreasing resistance also demands consideration where the conditions can be forecast or roughly estimated.

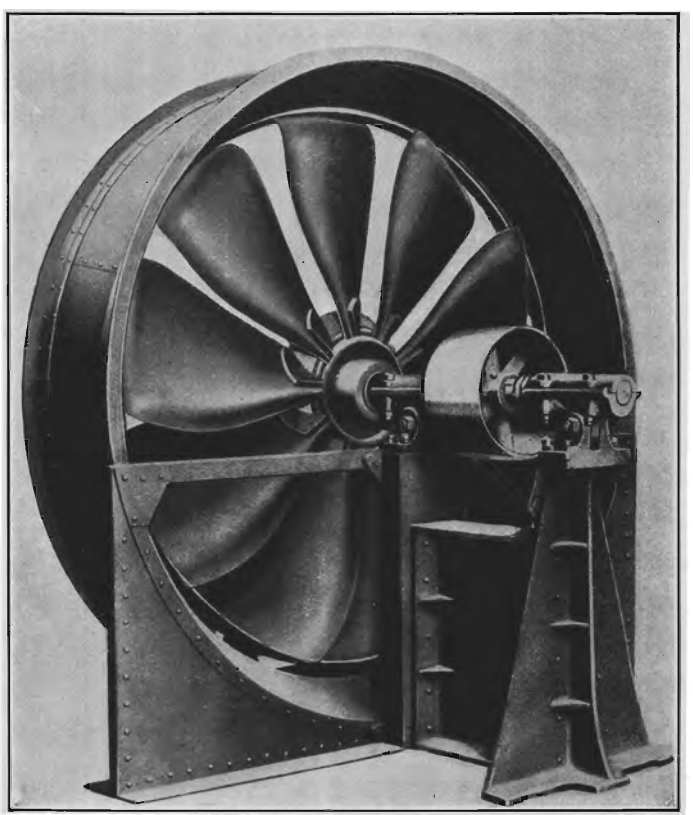
AIR-SCREW FANS

Air-screw fans, that is, disk and propeller fans, have found limited use in mine ventilation. Many of the ordinary disk type with short cylindrical casing are used in booster service in coal mines, but very few are found in metal mines. A special type of single-stage propeller fan, the Coppus, which includes a propeller, guide vanes to take the "spin" out of the air, and an evasé discharge section, has found large use in secondary and auxiliary service in the metal mines of this country; both single-propeller and multiple-propeller fans now under development may be expected to become strong competitors of the centrifugal fan for all mine service. One recent installation, which is also the heaviest-duty fan [33] in American metal-mine service, is shown in figure 45, *A*. It is a single-propeller design of a new nonuniform-pitch (Schmidt) type, which, like the Coppus, has vanes to take the spin out of the air and a good evasé for velocity-pressure conversion. This fan is rated at 300,000 c. f. m. at 9 inches static pressure for an air density of 0.057 pound per cubic foot (equivalent to almost 12 inches at standard density of 0.075), is built to withstand pressures up to 40 inches, and is reported to have a maximum mechanical efficiency of 83.5 percent.

[33] Richardson, A. S., Propeller-Type Mine Fan at the Moose Shaft, Butte, Mont.: Trans. Am. Inst. Min. and Met. Eng., February 1932, vol. 102, pp. 86–96.



A



B

FIGURE 45.—*A*, Propeller fan installed as main fan on the Moose shaft of the Anaconda Copper Mining Co., Butte, Mont.; *B*, standard-type disk fan.

The ordinary-type disk fan (fig. 45, *B*), as made for mine service, has about the same range of maximum efficiencies as the radial-blade centrifugal (50 to 60 percent). Air-screw fans obey the same laws of performance as centrifugals and are selected in the same way from characteristic curves or performance tables. The characteristic curves of the standard disk types, as shown in figures 41 and 42, differ mainly from centrifugals in that the power decreases as the resistance decreases and quantity of flow increases; that is, disk fans require the maximum power when delivering no air and the minimum when delivering the most air. The reverse is true for centrifugals. They are otherwise high-speed fans with steeply sloping pressure characteristics and adapted to parallel operation. Pressures developed at available speeds are low and do not exceed 1 to 2 inches of water, depending on the design. Special disk and propeller types, however, by converting velocity pressure at discharge into static pressure, can work at high efficiency against the high pressures required in metal-mine service and have a very flat power and efficiency characteristic.

The experiments of F. A. Steart [34] in Australia have shown that multistage air screws, in this case multiple, 2-blade, airplane propellers mounted on a common shaft, not only can develop high pressures at

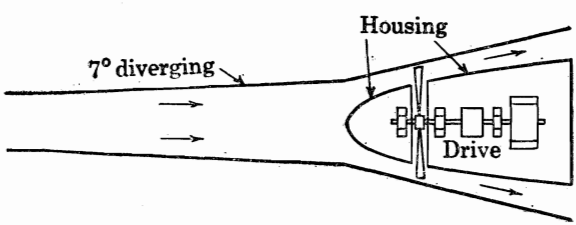


FIGURE 46.—Suggested form of installation for propeller fan.

efficiencies comparable to centrifugal efficiencies but may be adjusted as to number and pitch of blades to maintain maximum efficiency against changing resistance conditions, a very valuable characteristic. Several large-capacity installations of this type have been made in England and South Africa; development should be rapid on account of simplicity, ease of reversal, and ease of adjustment. Adjustment possibly may be facilitated by applying the control mechanism developed for an experimental type of airplane motor, in which, according to news accounts, the pitch of the blades is adjusted from the cockpit.

An air screw is so much more efficient near the tips of the blades in moving air that a partial vacuum tends to develop back of the blades near the center and induces reversed flow, which radically lowers the efficiency. In most designs of disk and propeller fans this reversed flow is greatly reduced by blanking off the central part of the wheel, although this apparently is not required with multiple-propeller types. As a result of their studies of air flow in wind tunnels and of propeller design Caldwell and Fales [35] propose the design sketched in figure 46 for arranging a single propeller and housing to

[34] Steart, F. A., The Application of Air Screws to Mine Ventilation: Trans. Inst. Min. Eng. (London), vol. 68, 1924–25, pp. 310–321.

[35] Caldwell, F. W., and Fales, E. N., High-Efficiency Air Flow: Jour. Am. Soc. Heat. and Vent. Eng., May 1920, p. 412.

obtain high pressures at high efficiency. Steart's original experiments with 2-blade propellers showed average maximum efficiencies of 60 to 65 percent; with more blades and better housing design these figures possibly have been, or at least will be, surpassed.

On account of their relative cheapness disk types of air-screw fans have been so misapplied to mine ventilation that the air-screw type, in general, is more or less in disrepute for this service. However, where they can be applied the installation is so much cheaper and simpler than for a centrifugal that efficiency can be sacrificed to some extent if necessary.

HIGH-PRESSURE MACHINES

High pressures are only required in mine ventilation for auxiliary units ventilating dead-end openings through long lines of tubing. The general tendency at metal mines is to use low-pressure fans installed in series rather than a single high-pressure machine for high-resistance conditions. Thus no specialized units are required and

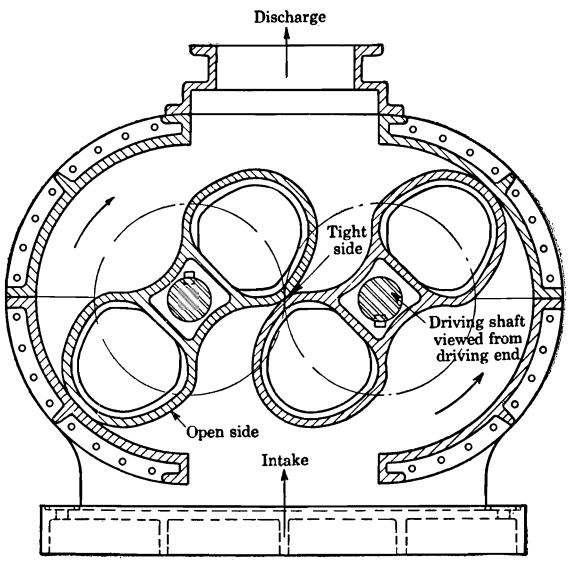


FIGURE 47.—Section of rotary positive-pressure blower (P. H. & E. M. Roots Co.).

both pressure differences and leakage conditions are comparable with those experienced with shorter lines and single-fan installations.

High-pressure machines capable of producing pressure differences measured in pounds per square inch (1 pound per square inch=27.7 inches water) rather than inches of water are especially applicable to the ventilation of long tunnels through small-diameter pipe during the driving period. Positive-pressure blowers have generally been used for this service, for which centrifugal compressors are also applicable. Positive-pressure blowers, represented by the Roots blower (fig. 47) are rotating displacement compressors which trap air between rotating impellers and force it through the system regardless of the pressure required, except for a small amount that leaks back from discharge to intake. The amount of leakage, termed "slip", depends only on the pressure and impeller clearances. Cen-

trifugal compressors act similarly to centrifugal fans and have similar characteristics. Both types of machines have a range of efficiencies comparable to centrifugal fans. The compressors require additional ducts for reversing the flow, whereas the flow through the blowers can be reversed merely by reversing the direction of rotation of the impellers provided the necessary clearances have been incorporated in the design.

The pressure changes are so large with high-pressure machines that the flow relations cannot be considered on a constant-density, constant-quantity basis, as has been done throughout this bulletin for low-pressure mine flow. Different methods of computing pressure drops and machine performance are required that take into account the changes in density and quantity that accompany pressure change. Although the same general method of determining performance by correlating system characteristic to machine characteristic is applicable, both characteristics vary with the position of the machine in the flow system and a more involved procedure is therefore required to determine performance under specific conditions. This procedure has been ably stated by Professor Weeks,[36] and since it has little application to metal-mine ventilation and might introduce an element of confusion were it set out here no discussion of the theory and practice of high-pressure ventilation will be presented.

AIR INJECTORS

Air jets, injectors, ejectors, and similar devices for producing air flow have found limited application in metal-mine ventilation. They are easily installed and cheaply constructed but waste so much power that they are used only in temporary or emergency auxiliary service. They depend on the conversion of the velocity pressure of a high-velocity jet, of what actually amounts to a separate flow system, into useful pressure to produce flow in the system where desired. In mine work the motivating system is usually a high-pressure, low-quantity system—compressed air discharging through a small-diameter nozzle—but it may be a low-pressure, large-quantity system similar in general aspects to the "Saccardo" system of tunnel ventilation discussed later. Air jets and injectors may take many forms—from nozzles designed to screw onto a compressed-air hose to well-designed venturi types, such as that shown in figure 48. The latter are used to a large extent in German and South African mines for auxiliary ventilation and have been used in a few of the metal mines of this country since their introduction at the United Verde mine several years ago. Simpler types, such as a jet discharging on the axis of a uniform-diameter pipe or on the axis of single or multiple cones for air-motion effects at dead-end faces, have also been used to a limited extent in American metal mines.

The over-all efficiency of pressure generation is the ratio of the useful work done on the main air-flow system to that required in the activating flow system. As Weeks [37] shows, the efficiency with which the energy of the jet is utilized in common designs is very low, and the over-all efficiency of both flow systems is still lower. Using compressed air in

[36] Weeks, W. S., Theory and Practice in the Ventilation of Long Tunnels: Eng. and Min. Jour., Nov. 17, 1928, pp. 779–781.
[37] Weeks, W. S., Ventilation of Mines: New York, 1926, p. 141.

the "Modder" design, after which most home-made types are patterned, Weeks obtained a maximum jet efficiency of 7.3 percent and an over-all efficiency, based on an average figure of 45 percent for efficiency of compressed-air production, of 3.3 percent. Higher efficiencies have been reported for special designs, but the corresponding maximums probably are not over 15 and 7 percent, respectively. The latter figure should be compared with the average of about 60 for centrifugal fans in determining the status of injectors. However, they are useful emergency and temporary installations and will operate under conditions that would ruin a fan and motor in short order.

Performances for injector designs may be approximated only roughly by theory [38] and must therefore be actually determined by tests against a range of resistance conditions, as in the case of fans. Similar characteristic curves of performance may be used in solving problems

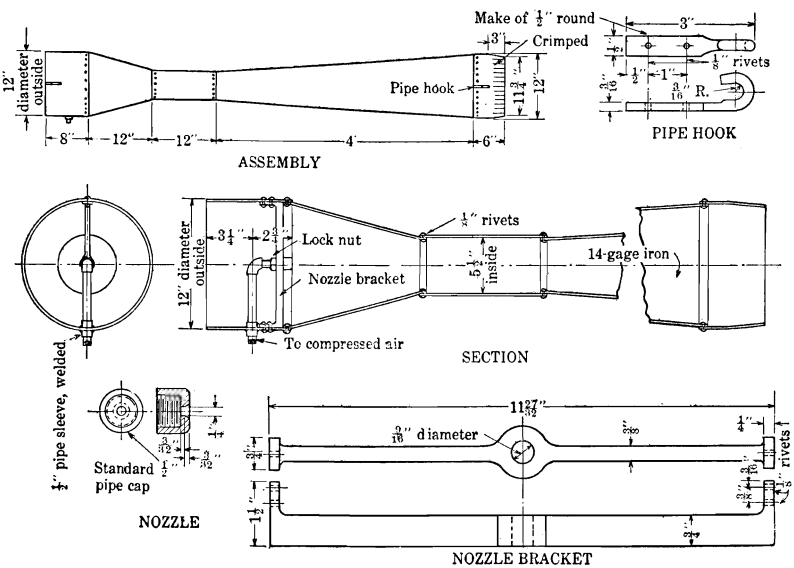


FIGURE 48.—Venturi type of compressed-air injector used for auxiliary ventilation in United Verde mine, Jerome, Ariz.

of application. Curves are required for each orifice size and orifice pressure with compressed-air jets and correspond roughly to the separate fan characteristics required for separate speeds and sizes but to the author's knowledge without the benefit of a set of laws similar to the "fan laws" to facilitate the determination of a multitude of characteristics from one set of tests or the use of characteristic-ratio curves.

Saccardo system.—Although the Saccardo system of injector ventilation was developed for ventilating tunnels and has been used exclusively for this purpose, it has a possible use under special conditions in metal-mine ventilation, that is, where operating advantages, such as those resulting from elimination of an air lock on a busy haulageway or shaft, would offset the relative inefficiency of power application. In this system air flow from a fan is led through ducts to a small

[38] Colliery Engineering (London), Ejector Ventilation: Vol. 3, November 1926, pp. 474–475.

opening or nozzle, where it discharges at high velocity into the main airway system in the direction of flow. A large part of the velocity pressure at the discharge of the nozzle is converted to static pressure in the airway system. The total pressure usefully applied to the system is the difference between the velocity pressure at the nozzle discharge and the pressure loss due to shock loss attending the change of area as between the nozzle discharge area and the area of the airway into which it discharges. The performance [39] varies with the ratio of nozzle area to airway area, direction of discharge with reference to the axis of the airway, and design of nozzle; like fans, each design will operate at maximum efficiency against but one resistance condition and at lower efficiencies against all others. The maximum possible efficiency in the use of jet energy, with abrupt expansion over almost half of the perimeter, can hardly exceed about 50 percent, which means a probable maximum over-all efficiency of the system of about 30 percent; lower values may be expected in actual practice, since the duct pressure losses from the fan to the nozzle are not included in this figure. The pressure losses in the ducts carrying the air to the nozzle increase at the same rate as the velocity pressure at the nozzle and limit the application of the system to relatively low-resistance ventilating systems.

PRESSURES GENERATED BY MOVING OBJECTS

Still objects in airways obstruct the flow and cause pressure losses, but when they are moving in an airway they not only act as obstructions to flow but also generate pressure in the direction of their movement. When they are moving in the same direction but at a slower rate than the average velocity of flow, the pressure losses exceed the pressures generated, and the net results are pressure losses similar to those experienced with fans which, under the action of other pressure sources, are forced to pass more air than they normally handle against no resistance. When objects, such as cars, skips, or cages, move through an airway at a faster rate than the average velocity of flow but in the same direction, the pressures generated exceed the pressure losses, and the net pressures generated act with or without other pressures to create air flow. When the movement is opposed to the direction of flow, the pressure generated is opposed to that causing the flow and is then equivalent to a pressure loss, as when natural-draft pressure is opposed to fan pressure. Theoretical calculations of pressures generated by moving objects are of doubtful and incomplete application if based on relative velocities and empirical data resulting from pressure-loss determinations for still objects as obstructions in an airway. It seems probable that characteristic curves of the net pressures generated by moving objects could be devloped from actual tests in which all factors were taken into account but to the author's knowledge no authentic data are available.

Large-volume circulations often are reversed temporarily by rapid movement of skips and cages in shafts and trains of cars on horizontal airways, where the moving object almost fills the airway or where the pressure inducing the circulation is small, as in natural-draft ventilation. In an open-timbered shaft the movement of skips or cages

[39] Bulletin of the International Railway Congress, The Ventilation of Tunnels on Saccardo's System: Eng. ed., vol. 8, April 1899, pp. 477–495.

operating in balance affects the air flow only slightly, since opposing pressures are generated and there is always plenty of clearance for the air flow in adjoining compartments. Where the air flow is in separately lined compartments with small clearance the piston effect is large, although usually limited considerably by openings between compartments at frequent intervals. As a rule, these are provided to facilitate repair of the shaft, but they also limit the piston effect of cage and skip travel and thus reduce interference with normal distribution.

DISTRIBUTION OF AIR CURRENTS

Natural distribution of air currents in mine openings is practically always inefficient and unsatisfactory; although circulations on the main openings may be good very little air is circulated through the working places where it is most needed. Some places receive more air than they need and others none at all. For effective and efficient distribution both direction and quantity of flow must be controlled.

General directions of flow are determined by the points in the system of connected openings at which the pressures are generated and the general quantities of flow by the intensity of the pressures generated. Both direction and amount, however, can be controlled not only by regulating the position and intensity of the pressures generated but also by artificially regulating or modifying the natural resistances to flow of the mine openings or, if necessary, by providing artificial airways. Resistance to flow on an airway may be increased by tight stoppings so as to stop the flow entirely, or it may be increased by loose stoppings to restrict the flow to leakage, or it may be increased a definite amount by regulators to provide only the desired amount of flow rather than the larger amount that would occur without regulation. Resistances to flow may be decreased by all the various methods of changing the particular physical conditions of the airway, or an equivalent result may be obtained by local generation of pressure on an airway circuit, as by a secondary or booster fan. Where there are no separate intake and return airways, as in dead-end development openings, a single airway may be divided by a wall or brattice to make two separate airways, or an artificial airway may be constructed within the original airway, as in auxiliary ventilation by fan-tubing units, or at the junction of airways to permit air currents to pass each other without mingling.

Where the natural or artificial features of the distribution system are not tight, leakage not only increases the power required to produce the desired circulation but increases the hazards at the time of fire through lack of control. If the leakage in the circuit does not go direct from the system to the atmosphere but is recirculated underground the fire hazards and likewise the waste of power are greatly increased.

POSITION OF MAIN FAN

Directions of flow, which largely determine the air conditions in operating zones, conditions of leakage and recirculation, and control of distribution both normally and in the event of underground fire are primarily determined by the position of the main fan or fans. The demands of service and safety sometimes coincide but usually are antagonistic either in whole or in part. With 2 surface openings

available, 1 in active use for operating purposes and the other an inactive air shaft, primary consideration of the ventilation problem involves 3 decisions—whether the general direction should be fixed or subject to reversal in emergency, what should be the normal direction with respect to the operating opening, and whether the fan should be on the surface or underground.

Actual conditions control the final decisions on these controversial subjects, but in most instances a reasonable regard for the safety of life and property requires a reversible fan, primarily exhausting, installed on the surface at the nonoperating opening. This makes the operating opening, which the men use in going to and from their working places and to which they naturally go at the time of an emergency, an intake opening and one most likely to provide a safe exit as well as a safe base for attacking the cause of the emergency.

USE OF REVERSING ARRANGEMENTS

The reversing arrangement should be installed regardless of whether the necessity for it can be foreseen or not. It is primarily an insurance against unforseen happenings and as such has proved its worth in a number of instances. Lack of reversing arrangements, on the other hand, has been held responsible by competent observers for much of the loss of life and property in certain of our greatest mine-fire disasters. Whether or not to reverse a fan during a mine fire is usually a serious decision—one to be made only by the highest officials immediately available after due consideration of all factors involved; the reversing arrangements should therefore be locked unless reliable attendants are employed. Time is an important element, and since time is required to ascertain the necessity of reversal and to carry out the operation it is important not only to insure smooth operation by occasional trials of the mechanism but also to plan other methods of control to insure safe exit of the men and to prevent the fire from gaining headway rapidly, such as installing fire doors and poss bly slowing down or stopping the fan or fans.

DIRECTION OF FLOW

The important consideration in determining direction of flow is that the usual means of exit should be on intake air; this usually means that the operating opening should be an intake. Other means of obtaining the same result are, of course, just as effective, such as making the operating opening a fresh-air outlet, or upcast in the case of a shaft, or putting it under pressure with intake air. In the average naturally ventilated mine the directions of flow cannot be controlled easily; in many in which the general directions are constant the operating shaft normally becomes an upcast, although this tendency is not as marked now as when steam pumps were commonly used underground. Where possible the flow should be reversed to make the operating opening an intake.

In cold climates one of the chief objections to the use of operating openings as intakes is the formation of ice in the opening and the freezing of water lines. Most of these difficulties can be largely overcome at moderate expense which, even in addition to the discomfort produced under certain operating positions, does not begin to balance

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the sacrifice in general safety conditions that would otherwise result. Because of the effect of rapidly changing temperature and moisture conditions on timber and rock, intake openings are naturally subject to higher maintenance charges near the entrance than the returns with their uniform air conditions, but return air currents not only may be highly corrosive to equipment but also make for wet, muddy, hot, foggy, and smoky combinations of air conditions, which do not promote efficient operation in a zone of great activity.

Where large quantities of dust are raised on or adjacent to the opening that should for safety be used as an intake, airway provision should be made, of course, to prevent the dust from entering the intake currents; such special conditions make every mine a separate problem for which only general results can be indicated with any degree of exactitude. In a large mine many conflicting features are involved, and the system should be planned to take care of as many of them as possible in the order of their importance, first, as respects safety and, second, with regard to service.

MAIN FANS ON SURFACE

In general, main fans should be installed on the surface where they are subject to maximum control. A fan underground may be put out of commission entirely during a fire; about the only control assured is that of cutting off the power. Some operating conditions, however, may make it desirable to install the main fan underground for normal operation, in which case a surface fan may be installed as a stand-by in case of a fire underground or separate power sources may be connected to the fan and remote control provided. In some lay-outs an underground installation may actually be more accessible under ordinary conditions than a surface installation, but it is always wise policy to provide surface equipment for emergency service. Mine fires occur so often and are so extremely costly that all the insurance possible is not too much.

Where all surface openings are required for operations, underground installation is the only solution usually considered in American metal mines, but in English coal mines surface installations involving operation of cages through air locks or double-door arrangements are common practice. These arrangements are relatively expensive where rapid hoisting through shafts is involved, but the operation of main fans underground is often much more expensive through waste of power than is generally realized.

In surface installations the greatest pressure differences between intakes and returns occur in the upper part of the mine where the stoppings ordinarily can be kept in good condition on account of their infrequent use. In underground installations the greatest pressure differences are on stoppings and doors in an active zone where they are used frequently and are difficult to keep tight. Surface installations usually result in lower-resistance air-flow systems than underground installations, as the latter usually must be placed so that the flow is confined to less than the full number of airways available. Underground installations usually require air locks on main haulage levels that might otherwise be left open and practically always result in considerable recirculation, a dangerous factor in case of mine fire and an undesirable condition under any circumstances.

FIREPROOF INTAKE AIRWAYS

Intake airways generally range from damp to dry and present a fire hazard against which it is difficult or impossible to protect the men underground, except through fireproof construction; this should, if feasible, be combined with smooth-lining for efficiency. Even with fireproof construction on the intakes, they cannot be regarded as so absolutely fireproof as to dispense with the reversible feature of the fan installation; burning materials in the openings or on stations may give off enough gases to create a dangerous condition. Fireproof construction is also desirable in main return airways, but as these are generally wet throughout such construction is mainly a question of insuring against property loss and interference with the continuity of operations.

FIREPROOF FAN INSTALLATIONS

Fan installations are in the same category as intake openings and should by all means be made of fireproof construction, as is the case for main fans at most metal mines. Operators are less scrupulous regarding secondary and auxiliary fan installations; these often present considerable fire hazards, particularly the forward-curved-blade centrifugals, which under certain circumstances overload and burn out their motors. Even the backward-curved-blade centrifugals with their constant-power characteristic are not entirely safe on this score, as without proper instruction the first arrivals may be set up to run backward or the reverse of their normal direction of rotation, in which event they put a large overload on the motor and may burn it out.

VELOCITY LIMITATIONS

High velocities often are uncomfortable or disagreeable but usually can be avoided except on the main airways. All airways having velocities above 500 f. p. m. should be lighted electrically. Carbide lights can withstand velocities up to 1,000 f. p. m. but only with difficulty. Reliance should not be placed on the lighting service in emergencies; high-velocity sections of emergency escapeways should be provided with regulators for reducing the velocities temporarily to permit safe travel with carbide lights. Dust at shift-changing periods and smoke at blasting periods often can be controlled by regulating the speed of the fan temporarily to reduce velocities of flow, by stopping the fan, or by using regulators on the main flow or on local air currents.

STOPPINGS

Virtually all devices used to control the distribution of air currents by introducing an obstruction to resist the flow or by using artificial division walls in the airway are included in the general term "stoppings", since the materials and conditions of installations are similar.

TEMPORARY BULKHEADS

A tight stopping placed across an airway to cut off the flow is generally termed a "bulkhead" in metal-mine practice. Temporary bulkheads are seldom used in metal mines except to check air currents during a fire until more permanent types can be erected. They may

be made of a variety of materials and are of various designs, depending on conditions. A skeleton frame covered with light boards is the common construction in metal mines because other materials usually are not readily available. It ordinarily is made tight by stuffing or covering the openings with clay, rags, or pieces of old canvas tubing; in rough rock airways any high degree of tightness is practically impossible except with clay. Brattice cloth is commonly used in coal mines [40] for temporary stoppings because they can be rapidly and cheaply constructed with it, but it is rarely used in metal mines because blasting concussions are stronger and because considerable labor is required to insure a good fit and minimum leakage against rough rock walls. Tarred roofing paper is probably a better material in the absence of blasting concussions, as it can be torn and wedged against the rock to make a good fit without the use of a multitude of cleats.

DEFLECTOR BRATTICES

Stoppings constructed without regard to tightness for deflecting air currents from their normal path are usually termed "deflector brattices" in metal mines or simply "brattices" in coal mines. Placed across an airway in a coal mine, they often serve to "hurdle" the air into a high spot in the roof in which explosive gas would otherwise collect; whereas, when so placed in a metal mine their most common use is to "hurdle" the air through a working place above or below the airway. Deflector brattices have been tried in metal mines for increasing the average velocity through working zones by limiting the space occupied by the flow. They usually are of rough wood construction. Canvas brattices are not successful in metal-mine stopes on account of the damage done by blasting concussions. Deflector brattices are used particularly in hot metal mines to prevent convection currents from idle dead ends entering a cooler intake airway, as such currents cause multiple small-temperature increases in the cooler air current.

PERMANENT BULKHEADS

Permanent bulkheads are used to only a limited extent in metal mines. Caved or fill material in openings is generally relied upon to prevent the flow of air through abandoned workings; stoppings are used in but few mines for this purpose. Ventilation systems and safety conditions would, in general, be improved by the use of permanent bulkheads, but in the average mine they are seldom used except for sealing off mine-fire areas. The common type in solid ground is a rough-board stopping covered with gunite—a thin coating of a sand-cement mixture applied with compressed air—over expanded metal lath or, less frequently, wooden lath or chicken wire. Although absolutely tight when first erected, it develops minor leaks from blasting concussions and ground pressures and requires periodic inspection and repair. Where a good permanent job is required for sealing fire areas over a long period it is now more common practice to use concrete wall bulkheads with or without light reinforcement. Such bulkheads usually are more satisfactory and, over a period of years, nearly always cheaper in total cost.

[40] Rice, G. S., and Owings, C. W., The Use of Brattice Cloth in Coal Mines: Rept. o iInvestigations 2872, Bureau of Mines, May 1928, 6 pp.

Comparative costs of various types of tight permanent stoppings are given in Bulletin 99.[41] The average cost of a gunited stopping for an ordinary isolated 5- by 7-foot rock airway is about $15 or approximately 40 cents per square foot. Larger isolated installations can be made for about 30 cents per square foot and large jobs of continuous guniting of airway surfaces for about 20 cents per square foot.

Wood-fiber plaster, the common material for sealing board stoppings in coal-mine practice, is seldom used in metal mines, but lime-sand-cement mortars have proved satisfactory where there are no blasting concussions. When used in highly humid atmospheres the natural clays found in some mines retain enough moisture to make a very effective sealing material against heavy concussions and ground pressures.

Where ground pressures are particularly severe even reinforced concrete stoppings crack and become displaced, and repairs are difficult and costly. Stoppings of timber packs or of timber and rock packs are especially suitable for this condition, as they crush into such a tight mass with increasing ground settlement that passage of air is not only prevented but the timber is preserved thereby.

Brick and concrete blocks are used only to a limited extent for permanent stoppings in metal mines because they cannot withstand ground pressure. Rough-board stoppings are never tight, even when battens are used, and should be covered with sealing material. Tongue-and-groove boards can be used to give tight construction but are actually no better than battened rough boards unless the edges are sealed to the rock.

DOORS

Where passage through a stopping is required only under exceptional conditions the stopping may be provided with a manhole or small hole with a removable cover, but where frequent passage for men and materials is required a door must be installed in the stopping. The door usually is almost as large as the stopping, and the installation of both door and frame is commonly termed simply a "door." Doors are used more extensively in metal mines for controlling distribution of air currents than in coal mines because, although the same difficulties of obtaining satisfactory service are presented, the dangers attending the use of doors in mines generating explosive gas are almost always absent. They are therefore used to a large extent in metal mines where stoppings would be used in coal mines, and transportation and travel are thereby facilitated.

Keeping ventilation doors shut in metal mines is one of the most troublesome operating features of any complicated ventilation system. The discipline required seems to most under officials at mines to be entirely out of proportion to the results obtained, and pressure must usually be brought to bear on them by higher officials conversant with the advantages of such installations to obtain even fair results. Under ordinary operation conditions the effects of large door leakages and of leaving doors open are not always readily apparent, and a detailed ventilation survey is sometimes required to establish their full significance. Mine fires often bring these matters forcibly to the attention of the operating force, especially when the fire gases permeate the whole mine before doors are closed and leakages reduced to

[41] Williams, R. Y., Mine-Ventilation Stoppings: Bull. 99, Bureau of Mines, 1915, 25 pp.

the minimum. The most successful and efficient ventilation systems are those that rely the least on doors, but scattered working places often require the use of a large number, particularly where the main fans are placed underground.

A mine door is more or less temporary, yet for most conditions should be as tight as possible. It receives rough use, requires frequent repairs, and allows rather large leakage through openings of apparently insignificant size. The only good door as regards leakage is an absolutely tight one with an absolutely tight frame, as there is difficulty enough in holding the leakage occasioned by the fit of door to frame to the minimum. Good sill blocks are also essential, and canvas strips should be used at least along the sill where the leakage

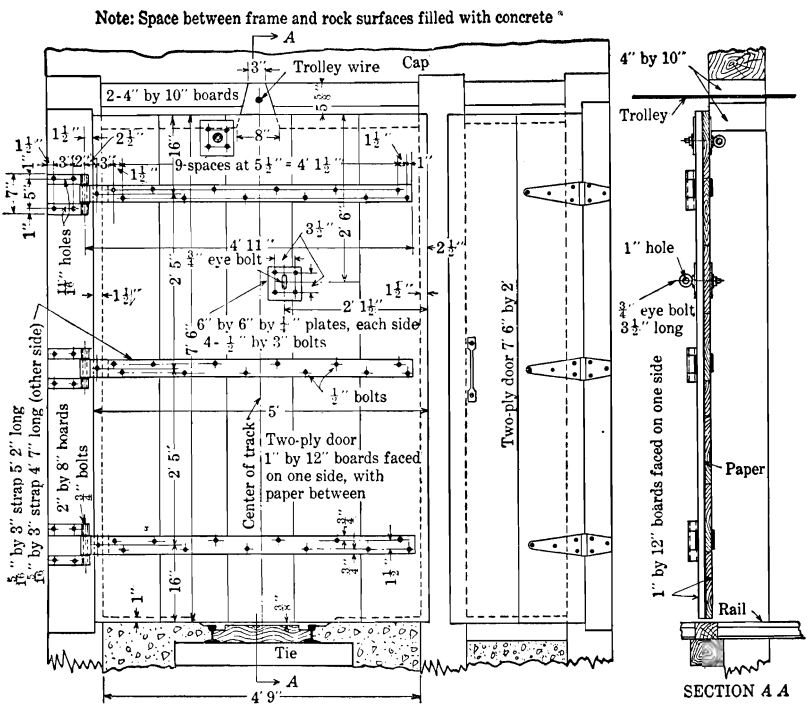

FIGURE 49.—Ventilation door used on motor haulageways in Calumet and Arizona mine, Bisbee, Ariz.

due to necessary clearance is usually large. Unless a little leakage through the door is required to keep the air clear and retard decay of timber, canvas strips or the equivalent of weather-stripping should be used on all edges. Drainage ditches without water seals and excessively large openings for pipes and trolley wires are common causes of excessive leakage through doors.

TYPES

The most common type of door in the metal mines of this country is a single, vertically hung, unpainted door made of 2 plies of 1-inch boards with cloth or paper between the plies. (See figs. 49 and 50.) One ply is placed horizontally and the other vertically. The plies

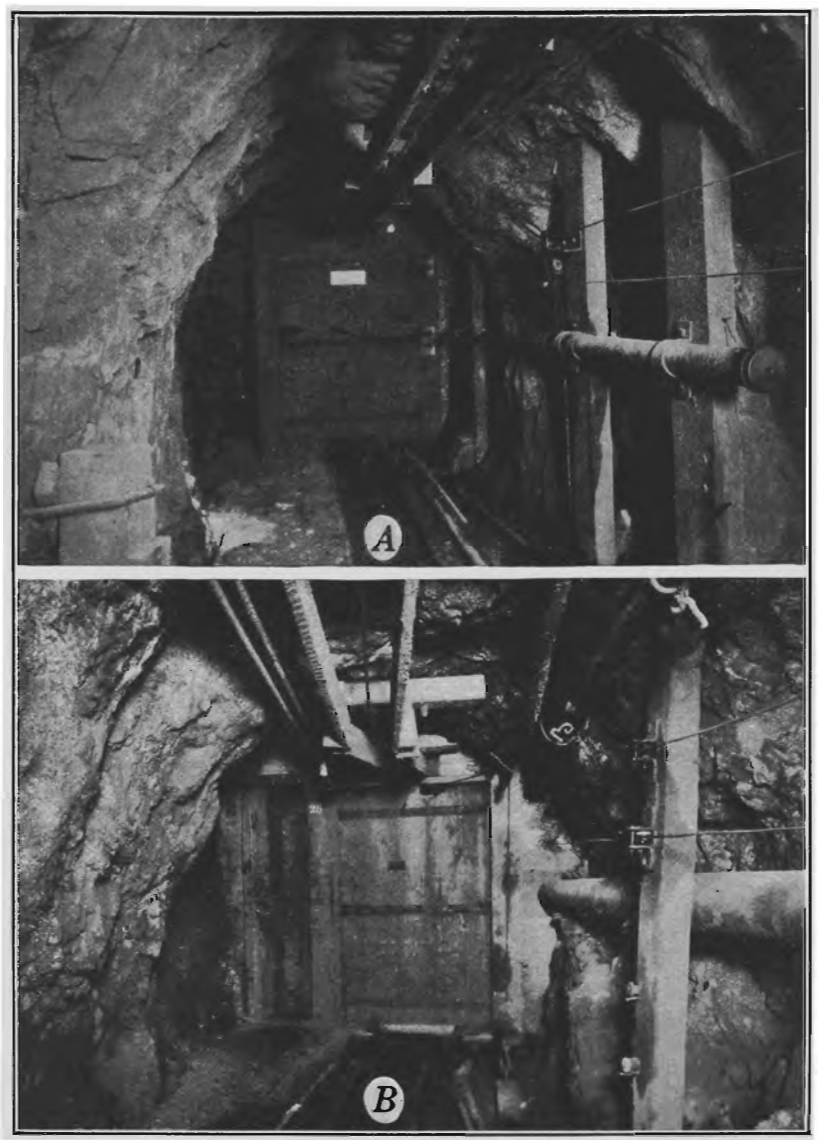

FIGURE 50.—*A* and *B*, Compressed-air-operated ventilation doors, with adjoining man doors, used on haulage drifts in Calumet and Arizona mine, Bisbee, Ariz.

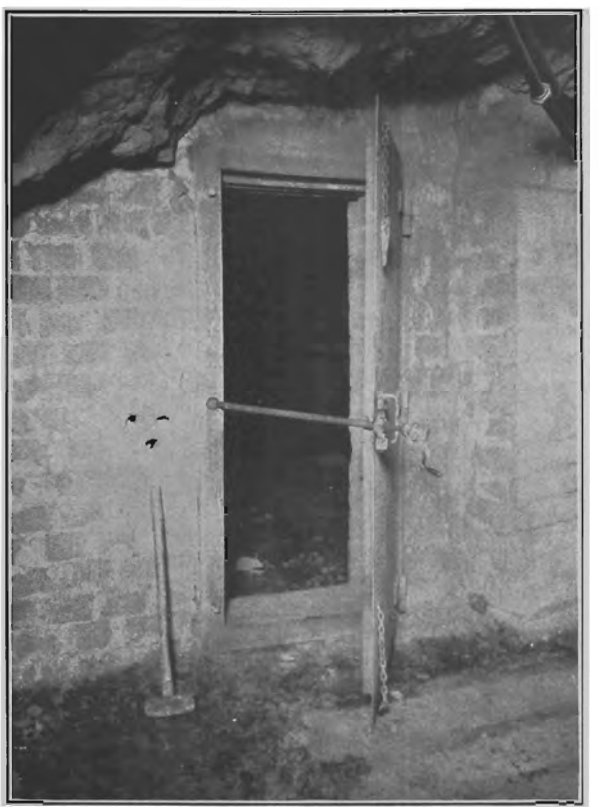

FIGURE 51.—Iron-door regulator, with feed screw setting, used in United Verde mine, Jerome, Ariz., to proportion amount of air entering each level from main-intake air shaft.

are held together by clinched nails; reinforcement strips are sometimes used but add to the weight without improving the construction. The combination of horizontal and vertical plies seems to resist warping better than diagonal plies and usually is found where the best door has been developed through trial of many designs. The door is hung on a frame of heavy timbers by 2 or 3 heavy strap-iron hinges extending the full width of the door; it can swing in but one direction and closes against the frame rather than in a jamb. In solid ground a jamb might give a better fit and less leakage, but doors are usually placed in ground subject to slight movement, and any settlement or movement of the frame would bind the door if set in a jamb. This method of flush closing requires only a facing strip on the hinge side of the frame and simplifies construction. Concrete sills sometimes are used but are hard to keep in repair. Wooden sill blocks, secured to the track tie or to a sill member of the frame, are more common; they provide clearance at the track and project 2 to 3 inches above the rails to serve as stop blocks.

Most doorframes are made of 6- by 8- to 8- by 10-inch timbers with the space to the rock walls filled with concrete or boarded over and gunited. Doors are usually provided with a heavy counterweight to facilitate opening and to keep the door closed under all pressure conditions. Printed cards often are tacked to the door to designate the normal position, whether closed or open.

Many other types of doors and frames are in use. Where leakage is not important single-ply doors made of canvas nailed to a cross-braced wooden frame have served this purpose, but they add to the fire hazard, warp readily, and seldom are used because more substantial types require fewer repairs and result in fewer interruptions to ventilation. Double doors are seldom used, even on the larger airways. Virtually all doors and frames, except those for fire control, are made of wood although, where corrosion conditions permit, light sheet-metal doors in iron frames [42] would be as effective at comparable costs. Iron doors set in iron frames embedded in a concrete bulkhead are commonly used for fire doors on timber-lined openings and are similar in construction to the regulator door shown in figures 51 and 52. These usually are provided with latches, wedges, or clamps, rather than counterweights, to insure their remaining tightly closed under all pressure conditions.

In relatively few designs is the frame set on a batter to insure the automatic closing of the door. This feature, apparently standard for coal mines, is seldom used in metal mines, except where the door is hung centrally from a timber set on a batter as respects the plane of the frame; the door can thus swing in either direction with automatic return to the normal closed position. A heavy bumper block, iron spring, or other arrangement is usually fastened to each face of the door so that it may be bumped open. These designs allow large leakages and are restricted to low-pressure conditions. An extremely sturdy door is required to withstand bumping by mine cars even for a very short time.

Two doors swinging in opposite directions are sometimes attached to opposite sides of the same frame to allow reversal of direction of air flow. In many Michigan copper mines emergency fire doors are

[42] Glaeser, O. A., New Type All-Metal Ventilation Door: Eng. and Min. Jour., vol. 132, 1931, p. 366.

hung horizontally and are normally supported in the open position against the roof on buttons attached to short rod hangers. Occasionally a sliding or roller-curtain type is adapted to a local condition.

AIR LOCKS

For positions where the pressure difference is high or where the door is open often or long enough to disturb ventilation seriously doors should be placed in pairs to form an air lock, so that only one door of the pair is open at a time. The use of air locks also reduces

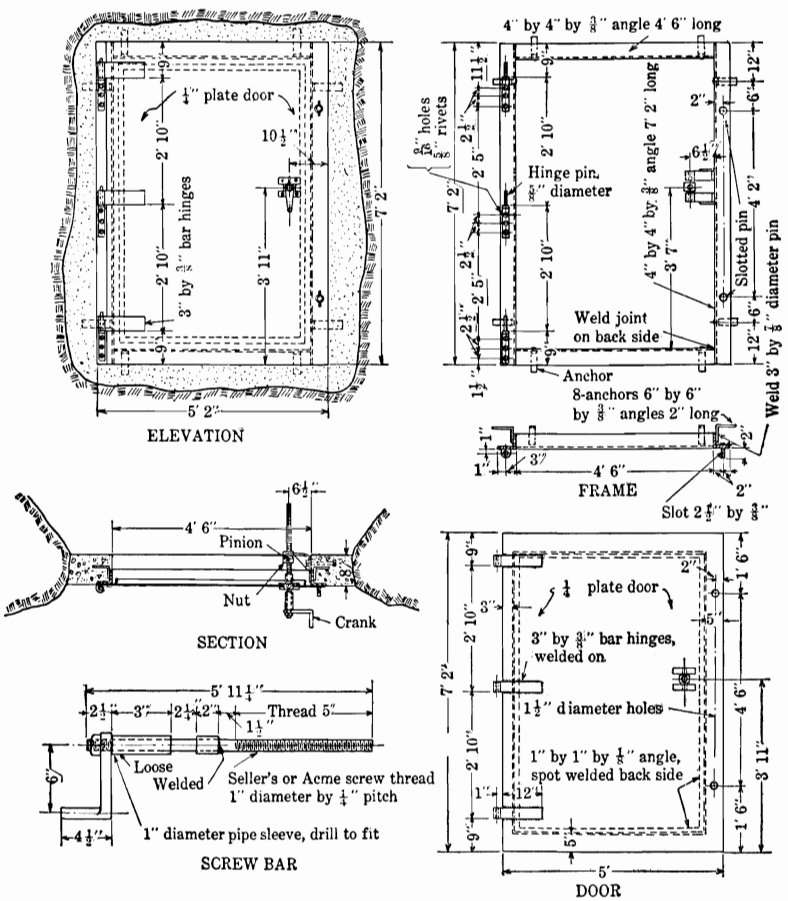

FIGURE 52.—Regulator door used at level intersections on intake air shaft, United Verde mine, Jerome, Ariz.

leakage to the minimum and enables the doors to be readily opened and closed when the pressure difference is high. They find most frequent use on motor haulageways where the doors must be far enough apart to accommodate the maximum length of train with ample clearance. Good positions for air locks are not always available if the openings have been made regardless of ventilation plans, and it is sometimes necessary to accommodate the length of the train to the length of the air lock.

AUTOMATIC DOORS

Automatic control of doors on motor haulageways is desirable; many types of automatic doors, controlled by movement of the cars, as by their weight on a false rail, by engaging a bell-crank lever attached through pulley and rope to the door, by engaging studs on an endless chain revolving on pulleys parallel to the track, or by engaging spring bows parallel to the track which are pushed out and straightened by the side of the cars, are either commercially available or designed locally and put into service. A rail-operated type, used largely in coal mines, is giving good service in the few metal mines in which it is used; in general, however, the metal-mine operator has found that his needs are served just as well and at less expense by remote-control doors.

REMOTE-CONTROL DOORS

Where opening doors by hand would be especially inconvenient and would waste time, a remote-control compressed-air mechanism usually is installed in western metal mines. These are made locally and differ mainly in size of air cylinder. The typical design illustrated in figure 53 operates the doors shown in figures 49 and 50. This design includes a small hand-operated side door for the use of the men—a desirable safety feature, as numerous accidents have been caused by single-door installations although many other devices, such as glass panels, warning lights, and compressed-air-operated whistles, are used to prevent them. The door is opened by a compressed-air piston operating in a pipe cylinder and is closed by a heavy counterweight when the pressure is released. The valve on the compressed-air line is controlled from the moving train through levers attached by long wire ropes to a single three-way valve. If there is no side door additional levers attached to the common wire-rope system are installed a few feet on either side of the door for the use of the men in passing. In curved drifts simple pipe rollers are provided to reduce friction. The wire ropes stretch and must be adjusted frequently through turnbuckles provided for this purpose. The pressure of the compressed air must be adjusted through an independent supply valve to conform with the air-pressure conditions to give free operation without too violent action. These doors require considerable attention but have proved very satisfactory as applied to a wide range of metal-mining conditions.

Some similar designs operate by water pressure; these appear to have the advantages of smoother operation and lower maintenance charges but require corrosion-resistant cylinders at a higher first cost, and disposal of the water is often a problem. The compressed-air valve of a few designs is operated by solenoid switches, the levers of which are thrown by the motorman in passing. The wire-rope control is satisfactory up to about 200 feet but has too much stretch for long-distance control. For exceptionally long trains the United Verde mine installs separate three-way, direct-throw, compressed-air valves at the operating positions, which are connected direct to the cylinder at the door; air pressure thus serves both to open and close the door.

FIRE DOORS AND THEIR CONTROL

In case of fire it is desirable to reduce the velocity of certain ventilating currents immediately without changing the directions of flow. About the only sure way to accomplish this under all conditions is to be able to shut off all main airways with doors. These are normally blocked open and used only in case of fire; they are therefore termed

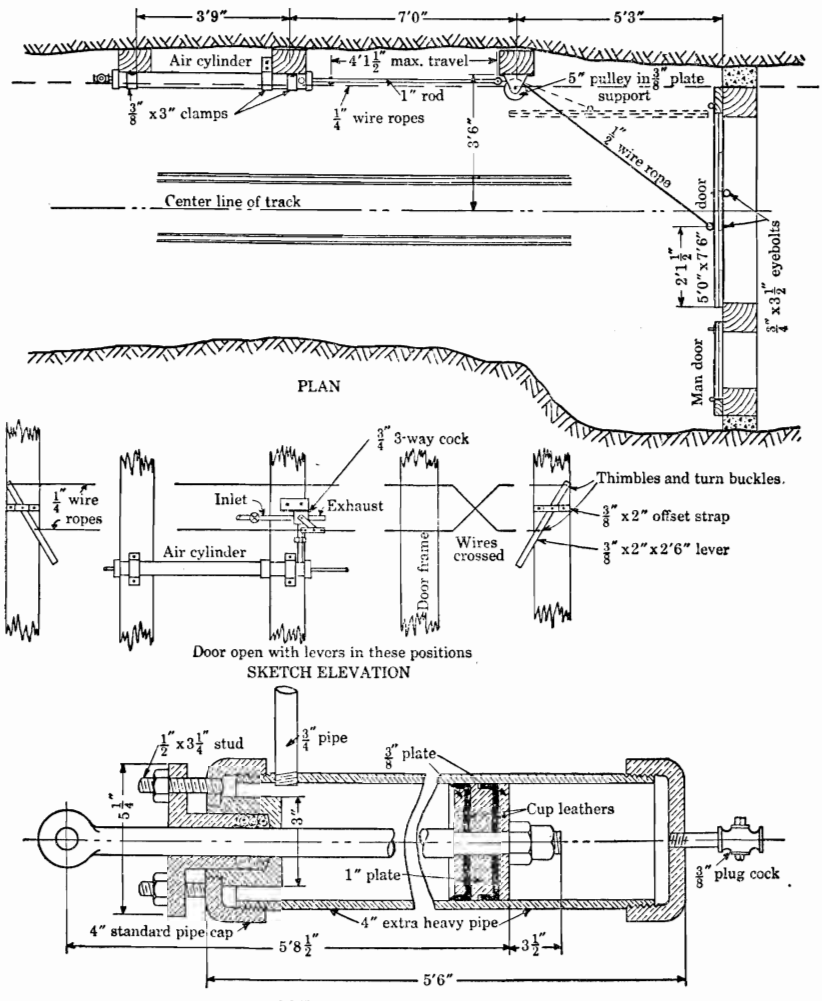

FIGURE 53.—Device used in Calumet and Arizona mine, Bisbee, Ariz., for operating ventilation doors by compressed air.

"fire doors" to distinguish them from ventilation doors, which they resemble in all respects, except that those installed in timbered sections for this purpose usually are of fire-proof or at least fire-resistant construction.

The best position for the main group of fire doors is on all the open airways off the main or operating shafts; many metal mines are so

provided, at least in theory. In practice, the connections most recently opened often have no doors; the advantage of previous installations is thus largely nullified, as a chain is no stronger than its weakest link, and the time required to close off such openings is just the time that such a system is designed to save.

In general, the fire doors are placed in position, and the operating force is relied on to use them in an emergency to cut off the flow from the fire area and thus retard the spread of the fire and fire gases. Realizing the importance of the time element, a number of mines have installed remote control on the sets of doors adjacent to particular shafts, so that all may be closed from one or more positions, particularly from the surface. Water pressure, compressed air, and electrically operated mechanisms have been installed. They take the form of a latch, which holds the door open normally against the pull of a counterweight and which is drawn back when it is desired to close the door. The latch is withdrawn from a distance by either applying or removing current or pressure. Under mine conditions continual inspection and trial of such mechanisms are required to insure their successful operation when wanted; failures through ground movement and corrosion have been common enough to retard the spread of this very useful idea. The more successful types apparently are those that rely on water or air pressure to keep the latch in position, as failure, which is indicated by the closing of the door, is soon noted. Stopping the fans does not stop the air currents in a mine, and such a set of remote-control fire doors on operating shafts is one of the most effective means of handling an underground fire with minimum loss of time, which in most instances means minimum loss of life and property.

PRESSURES ON DOORS

Pressures on single doors should not greatly exceed 1 inch of water, because of the difficulty of opening them against higher pressures, even with pry-bar arrangements. For higher pressures an air lock is desirable, as this distributes the pressure difference over both doors; or more than two doors may be used for this purpose. In practice, pressures on air locks are seldom above 2 inches of water; where the pressure difference is higher passage through the doors is difficult and possibly dangerous unless they are equipped with supplementary small doors through which a man can crawl in an emergency. The small door can be opened easily, even though the pressure is high, and its opening equalizes the pressure on both sides of the large door; the large door may then be opened and closed easily. After the first door of the air lock is passed and both large and supplementary doors closed the process is repeated at the second or succeeding doors of the air lock. Doors under high pressure may also be equipped with mechanically operated devices, such as compressed-air or water-pressure cylinders; such installations are desirable on frequently used doors that operate against pressures greater than one half inch of water. To facilitate pressure-difference determinations at doors during surveys and as a check on leakage, a ½-inch-diameter hole through either door or frame should be provided; the same idea should also be applied to stoppings, particularly at underground- and surface-fan installations. The leakage through such a hole is

insignificant—the resistance factor is about 45,000,000, and the leakage at a pressure of 1 inch is only about 5 c. f. m.

COSTS

A few data collected in 1928 in Arizona copper mines indicate that costs for substantially constructed and installed doors and frames with about a 4.5- by 6.5-foot opening are about as follows: Replacing door only, $25; door in concrete frame, $50; door in concrete frame, air-operated with remote control, $175; air-operated door with adjoining narrow man door, $225; and water-operated door with adjoining narrow man door, $250. An automatic door with adjoining man door costs approximately $400. A less substantial wooden door and frame of average size can be installed for $15 to $25.

LEAKAGE

Leakage at stoppings, particularly doors in active sections and at fan installations, is a common cause of inefficient distribution of air in mines; it results in poor ventilation of working places, waste of power, and where the leakage is recirculated increases the hazard during an underground fire. Actual tests in a large number of coal mines show that in the average mine not more than 30 to 50 percent of the air handled by surface fans reaches the active workings. Average conditions in metal mines, on account of the general differences in distribution lay-outs, are somewhat better, probably ranging from 50 to 60 percent for the average mine and lay-out to about 85 percent for the best. An allowance of about 15 percent for unavoidable leakage is the minimum that is ordinarily practicable under the best lay-out. If all main fans are underground and close to active workings it is almost impossible to avoid considerable recirculation; ordinarily, such fans handle 1½ to 3 times the amount of air that actually enters the mine. In a warm or hot mine increased velocities usually improve comfort; in general, however, recirculation of air currents results in increased air temperatures and decreased comfort.

Leakage and recirculation can be largely avoided by a properly planned lay-out, substantial construction, and close attention to details. Individual leakages are often insignificant and difficult to detect, but the cumulative effect of small leakages is by no means insignificant; it is therefore imperative to use airtight construction wherever possible, as unavoidable leakages will cripple the system enough without loading it with avoidable leakages.

FAN INSTALLATIONS

Pressure differences are highest at fan installations, and leakage is particularly important—more important, in fact, than the average construction crew realizes. Steel-plate housing and connections, even though carefully erected by a reliable crew, are rarely tight. Leakage conditions vary, particularly with the amount of surface exposure and the number of doors. At pressures of 3 to 4 inches of water the leakage at a single-inlet nonreversible fan with short duct connections may be as low as 1,000 to 2,000 c. f. m., but that at the usual large, reversible, double-inlet installation offset from a shaft collar usually is as high as 3,000 to 10,000 c. f. m., depending on the use of air locks on openings and the proportion and surface exposure of steel plate to

concrete or other airtight construction used in the housing and connections. Wooden ducts, even though gunited, generally are poor construction as regards leakage and add to the fire hazard.

Fans often are installed at shaft collars built up of waste rock from sinking operations. Unless the shaft is lined to solid rock with concrete or its equivalent such a rock fill will "leak like a sieve", and the fan installation will be a failure or at least require remedial measures, in the way of sealing the shaft walls, that are not only expensive but usually effective over but short periods. Concrete linings sealed to solid rock will show a high first cost but will practically always show a lower final total cost at main-fan installations.

SHAFT COMPARTMENTS

The practice of carrying main intake and return currents in adjoining shaft compartments, although common in coal mines, is comparatively rare in metal mines. In theory leakage can be held to the minimum by airtight construction, but in practice such lay-outs have proved very inefficient on account of leakages. They add to the danger in case of fire and are properly condemned by mine safety engineers.

ADJOINING AIRWAYS

The practice of carrying main intake and return currents in parallel airways separated by a short distance and connected by stopped-off openings at frequent intervals, although common in coal mines, is also comparatively rare in metal mines. In general, surface openings at metal mines are considerable distances apart; but where two are close together the general practice is to use them as parallel airways with free intercommunication and, if no other surface opening is available, to drive a new opening, usually a raise, or raise to an adit solely for ventilation. With surface-fan installations the pressures on the stoppings and doors separating parallel airways usually are high, and leakages are difficult to control, particularly if operations require frequent passage of men and materials from one to the other.

RESISTANCE FACTORS FOR LEAKAGE CONDITIONS

A leakage path is simply a path parallel to the main flow of which it is a part, and resistances to flow are encountered on this path just as well as on airways. Leakage paths usually involve flow through small orifices at such velocities that the pressure loss conforms to the square law, or the same pressure-quantity relation as that for airways. Where flow through finely packed material is involved the velocities may be so low that the quantity varies almost directly as the pressure (see p. 69), but this condition is relatively unimportant in metal-mine ventilation and may be ignored.

Resistance factors for leakage conditions are relatively high, indicating relatively high resistance to flow. If a stopping is so tight that no flow can occur, regardless of the pressure, the resistance factor R is infinity; but some air practically always leaks through stoppings, and actual values of R, though large for tight stoppings, decrease rapidly as leakage increases. At one end of the resistance scale are tight stoppings for which the leakages are measured in units and tens of cubic feet and the resistance factors in millions or hundreds of thousands; these leakages are so small that they may be ignored. At

the other end of the resistance scale are regulators or stoppings constructed or used to provide a definite resistance to flow; the resistance factors for these are generally less than 10 (see p. 151). Between these extremes are hosts of leakage conditions, such as those represented by doors, joints in fan housings and connections, and tubing connections, for which the resistance factors may vary with time and pressure, but for which approximate values may be obtained. Heretofore, leakages have been either ignored in mathematical solutions or handled by a system of rough allowances. Actually, they can be put on a comparative basis through the use of resistance factors, and mathematical solutions are thereby facilitated. Leakages often cannot be measured, but where pressure and volume measurements can be made they are not only much more reliable than visual inspection for determining the actual condition but also serve as a base for estimating conditions where the leakage, although an important factor, cannot be measured directly.

RESISTANCE FACTORS FOR DOORS

The higher the resistance factor for a door the better it serves its purpose, except where a small leakage is desirable, in which case the door acts as a regulator. A good door and frame without any special precautions against leakage should not pass more than 3,000 c. f. m. under a pressure of 1 inch of water. The corresponding resistance factor is about 1,100. Under many conditions such leakage is permissible, but where many doors are involved or where a door is seldom used this leakage is larger than desirable. With the same door weather-stripped with pieces of heavy canvas, particularly at the bottom where most of the leakage occurs, the leakage might be reduced to 500 c. f. m. at a pressure of 1 inch; the corresponding resistance factor is 40,000. Old belting or similar materials may also be used as weather-stripping to reduce leakage. Doors that do not have tight frames—those with wooden frames stuffed with rags or with tight frames carried only to the lagging instead of the rock surfaces—may have resistance factors as low as 300; where other major leakages occur concurrently, as through lack of sill blocks, unsealed drainage ditches, holes for pipes, and excessively large trolley-wire openings, the author has obtained factors as low as 30 for doors that at a casual glance would be considered pretty fair installations.

REGULATORS

Regulators are stoppings in which the size of the opening purposely provided is adjustable, so that the resistance offered to flow may be adjusted, in series with the normal resistance of the airway or section, to allow only the quantity of flow desired. The position for a regulator should always be selected with regard to creating minimum pressure differences on stoppings—on the intake side of the circuit in a pressure system and on the return side in an exhaust system.

Regulators are used extensively in coal mines but rarely in the average metal mine. A few metal mines, however, use specially designed regulators (see figs. 54 and 55). Most metal mines could use them to advantage, particularly for controlling distribution through stopes, as often 1 stope of a series between 2 levels will take most of the air flow on account of its relative position and size of openings. Opera-

tors realize the conditions, but many have abandoned the use of regulators on stopes because of the difficulty of keeping them in place on active fill raises and manways.

The operator of the average metal mine wants all the air he can get to a certain working zone or practically none at all. Distribution, therefore, usually is regulated by reducing the resistance to flow rather than increasing it and thereby reducing the total quantity in circulation. Regulation by adding resistance has been confined mainly to inactive openings where a little flow is sufficient for the work in progress or is required to prevent falls or excessive decay of timber. Usually a door is fastened open, blocks being placed at both the cap and sill under high-pressure conditions to prevent undesired warping of the door. Many metal mines have used regulator slide openings in doors that, after installation, were found to pass more air than desired with the slide closed.

Since a regulator is merely an orifice in an airway pressure losses and resistance factors can be calculated for constant conditions from data on orifice pressure losses (see p. 66). Few data are available, except for symmetrical square-edge orifices; although the author has developed a chart [43] for determining the area for particular conditions of use of this type, in general practice the conditions of possible use are more or less unknown at the time of installation, and flexibility in use requires that a large opening be provided, which can be adjusted in service to give the desired results more quickly and accurately than the adjustments could be calculated. Even though enough data on pressure losses were available for the particular type and conditions of installation, it would usually be wise to make a larger opening than required to provide for unforeseen demands, particularly in metal mines, as they rarely can control distribution with such precision as coal mines. The approximate area of a regulator opening, in square feet, for average mine conditions where the opening required is not more than about 20 percent of the area of the airway is:

$$\text{Area} = \frac{1{,}510}{R}, \tag{36}$$

where R is the resistance factor for the resistance added by the installation.

CROSSINGS

Crossing of air currents in metal mines is rarely necessary because the vertical openings are in different planes or the surface openings are more or less regularly spaced where the workings are all in one plane; where it is required a few doors or bulkheads are usually all that is necessary. Elaborate structures, such as the overcasts and undercasts used in coal mines, are seldom encountered. Coal-mine practice could sometimes be followed to advantage, but with few exceptions the only separate structure used is a pipe carrying a minor return across or through an intake current with which, on account of high temperature, possible gas content, or possible fire hazard, it is considered undesirable for it to be mixed. The pipe ordinarily extends between 2 doors or stoppings erected in a return airway, 1 on either side of an intake airway.

[43] Weeks, W. S., The Air-Current Regulator: Trans. Am. Inst. Min. and Met. Eng., vol. 76, 1928, p. 145.

EFFECT OF MINING METHODS ON DISTRIBUTION

The exact conditions of distribution at working places where the ore is extracted depend largely on the details of the mining method, both as to position and type of openings and time sequence of driving openings. In general, the larger and more numerous the openings the better the air conditions at working places, and the larger the opening the farther it can be driven as a dead end with comparable air conditions. However, the primary necessity for good ventilation is two or more openings to all working chambers, usually termed "stopes" in metal mines, so that through-circulations may be established. Quantity of flow usually is less important than velocity of flow, especially for removing blasting smoke to the main flow, where it may be diluted, or for providing air-motion cooling effect in warm atmospheres. Quantity of flow is increased as openings are increased in number and size, but velocity of flow is increased by restricting the area of cross-section of the openings. The openings should therefore be the maximum size at all other parts of the circuits and the minimum size at working places. Exactly the opposite condition normally obtains with all mining methods, since the working places are large openings and the connections to them relatively small. The velocities of flow in working places are thus relatively low, but they can be increased if desirable by proper attention to details of size of openings used in mining or by a change of mining methods. This feature usually requires little attention because even very low velocities on through-circuits, together with diffusion and convection currents in openings off through-connections, usually suffice to move blasting smoke out of the working places in the interval between 1- and 2-shift operations; but with continuous operation, blasting during the shift, high-temperature conditions, or gas conditions it cannot be neglected.

Where the working places and their local connections are all on the same horizontal plane the positions of the connections to a working place are not particularly important if there are two or more, but where the working places are on an inclined plane, as in metal mining, the relative positions of the connections with respect to differences in elevation are important on account of natural-draft and convection-current effects. Two or more openings to the same airway rarely produce the desired conditions, except occasionally where the connections are from an airway to a working chamber below it. The essential requirement is that the connections be made to airways above and below the working opening, so that ascending through-circuits are possible and maximum advantage is taken of natural-draft pressures. The next essential requirement in utilizing available flow is that the position of the through-openings be offset with respect to the working opening so that the actual working place is not a long dead end off a through-opening.

In general, systems of mining that normally result in large openings and large quantities of flow at low velocities are used on ore deposits that have low rock-temperature gradients and normal air vitiation only, for which the general conditions of low-velocity distribution give satisfactory results. Systems of mining that normally result in small openings and small quantities of flow are usually applied to deposits that require relatively restricted openings at working places owing to the necessity of back-filling openings, which automatically

increases the relative velocities of flow through working places. Increased velocities are usually required in mines of this type by such natural conditions as high rock-temperature gradients and gases resulting from oxidation or coming from the rock strata.

OPEN-STOPE METHODS

Ground that can be worked by open-stope mining methods usually is easily ventilated, as the requirements are low and many large openings usually can be incorporated in the mining method without perceptibly increasing costs. The essential requirement is at least one opening from each large stope or working place to the level above, as indicated in figure 54, *A*. Velocities of air travel are usually extremely low, but when blasting is confined to the end of the shift the products of general blasting have only a short distance to travel before reaching inactive openings through which their time of passage is relatively immaterial.

Depths of working sufficient to encounter high temperatures usually also require changes in mining methods, as from advancing stoping to retreating stoping, and openings cave more readily, automatically confining the flows to active openings where the requirements gradually increase because of higher temperatures. Recently caved ground adjacent to active openings usually is relatively open as respects air flow and constitutes a good second opening of low resistance for the retreating working faces, as indicated in figure 54, *B*.

At considerable depth the returns may be depended upon to rise to more open caved ground above and eventually reach the upcast main opening. Even in ground that eventually becomes quite compact considerable air usually will flow through caved sections but will be concentrated along the edges of pillars and solid ground. Eventually, with increased depth it may be necessary to provide special return airways connected to the caved ground or to partly maintained openings through it at strategic horizons, as at every third level where the retreat is made in groups of three levels.

SHRINKAGE-STOPE METHODS

Ground that can be worked by temporary-fill methods, such as shrinkage-stope methods, is also easily ventilated, since the openings usually embrace at least one at the bottom and another at the top of the stope. The working opening is also restricted, and with blasting confined to the end of the shift the small quantities and low velocities of the normal air flow result in good air conditions, sometimes good enough to permit occasional blasting of large slabs during the shift without detriment to the general air conditions, although the blasting of boulders generally is confined to the half-hour lunch period or end of the shift. Safety requires at least two openings to each stope. Good ventilation demands that the openings connect both to the level below and the level above, the usual practice (fig. 55, *A*). In continuous shrinkage stoping the development openings driven for chutes in advance of active stoping sections usually provide the necessary bottom opening, but even where the chutes have a shallow depth of coarsely broken ore enough air will seep through a small number of such chutes to ventilate the stope adequately under many conditions. However, where a floor pillar is left under the level above,

it is necessary to pierce the pillar at intervals to avoid long and poorly ventilated dead-end sections in the stope. Shringage stopes driven without an opening to an upper level usually are poorly ventilated,

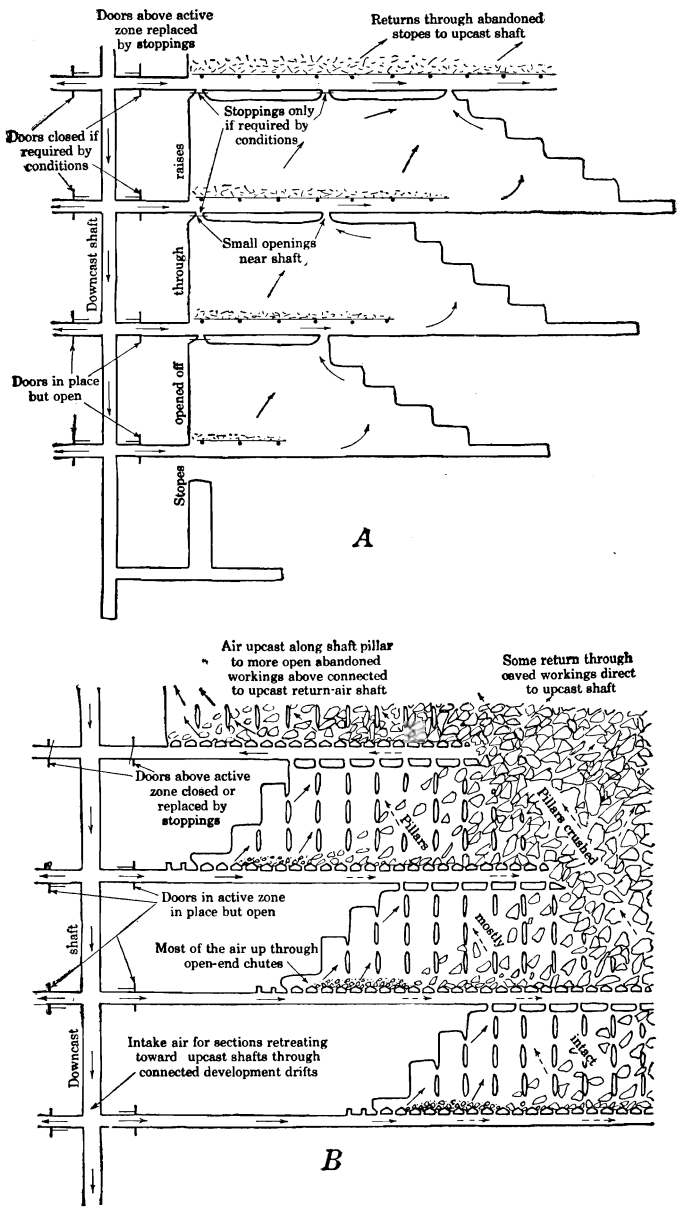

FIGURE 54.—Method of ventilating open stopes: *A*, Advancing; *B*, retreating.

particularly where the rock in the stope is warmer than the air on the airway to which it has two or more connections. When the wall rock of the stope is colder than the air on the airway natural-draft

currents which may be quite effective are set up. Ore pulled from chutes draws some air from the airway into the stope by displacement, but this is a slow process for a large stope opening although it is a

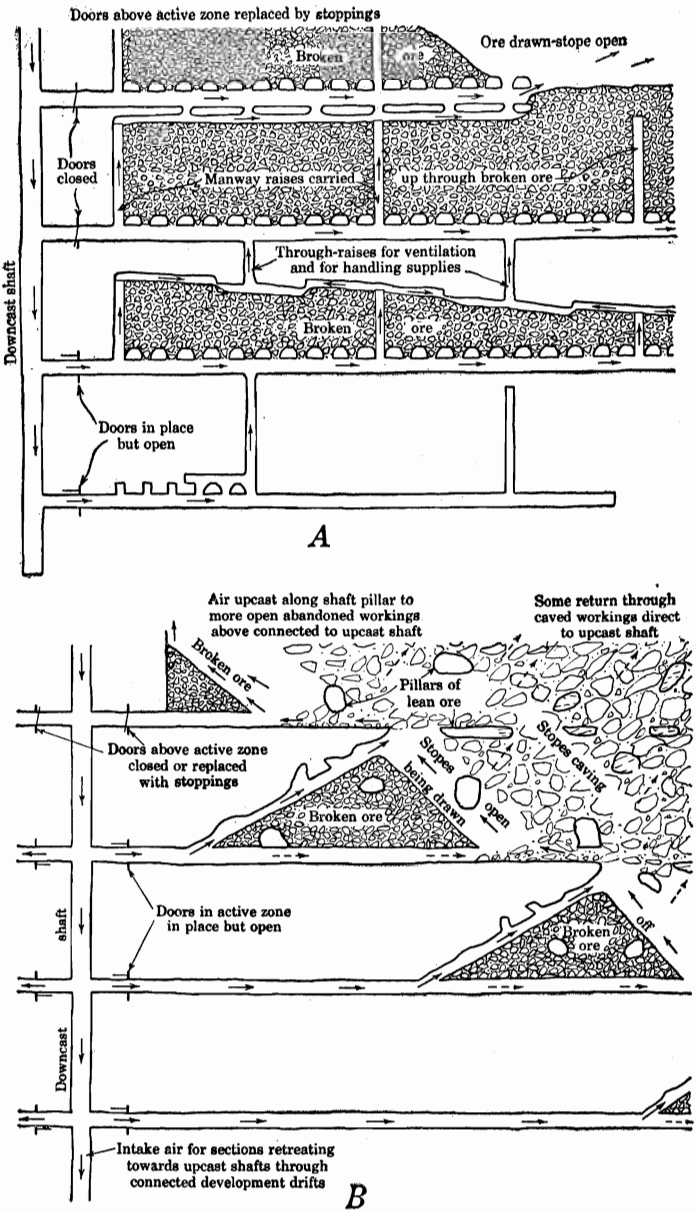

FIGURE 55.—Method of ventilating shrinkage stopes: *A*, Advancing; *B*, retreating.

fairly effective means of ventilation at the tops of manways adjoining chute compartments through which large quantities of ore are drawn,

as for chute-control stations at the tops of dead-end raises in undercut-caving operations.

Often the ventilation of shrinkage stopes that do not connect by raises to the upper levels could be easily improved by erecting a loose curtain or canvas door on the drawing level between the end openings and hurdling some of the air through the stope. (See p. 140.) This method is seldom used in metal mines but could be used to advantage.

Retreating methods of shrinkage stoping on moderate slopes provide exceptionally good ventilation conditions (fig. 55, *B*), as the working chamber is a relatively confined space with large openings to both the level below and the level above and the slowly caving ground back of the broken ore makes a low-resistance return. If desired, the flow can be confined to the stopes by temporary stoppings or doors inside the last active drawing chute.

FILLED-STOPE METHODS

With most of the mining methods that require filling soon after extraction the natural lay-outs for mining are favorable for ventilation of stope openings, since openings from the level below and the level above are required. However, the ground conditions are likely to be such that the airways of the main circulations are relatively small, and likewise the quantities, even though mechanical ventilation is used to augment them. The larger compartments of the inclined openings are generally full of ore or fill so that the actual openings for circulation depend on the manway compartments being kept open. Usually two manways are kept open for safety at all times. In the early stage of stope operations both manways may lead off the lower level, and in the later stages the only manways maintained may lead to the upper level. Again, the one raise manway may lead directly from level to level; the stopes must then depend largely on eddy currents off this manway for ventilation. Good ventilation usually requires that the details of the method be laid out with ventilation in mind so as to have ascentional flow with the air coursed through the actual working area, as shown in figure 56, *A*. Manway design is particularly important, as the free area provided largely determines the amount of air flow. A good design used in many western metal mines is shown in figure 57. With increase in velocity requirements, as in high-temperature ground, the size of the working openings must be kept to the minimum; filling must therefore follow extraction closely, regardless of the actual necessity of the fill for immediate support. Rill methods of fill stoping generally fulfill this requirement naturally and favor good ventilation, as indicated in figure 56, *B*. Retreating methods of rill stoping are particularly good with respect to facility with which the stopes may be ventilated.

TOP-SLICE METHODS

Top-slicing methods, in general, provide about the poorest ventilation conditions—dead-end workings with abnormally high ventilation requirements due to the heat generated by the decaying crushed timber in the mat. Good results can be obtained only by paying considerable attention to details of ventilation in laying out the openings, and special openings for ventilation must often be incorporated in the system. Three general methods are applicable, according to the particular conditions—hurdling from the level below by means of a

well-arranged system of stoppings and doors; connecting at each floor in succession to a side raise in an adjoining section developed for mining by this or a different method but connected with the return airway; and connecting from the open space above a light mat and fill to return airways. About 1920 the Miami mine (Miami,

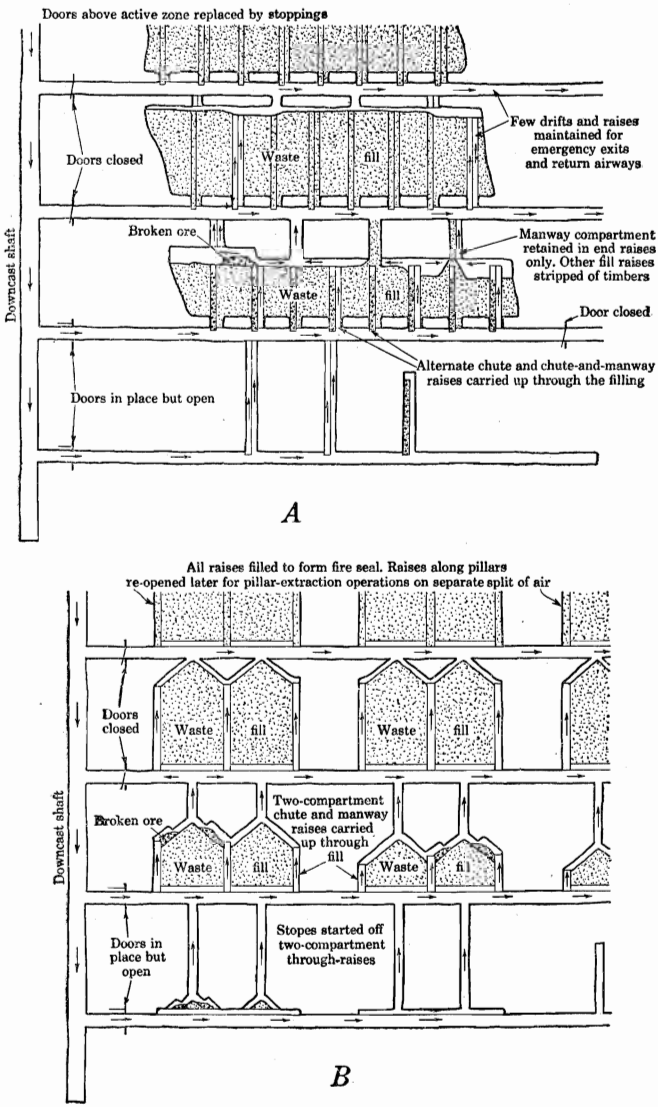

FIGURE 56.—*A*, Method of ventilating horizontal cut-and-fill stopes; *B*, method of ventilating rill cut-and-fill stopes.

Ariz.) used a ventilation level of closely spaced openings provided with a multitude of doors and stoppings for hurdling air through closely spaced raises to extensive top-slice operations by means of a mechanical ventilating system and obtained excellent results that

permitted continuous blasting in the top-slice stopes. In general, the hurdling method (fig. 58) is applicable to the common conditions encountered, whereas the other two are applicable only to special

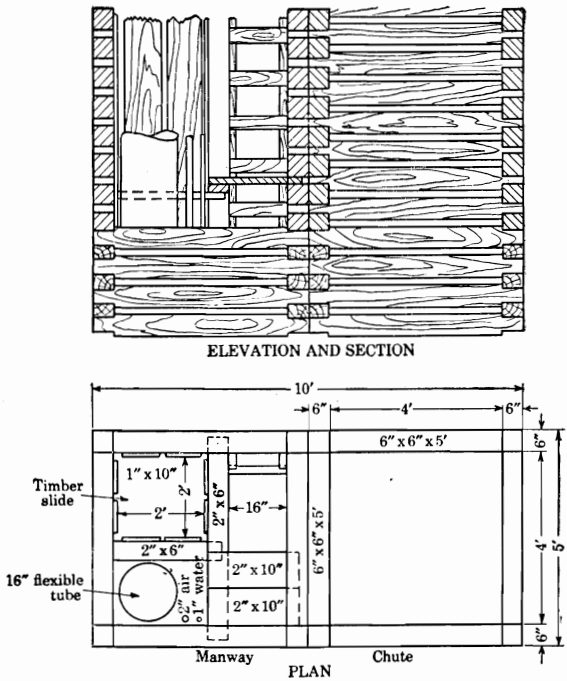

FIGURE 57.—Two-compartment cribbed raise with timber slide in manway compartment.

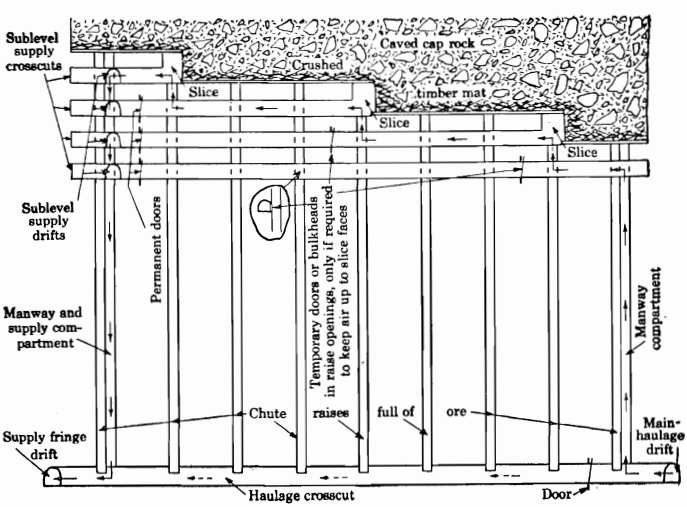

FIGURE 58.—Hurdling method of ventilating top-slice stopes.

conditions of regularly advancing or retreating methods or of mining small sections or pillars in ground that stands well.

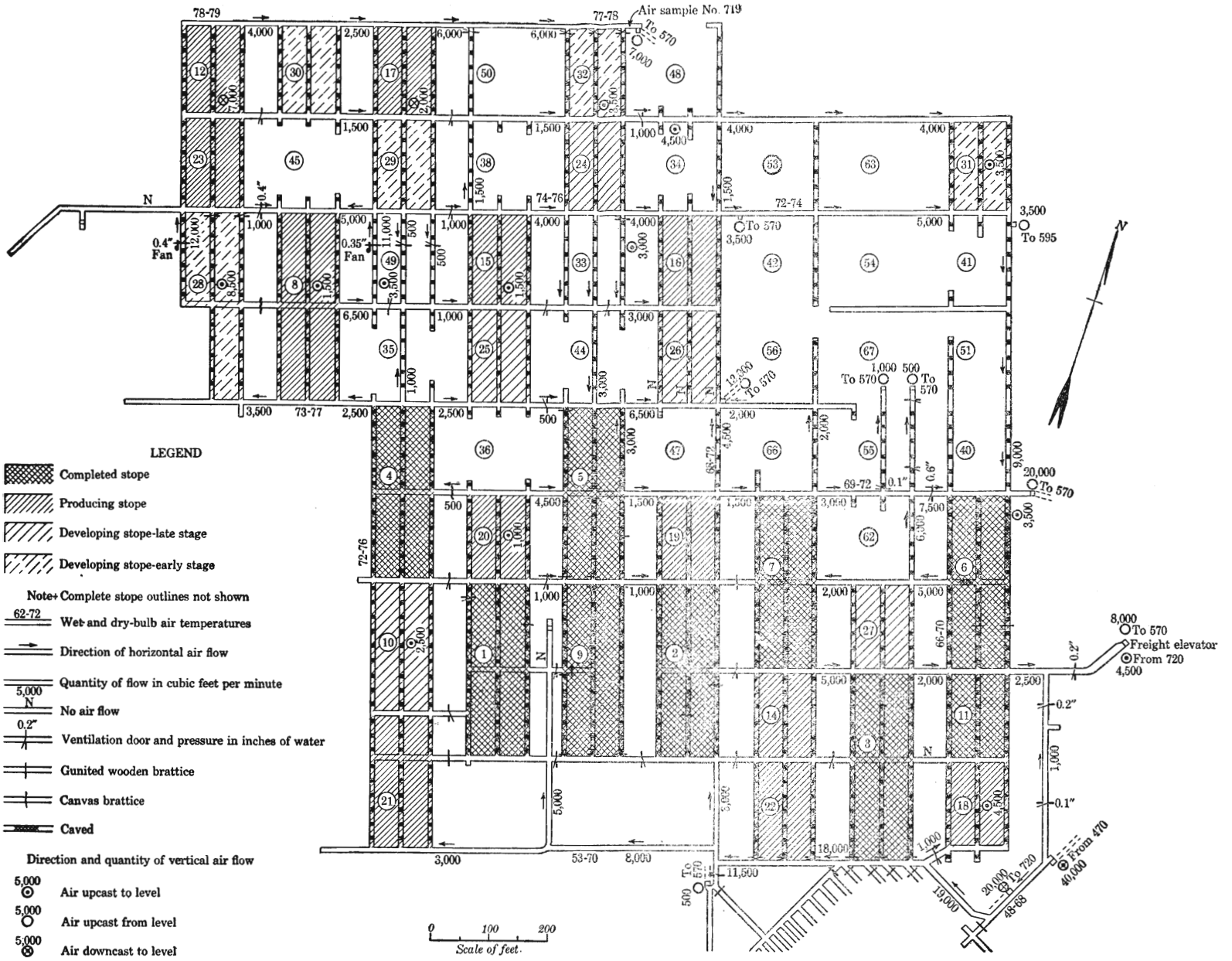

FIGURE 59.—Ventilation map of 620 grizzly level of Miami mine, Miami, Ariz., May 1929.

UNDERCUT-CAVING METHODS

The undercut-caving methods used so extensively in Arizona and Nevada require the driving of many relatively short dead ends in development that are difficult to ventilate; fair ventilation is obtained, however, by limiting operations in certain classes of working places to one shift. All these mines have mechanical systems that provide for rapid clearing of the grizzly drifts, where almost continuous blasting is required, and other active operating places. With extended experience on certain types of ground the need for continuous blasting is gradually decreased as methods for obtaining more uniform crushing of the ore are developed. The chief requirement for the ventilation of active operations seems to be that the main returns be connected direct to the active grizzly drifts and that there be no direct connections between the main returns and the main-haulage level; the latter should be connected only to the intake airways. The general plan is that the intake currents supplied to the main-haulage level rise to the grizzly level through the raises of new development only and pass to the active drawing drifts and from them to fringe drifts connected to the returns. The difficulty of keeping the raises in drawn and drawing sections full of waste or ore makes control difficult; if all but the active drawing drifts of the grizzly level are closed at the fringe drifts by doors or stoppings fair control may be obtained for certain lay-outs. The general lay-out, however, is often such that simple methods are not applicable, and the Miami mine in 1929 was considering including a special ventilation level for a new section so that the ventilation could be controlled more accurately. Figure 59 shows a ventilation map of a typical grizzly level of an undercut-caving operation.

Development for this system of mining involves many openings without upper connections, unless these can be provided by connecting to old prospect openings, which are often present but irregularly distributed. The numerous openings between the undercut and grizzly levels usually induce small convection and natural-draft currents which supply enough circulation to permit one-shift operation; but, in general, the conditions in boundary caving shrinkage stopes and drifts are poor unless a well-planned system of connected openings is incorporated in the general system with ventilation in mind.

BOOSTER FANS

The development of metal-mine openings is often so irregular, on account of the variations in occurrences of ore bodies, that the resistances of the separate districts or splits into which the flow may be divided vary widely. Certain districts may offer such high resistance to flow that they naturally receive a small percentage of the total flow. A larger flow may be obtained (1) by reducing the resistance of the split by actual changes in the physical characteristics of the airways, as by smooth-lining, removing obstructions, easing sharp bends, and increasing areas of cross-section; (2) by increasing the resistance of the other sections until the desired proportional flow is obtained in each and then increasing the speeds of the main fans until the total flow, which was reduced by the added resistances, is brought up to the desired quantity; or (3) by generating pressure directly on the high-resistance split, as by a fan. Under average metal-mine

conditions the third method is generally the cheapest and most effective, and secondary fans operating in the booster position (booster fans) are used freely. They regulate the flow by reducing the equivalent resistance of the section with respect to the main-flow system.

FAN INSTALLATIONS

Booster fans present practically the same conditions as any fans underground but usually are installed in active workings where provision must be made for the passing of men and ore trains without disturbing the flow. Where two openings are available the fan is placed in one with discharge through a tight bulkhead, and an air lock is placed on the other. Where there is only one opening at the desired location a run-around drift is sometimes driven to take the fan installation; where the ground stands well it generally is cheaper to widen the drift over a sufficient length and provide an airlocked passage alongside the fan installation. Figure 60 shows a typical lay-out of the latter type, which could be improved somewhat by carrying the full width of widening to the end of the air lock on the discharge side. Where the fan required is small and passage requirements are limited to single cars the fan is often mounted against the

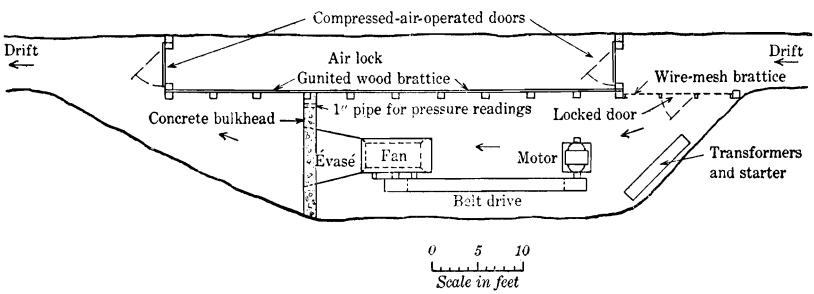

FIGURE 60.—Typical underground installation of fan in booster position.

wall at a short air lock on a drift or crosscut, and the discharge is carried through the air lock by a short length of pipe having about the same area as the fan-discharge connection. Provisions for minimum resistance to flow are rarely given much thought at booster installations; the resistance added to the section by the fan installation is often nearly equal to the natural resistance of the section. Leakage at booster-fan installations is also neglected in most metal mines and is often a high proportion of the total duty of the fan. Likewise the necessity for fireproof installations is not generally realized, and serious fire hazards result.

POSITION IN OPENINGS

The position of the booster fan is largely controlled by the lay-out of the openings available. Minimum resistance to flow and minimum leakage of return currents to intakes are the major factors to be considered. Minimum resistance requires that the total flow, necessarily confined to 1 airway at the fan location, be limited to 1 airway over as short a distance as possible and that maximum use be made of the airway areas available. Minimum leakage of return

air to intakes generally requires that the booster fan be on the same side of the ventilated openings as the main fan, that is, on the return side with a surface exhaust fan and on the intake side with a surface pressure fan. Minimum leakage also requires that the fan position be in relatively inactive territory, but this condition must often be subordinated to the others mentioned.

The requirements listed also apply, in general, to selection of the position of a main fan in series with another surface fan except that, if possible, the fan in series should be installed on a surface opening for maximum control in case of fire. Seldom, however, is more than one surface position available, and the driving of separate openings solely for ventilation is often beyond economic consideration.

SELECTION

Determination of the duty of a booster fan by the operator is usually a rough estimate based on previous experience, or a selection is made from the fans on hand; the actual duties of the installation often are highly variable because of train movements, door-controlled distribution, and changes in number and conditions of openings. It is therefore desirable to install a type having a nonoverloading power characteristic and provide for speed changes. The general selection is a constant-speed motor with a short-center-type drive; speeds are changed by changing pulleys. Multispeed motors would be desirable under most conditions but have not been used for this purpose.

For fairly constant resistance conditions the approximate resistance of the split circuit with the fan installed may be calculated, and after allowance for normal leakage the approximate duty of the fan may be set at this figure. The actual duty of the fan is affected also by its relation to the pressures generated on the main-flow circuits; for most installations, however, this effect may be disregarded, and the approximate duty may be determined accurately enough with reference to its own split. This statement, of course, does not apply to fans operating underground in series with a surface fan and handling all or a large proportion of the total flow. These are often referred to as booster fans because they boost the pressure on the main flow, but they are not strictly booster fans in the sense in which the term is used here.

Whereas main fans usually operate against pressure-quantity relations expressed by resistance factors of less than 10, the equivalent factors for booster-fan operation generally fall in the range of 20 to 200; this fact explains in large part the average negligible effect of main-fan operation on the duties required of booster fans.

AUXILIARY VENTILATION

The general designation "auxiliary ventilation" may be applied to all methods of ventilating unconnected openings, which are commonly termed "blind" or "dead-end" openings, although in practice the use of this term has been limited to fan-tubing methods of auxiliary ventilation. The length of dead-end opening to be ventilated is of primary concern; it is desirable therefore first to discuss the effect of methods of extending openings into new territory, as required for prospecting and development, in limiting the length of dead-end openings.

DOUBLE-ENTRY METHODS

The general method of extending workings into new territory in coal mines, as developed by experience for maximum control of explosive gas given off by the strata and later incorporated in State laws, is by the double-entry method (upper part fig. 61) of driving parallel openings in coal, connecting these by cross-openings at uniform intervals as they are driven, and sealing all but the last cross-connection. One of the parallel openings is used as an intake and the other as a return airway, and air is circulated close to the advancing faces, which are extended but short distances as single openings. The distance, usually specified by law, is seldom over 100 feet for the opening, including the cross-connection. Multiple-entry methods or methods using some multiple of two parallel openings are also used in coal mines to provide larger air-carrying capacities and to avoid the disadvantages of relying on the tightness of numerous stoppings separating main-intake and return air currents. The openings are driven in pairs by double-entry methods, but the separate pairs are connected only as required for operating purposes and, when connected to the general ventilating system, become two airways in parallel.

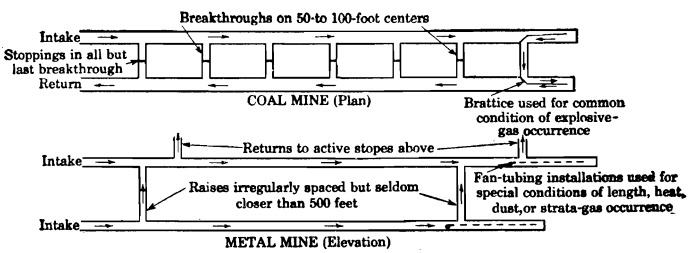

FIGURE 61.—Double-entry methods of advancing into new territory.

The general methods of developing new territory in metal mines are also equivalent to double-entry methods of advancing (lower part fig. 61), but the exceptions are more numerous owing to the irregular occurrence of the ore. Although the details of application differ widely from coal-mine practice the general method that proves most satisfactory as respects ventilation is to drive two or more parallel openings, usually drifts on an inclined vein, 100 to 150 feet apart and to connect them at intervals with a cross-connection, usually a raise driven from a lower level to an upper level.

Occasionally, the parallel openings may be at the same elevation, as in driving long tunnels by the double-entry method to facilitate both haulage and ventilation or in driving a heading parallel to a vein in the rock for haulage coincident with the driving of a drift on the vein.

Explosive gas rarely is encountered at metal-mine faces, and compressed-air exhaust from drills is usually available for auxiliary ventilation and for clearing faces of blasting gases; the cost of driving cross-connections in rock or ore is high, and the disposal of waste rock from development often is inconvenient and costly. For these reasons, there is no point in limiting the length of dead-end faces only insofar as to avoid extra expense for ventilation. The average

distance between cross-connections in development in metal mines is about 500 feet and the average length of dead-end advancing faces about 100 feet longer than the cross-connection spacing. Closer connections usually are required for extraction of ore but usually are driven after development. The positions of cross-connections are largely limited by the occurrence of the ore, as it generally is desirable to drive in ore or to prospect favorable ground. Where there is no other opening to connect to in prospecting new ground or the cost of driving the connection would be very costly openings are often driven as dead ends up to 2,000 or 3,000 feet without artificial ventilation; but, in general, fan-tubing installations are installed for lengths in excess of 1,000 feet, although the necessity for artificial ventilation is always comparative, being governed by the exact conditions that obtain. The common coal-mine practice of ventilating dead ends by bratticing or dividing the opening into a separate intake and return passage by a division wall is rarely applicable to metal-mine openings because the intensity of the blasting concussions would require heavy walls with pressure-release panels. Metal-mine drifts are usually too small and space too much at a premium to even consider this method.

The average metal-mine operator scrutinizes costs closely, particularly the extra costs of driving connections before they are actually needed in the sequence of operations of ore extraction. However, the indirect effects of ventilation conditions on costs of driving and extraction generally are given too little consideration, because they are largely unknown for many combinations of conditions and can only be roughly approximated by experience. Costs of driving as affected by time, on the other hand, generally can be estimated within close limits and naturally are considered more carefully, with the result that many opportunities for reducing final costs by driving connections in time to get maximum use of them with respect to ventilation are neglected. This statement applies especially to the "hot" mines where opening up new territory for free circulation of the ventilating currents as much as a year or two in advance of stoping operations yields large dividends in decreased stoping costs resulting from the cooling of large blocks of ground before extraction. Operators, in general, are converted with difficulty to mining methods involving the free use of raise connections, but once such methods are adopted they are seldom discarded, regardless of changes in mining conditions. On paper the extra costs of such methods loom large, but in practice it is generally found that final costs are lower, without regard to the humanitarian feature of providing more healthful and safe working atmospheres.

USE OF COMPRESSED AIR

The fact that compressed air is used generally as the motive force for operating drilling and other appliances at metal-mine faces is a powerful factor in obtaining better air conditions than would otherwise obtain. It combines both motive force and considerable auxiliary ventilation and for this reason will always be difficult to displace with other types of power for general utility in metal mines.

In the absence of compressed air at a dead-end working face or during periods of nonoperation of appliances using the compressed air diffusion and convection currects only are available for ventilating the face. Under ordinary circumstances their action is very slow but

may permit one-shift operation of short dead ends—not over 300 feet—without serious discomfort if air currents are circulating through the adjacent openings. Diffusion alone is a very slow process, but its action is increased by movement of men and cars and by local convection currents produced by the heat of men, animals, and lights and by the heat liberated by blasting. Under low-temperature conditions the convection currents that act between the dead end and the adjoining airway, although very feeble, discharge blasting gases to the main air circuits and prevent more than a small rise in the carbon dioxide content of the air due to the presence of men, animals, and lights. If there is no circulation in the adjoining airway or connection the air conditions throughout a whole section or mine may become very bad owing to the gradual accumulation of blasting gases diffused into the atmosphere; the air conditions in the dead ends, where diffusion at any particular time is not complete, may become extremely bad.

If compressed air is used intermittently at the face for operating drills conditions at the face, regardless of length, and throughout relatively short dead ends—under 500 feet—are generally good in the absence of high temperatures, strata gases, and dust. They would not be good for some hours following a blast were it not for the fact that it is general practice to blow a relatively large quantity of compressed air close to the face for 15 minutes to an hour or more after a blast. The blowing usually is sufficient to remove the blasting gases from the dead end entirely or to force them out the opening to some distance from the face, where trammers and others pass only occasionally and are exposed to the diffused gases for short periods only. Such exposure seldom produces noticeable physiological effects, although in long headings—over 500 feet—a general feeling of depression or a slight headache may result. In places the muck may be so fine that it traps blasting gases, which are released gradually as mucking proceeds; if no drills are operating at a face that has been blown out after the blast the gases may be present in such harmful concentrations as to cause severe headache. This condition is more pronounced when scrapers and mechanical mucking machines are used on fine muck, because of the more rapid release of gases; mechanical mucking sometimes requires fan-tubing ventilation where hand-mucking would not.

Air conditions at advancing dead-end faces are best when drilling and mucking progress concurrently on the same shift; usually this is the first change made to increase comfort under rising temperature conditions. A drill exhaust averages about 100 c. f. m. and usually is 10° to 20° colder than the surrounding air; it therefore affects comfort at warm rock faces markedly. In addition to producing air motion where it does the most good a drill operating at the face of a small heading (5 by 7 feet) cools the air at warm rock faces usually 3° to 5°; two or more drills operating in a larger heading will give the same result.

There is a tendency, though not among mine operators, to consider the use of compressed air for auxiliary ventilation wasteful under all conditions except that for actual drilling. This, of course, is not true, for although compressed air is costly final cost is a matter of unit cost and a host of other variables, particularly length of time of use and installation costs. With no cost for installation

and short period of use undoubtedly auxiliary ventilation by compressed air is the cheapest form available for a comparatively large range of use. The difficulty is that habits of wasteful use gradually grow in the ordinary mine with attempts to ameliorate discomfort due to increase in air temperatures through increase of rock temperatures accompanying increasing depth. Although compressed air may be used economically for short periods to combat local and temporary discomfort due to air temperature or to unusual gas, smoke, or dust occurrence, its continuous use for such purposes generally is extremely wasteful, costly, and a detriment to the primary purpose of operating drilling and other appliances on account of the low pressures that result.

The effect of compressed air as used to improve comfort at dead-end working faces in warm rock is almostly strictly limited to the air motion it produces, since the temperature of the compressed air at free discharge from a hose is seldom more than 5° less than that of the air under average conditions. The air motion is desirable for improving comfort and the efficiency of the miners but can be more cheaply provided than by direct discharge from a high-pressure pipe or hose. Injector devices that use only a small quantity of compressed air but entrain and give motion to many times this quantity of free air have been developed for producing local air motion in dead ends and in other working places where the quality of the air is sufficiently good. Many mines have tried such devices but, in general, their success has been limited by the general lack of cooperation given to such efforts by the miners, and the devices have been discarded.

LOCAL AIR MOTION

In addition to the injector devices tried at dead-end faces many mines have tried various devices for improving comfort by increasing the air motion locally in warm working places where the general ventilating currents provided air of good quality. Injectors of types similar to those tried in development faces and various designs of compressed-air-operated and electrically operated fans have been used. The objection to most devices has been the miner's complaint about their noise, particularly the shrill and penetrating noise of exceptionally high-speed operation; in general, the devices have met with so little success compared with the discipline required to effect their continued use that they are either disabled or discarded after a short time, regardless of the actual benefits of increased comfort and efficiency in working.

Most miners will stand considerable discomfort before they will exert themselves to alleviate conditions; this fact greatly retards the introduction of devices for auxiliary ventilation, since the supervision and discipline required for the successful operation of a new device—new to that particular mine—are out of all proportion to the benefits to the management, at least in its initial stages. The miner, as a rule, is the one who would derive the greatest benefit but is the last to encourage efforts for improving his own comfort. Turning on a compressed-air hose to induce circulation of air is so simple that it is the common recourse of the miner unless prevented by severe discipline. Where discipline is lacking or ineffective in a warm mine the waste of compressed air by "blowers" is a serious problem to the management

FAN-TUBING METHODS

DEVELOPMENT FOR LOW-TEMPERATURE CONDITIONS

Fan-tubing ventilation methods are very old; in fact, they are mentioned by Agricola (seventeenth century) as a common means of ventilating long tunnels, although the motive force usually was bellows which forced or withdrew air through wooden-box conduits. In time, iron pipes largely replaced the wooden boxes, since wooden-box conduits generally are unsatisfactory as regards leakage and weight, and fans replaced the bellows. With the introduction of compressed air longer dead ends could be driven without recourse to fan-tubing methods, and such methods were limited in actual use to exceptionally long openings, seldom under 1,000 feet, or to strata-gas or high-temperature conditions.

In general, the main consideration under low-temperature conditions is removal of blasting smoke and gases at a point near the face to keep the opening clear of the regularly spaced zones of smoke and fog that develop in long headings ventilated by compressed air only. Exhaust units are commonly used; that is, the fan is placed at the portal end of the tubing and draws air in through the mine opening and out through the tubing. This is a logical installation under low-temperature conditions, particularly where much dust is produced by drilling with dry drills; these, however, have been largely displaced by wet drills during the last 10 years. The general practice of moving the

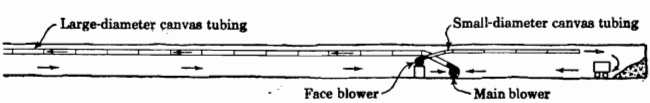

FIGURE 62.—Fan-tubing installations for ventilating long dead-end openings with canvas tubing.

blasting gases from the face to the end of the tubing, which usually is kept 50 to 150 feet from the face to avoid mechanical injury during blasting, is to blow a sufficient amount of compressed air at the face immediately after the blast, the pressure and amount being regulated to give the desired speed of movement. Sometimes the exhauster is shut down after the blasting gases have been removed, but the more general practice is to keep it operating, often at reduced speed, to maintain clear and constant air conditions and to remove the invisible rock dust produced at the face. Where air motion increases the comfort of the miners the direction of air travel is usually reversed, after the blasting gases have been handled.

In headings of large cross-section the amount of compressed air required to move the blasting gases back to the tubing end becomes important; it is often cheaper to reduce the amount required by using a supplementary blower with a short length of small-diameter tubing to bear the brunt of mechanical injury involved in keeping the circulation close to the face. Under moderate temperature conditions this arrangement is especially desirable on account of the air motion produced but is effective only where the tubing end is kept within about 25 feet of the face. Canvas tubing cannot be used with exhausters, but the installation shown in figure 62 is sometimes used; the fan is placed inside the opening and blows toward the portal. At long intervals it is moved closer to the face, and ventilation between the

fan and the face is maintained with a supplementary blower fan of smaller capacity; this is placed just outside the main fan and blows air through canvas tubing of smaller diameter to a point close to the face. After a blast compressed air is blown to force the gases back to the point where they come under the action of the small blower and are carried rapidly back to the inlet of the large blower, which discharges them through its tubing to the portal. Results are not quite as good as those obtained with a main exhausting unit, as leakage results in some recirculation of blasting smoke in the opening outside the main-fan position; this, however, occurs only over a short interval in an inactive zone. Canvas tubing, which may be standard for other fan-tubing installations, is thus used throughout.

DEVELOPMENT FOR HIGH-TEMPERATURE CONDITIONS

The highest development of fan-tubing methods of auxiliary ventilation has been in connection with the high-temperature conditions generally found in deep mines. Here the greatest factor is air movement required for comfort. Although a few mines have attempted to use exhaust units for these conditions, following development under low-temperature conditions, the general choice has been blowing units to obtain the advantage of maximum air movement at the working face.

The first western metal mines to encounter high-temperature conditions were those of the Comstock Lode at Virginia City, Nev.; their experiences in driving dead ends under exceptionally high-temperature conditions probably will remain for a long time to come the world record for mining under difficult conditions. Starting about 1868, these mines used many fan-tubing units comprising positive pressure blowers and iron tubing with bell joints. Eliot Lord [44] mentions the use of canvas tubing, but later accounts mention only iron pipe.

The next large district to encounter difficulties from heat on a large scale was Butte, Mont.; here the use of commercial types of small centrifugal fans to blow air through small sizes of galvanized iron pipe started on a comparatively large scale about 1915. The average size of pipe was gradually increased, and special narrow widths of standard commercial designs of forward-curved-blade centrifugal fans were adapted to this service. The need for a more flexible type of tubing than that afforded by iron pipe under average Butte conditions led to the development of types of canvas tubing in which Butte operators took a leading part. For many years Butte has been the proving ground for new ideas in fans and tubing for fan-tubing installations, although the use of fan-tubing methods has spread to large numbers of metal mines and is making rapid headway in coal mines even against opposition [45] based on the natural dangers inherent in the use of this method under explosive-gas conditions.

For high-temperature conditions the best installation is a non-reversible blower unit, as local cooling by movement of air and rock cooling by volume of flow are required continuously and the presence of diffused blasting gases and smoke in the opening back of the face

[44] Lord, Eliot, Comstock Mining and Miners: U. S. Geol. Survey Mon. 4, 1883, p. 339.

[45] "In the interest of safety, the Bureau of Mines, Department of the Interior, recommends that auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine." (Decision 4, Mine Safety Board.)

is comparatively unimportant. Under most explosive-gas conditions a continuous nonreversible blower installation or its equivalent is also desirable, as the velocity at discharge is required to diffuse the gas rapidly as it is liberated at the advancing face; even a short-time reversal has proved a dangerous expedient where explosive gas is generated, as the end of the tubing never extends entirely to the face and an unventilated pocket is formed in which the gas accumulates during the period when the fan is operated as an exhauster, often with fatal results if any source of ignition is present on the opening traversed by the concentrated gas cloud when blowing is resumed. However, similar gas clouds may be formed by large gas "feeders" back of the face, and unless such flows can be safely diffused by hurdles a continuous nonreversible exhaust installation, with small branches leading to high pockets in the vicinity of the feeders, is safer.

AUXILIARY BOOSTER INSTALLATIONS

The development of methods of using fan-tubing installations to combat particularly difficult high-temperature conditions has led to many interesting types of installations. One developed independently at several mines, where natural conditions prevented or were thought to prevent an increase of main circulations that would effectively

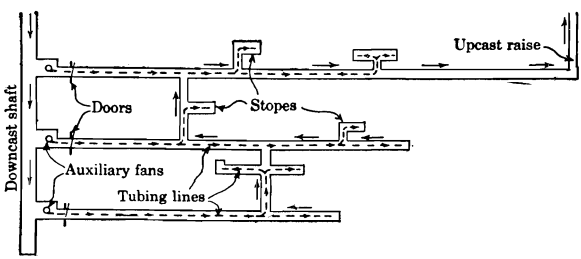

FIGURE 63.—Lay-out of fan-tubing auxiliary units for stope ventilation under high-temperature small-quantity-of-flow conditions.

combat the high-temperature conditions of working places, is shown in figure 63. The main fans are used only to supply air to an intake shaft, which is used as a plenum chamber, from which fan-tubing installations draw air and, discharging through air locks, blow it through tubing to all working faces; thence it moves at extremely low velocities to the return shaft. The costs per unit of quantity circulated are high, and air conditions throughout all parts of the mine except the working faces and the intake shaft—throughout the more inactive parts—are very uncomfortable; air conditions in active zones, however, are much better than they would be if through-circulations were allowed and dead ends were ventilated with fan-tubing units installed at the nearest point of supply, which would be on a warm, slow-moving current. The method is particularly adapted to mining methods involving many dead-end working places in high-temperature rock and their ventilation with a limited supply of air. For a small mine, under certain conditions, the method may be economical, but for a large mine with large ore bodies its final economy is questionable. In many mines the increased cost of operation due to unfavorable working conditions would be more than sufficient to amortize a heavy investment in ventilation openings.

DEVELOPMENT TRENDS AT METAL MINES

The history and development of fan-tubing methods in any particular metal mine follow certain definite trends and encounter the same difficulties. In a warm mine the use of fan-tubing units for auxiliary ventilation generally precedes primary mechanical ventilation by a number of years, such units being used to ventilate local headings or stopes where the need for mechanical assistance to natural circulation first becomes manifest. The original installations are made on a basis of first cost rather than results and usually comprise small fan units and small-diameter tubing, 2.5- to 5.0-hp. fans and 6- to 8-inch tubing; and tubing ends are kept about 150 feet from the face to avoid mechanical injury to the tubing. As the air conditions become more uncomfortable the tubing installations are made with more regard to tightness, and finally larger tubings and fans are employed; but the tubing ends are still not carried close to the faces even though the air conditions there have become quite uncomfortable. Carrying the tubing lines close to the faces involves the dismantling and erection of 50 to 150 feet of tubing every time the face is blasted. If the fan is in operation during the blasting period the damage extends to greater distances, particularly with canvas tubing; the operator has therefore learned to have the fans shut down at this time.

As the air conditions at the face become more uncomfortable the miners can be persuaded to keep the ends of the tubing lines close to the face, within 25 feet at the maximum to benefit appreciably from the air motion, but they rarely take down enough tubing before blasting unless severe disciplinary measures are used; although the expense is reduced by the use of old tubing or smaller-diameter tubing near the end the cost of replacement per foot of advance in hard rock is often high and may easily exceed $1 per foot. Generally, operators go to considerable expense in providing conditions that permit good ventilation at development faces, both to promote comfort and working efficiency, only to have their efforts nullified by the indifference of the miners, who become acclimated to high-temperature conditions and, up to a certain limit, prefer uncomfortable conditions to the labor and bother of keeping the tubing in proper use. The entire benefit of the installation is not lost, of course, if the end of the tubing is not kept close to the face, as blasting gases are rapidly discharged to the main circulation, rock-dust concentrations are diluted, and comfort conditions are improved along the tubing; and convection currents between the discharge and the face act more or less feebly to accomplish the same results on a smaller scale in this more active zone. The rock walls are thus gradually cooled and average temperature conditions improved throughout the dead-end opening. The major benefit of motion of air in promoting comfort and efficiency is lost, however, when the tubing end is not within 25 feet of the face. Operators realize this fact, but the remedy requires service at so many scattered points at about the same time that the delegation of this service to special employees usually is not considered feasible or economical.

When the temperature conditions rise above the equivalent of 90° saturated still air (90° Effective Temperature) the miners can easily be persuaded to make effective use of fan-tubing installations, but without rigid discipline the cost for replacements is always high.

122422°—35——12

FAN SELECTION

The majority of the fans employed for fan-tubing auxiliary service are standard high-speed types developed for industrial use; some are suitable, but many are not. The average conditions of fan-tubing installations in metal mines demand a low-quantity high-pressure fan; a number of types have been developed for this service in the last 10 years. The most recent type and apparently one of the most suitable is the direct-connected, high-speed, backward-curved-blade centrifugal. In addition to light weight through use of high-speed motors these types have a nonoverloading power characteristic and cover a wide range of resistance conditions with little change in efficiency and quantity of flow. Special types of single-propeller fans have similar advantages, in addition to the advantage of axial flow, but are larger for equal pressures. Since they must be used against slowly changing resistances in most cases it is possible that variable-pitch, multiple-propeller types will be developed for this class of service as well as for main-fan and booster service.

In the average installation the leakages that develop at joints and elsewhere make it necessary to use a unit that will handle approximately twice as much air as is required at the end of the line. If resistance factors for average conditions of leakage can be estimated for the actual conditions of service or determined by tests on an average installation the equivalent resistance of the line may be closely calculated by considering the leakage as an airway in parallel with the flow through the tubing from the point of leakage to the point of discharge. The leakage for individual connections may range from 0 to 100 c. f. m. at pressures up to 6 inches of water; where the sections themselves are not tight owing to type of construction, deterioration, or damage the leakage per section, including a connection made underground, may range up to 500 c. f. m.

Virtually all auxiliary fans are driven by direct connection to motors, usually a.-c. induction motors, although d.-c. motors occasionally are used to permit supply from trolley wires of electric haulage. A few compressed-air-operated types are available, but economy demands that they be used only for temporary service in remote places or for emergency service, particularly fire-fighting. Compressed-air injectors are also available or can be made up easily for emergency service.

Auxiliary installations usually have a comparatively short life, a month to a year at any one location; for this reason the fan is often supported with light timber, which presents considerable fire hazard.

In most metal mines the fans are operated singly as blowers. They are seldom used in the exhaust position, except in combination with a face blower on exceptionally long development headings, but are sometimes used in the booster position when a good natural position for the fan is not at the same point as the intake air desired. Where the length of the tubing is too great for a single fan of the types available the fans are used in series with either iron pipe or flexible tubing. A high-pressure fan or a positive pressure blower could be used, but ordinarily the amount of such work is limited and the tubing available is not provided with tight enough joints to give good service under the high-pressure conditions that would result from the use of a single fan. When fans are used in series the maximum pressure differences between tubing and atmosphere are diminished, and leakage is reduced. Difficul-

ties are experienced in the use of flexible tubing in series blower lines, since negative pressures must be avoided. If all the resistances, including leakages, are accurately known the positions of the fans can be calculated accurately. Experience has shown that little trouble will develop if the second fan is placed one third the length of the line from the fan at the inlet. More than two fans are seldom used on canvas lines, but equivalent positions can be selected where more are used.

Selection of an auxiliary fan involves the same principles as for main-fan selection. Speeds, however, usually are limited to the constant speeds of the motors available, which vary for alternating current with the phase and cycles of the current; only a limited number of speeds are therefore available. Careful selection demands that the pressure-quantity characteristics of the fans be available for the available speeds and that characteristics of different types, lengths, and diameters of tubing be plotted to the same scale with the use of equivalent resistance factors that include average leakage effects rather than theoretical resistance factors based only on type, diameter, and length. If the proper material is available selection of a suitable fan for a particular piece of work is merely a matter of graphically determining where characteristics cross.

IRON PIPE VERSUS FLEXIBLE TUBING

Although many different materials may be used for tubing lines and are used for comparatively permanent installations where great length demands maximum conditions of tightness of both tubing and joints flexibility in use is the predominant characteristic for comparatively short and temporary installations used for auxiliary ventilation in metal mines. Canvas and jute tubings, therefore, have largely replaced galvanized-iron pipe for mine installations; and the iron pipe used (actually zinc-coated sheet steel) commonly has stove-pipe joints. Considerable discussion centers on the comparative economies of using iron pipe with wrapped joints or flexible tubing with patented couplings. Both are commercially available or can be made locally in different weights of material. Comparable weights cost about the same per foot for equal diameters; that is, lightweight flexible tubing costs about the same per foot as lightweight pipe of the same diameter for mine service. Because iron pipe is handled more frequently and more roughly in mines it is made of material averaging a gage or two heavier than for commercial work on surface installations. It costs about three times as much as flexible tubing to install and has other disadvantages as respects handling, storage, and ease with which the ends are deformed. Although canvas tubing will last as long as iron pipe under very favorable conditions—freedom from fungus, acid mine water, and other conditions of deterioration against which its chemical treatment may not prove effective—it generally lasts about one third as long and under certain air conditions encountered in mines will last such a short time that its use is almost impossible. Circumstances should decide the most suitable type. For the irregular, crooked openings of the average metal mine the advantages of flexible tubing outweigh its disadvanatges, and it will be found to be the material in common use. For straight and more permanent installations iron pipe appears to have enough advantages to warrant its use; it has a normally lower resistance to flow, in the ratio of about

3 to 5, can be made more nearly airtight than canvas tubing with the couplings now available, and for the same mechanical wear can be kept within about half the distance from the face during blasting as that required for flexible tubing under average hard-rock conditions. Iron pipe is not very satisfactory where many bends are required, not only because the bend is difficult to make but also because the pressure waves from a blast, by causing a vacuum inside the pipe and pressure outside, occasionally flatten the pipe at a bend unless a heavier gage is used than for the straight sections.

NEED OF LIGHTWEIGHT INSULATED TUBING

The most important improvement in fan-tubing practice would be the development of a lightweight, insulated, flexible tubing. The major use of such installations in metal mines would be to combat high-temperature conditions in long dead-end openings. Uninsulated short tubings are very effective for this service, but effectiveness decreases rapidly with length of tubing, as they are in the low-velocity return current from the face, which is heated by the rock and in turn heats the air in the tubing. Larger tubings and quantities of flow obviate this decrease to some extent; but in dead-end development openings over 1,000 feet long the dry-bulb temperature of the air discharged from the tubing is often virtually the same as the rock temperature, and although the wet-bulb temperature at discharge, on account of heating of the air in the tubing without access to moisture, may be considerably lower it generally has relatively little effect except very close to the discharge. The effectiveness of the installation is therefore limited almost strictly to velocity effects. The abstraction of heat from the rock continues, of course, but the relative effect is less because of the large surface exposure, and the effect on face conditions is small because it is spread along the opening. Conditions at the face could be improved by intensifying the abstraction of heat through the use of insulated tubing, for comfort at the face is the objective rather than precooling of the rock; the latter can be accomplished more effectively in the territory where it will serve a useful purpose by large-quantity flow under low pressure after connections have been made.

Actual costs of developing ground in hot rock are not so much a question of cost of fan-tubing methods of ventilation as of the most effective use of double-entry or equivalent methods of opening up ground so as to reduce the length of openings necessarily driven as dead ends.

USE OF BEADING TO STIFFEN IRON PIPE

In connection with the use of iron pipe deformation of the pipe, particularly the ends, always has been a troublesome factor. This is largely being eliminated at progressive mines by a bead or semicircular circumferential projection rolled close to two opposite edges in the separate sheets used in making up the pipe. These have a radius of ¼ to ½ inch and are 2 to 3 inches from the ends; without increasing the weight of the pipe they increase its resistance to deformation enormously. In addition to the cover of tarred cloth ordinarily used to prevent leakage at assembly joints the Magma mine, Superior, Ariz., covers the wrapped joint with a two-piece bolted

clamp which has two corresponding beads properly spaced. This makes an exceptionally rigid and tight line.

INSTALLATION SPECIFICATIONS AND COSTS FOR AVERAGE CONDITIONS

For average conditions of metal-mine installations, where the basic factor is improvement of comfort conditions at working places and consequent increase of working efficiency, suitable equipment may be approximated roughly for 30- to 50-square-foot openings in warm rock (85° to 90°): 8-inch tubing with a fan operated by a 3-hp. motor at a speed to give about 1,000 c. f. m. at 4 to 5 inches pressure for lengths up to 250 feet; 12-inch tubing with a fan operated by a 5-hp. motor at a speed to give about 2,000 c. f. m. at 5 to 6 inches pressure for lengths up to 500 feet; 16-inch tubing with a fan operated by a 10-hp. motor at a speed to give about 3,000 c. f. m. at 6 to 8 inches pressure for lengths up to 1,000 feet; and similar fans in series for lengths much in excess of 1,000 feet. If the tubing lines are in average condition about 40 to 70 percent of the air handled by the fan will be discharged at or near working faces.

For higher-temperature rock and larger openings larger quantities of flow are desirable and can be obtained with larger tubing and larger-capacity fans (quantity rather than pressure, since the benefit of increased pressures would generally be lost through increased leakages). For very high temperature conditions openings large enough to permit motor haulage and mechanical mucking are usually the most economical and are often required for future ventilation needs. Where the size of opening is restricted the use of tubing lines in parallel often solves space difficulties.

Cost of fans, motors, foundations, cable extensions, auxiliary electrical apparatus, and installation labor vary greatly; in general, the total cost of the fan installations mentioned, in place and ready for operation, will range from about $500 for the smaller sizes to $1,000 for the larger sizes. Tubing costs vary with time, quality, weight of material, length of sections (number of couplings), and quantity requirements. An average weight of 12-inch tubing, in 10-foot lengths for iron tubing or 50-foot lengths for flexible tubing, will cost approximately $0.50 per linear foot in small quantities. Costs for other sizes vary about as the material content, that is, roughly as the diameter. For flexible tubing the cost per foot is about 10 percent lower for 100-foot sections and 10 percent higher for 25-foot sections.

The major factor in obtaining satisfactory and economical results with fan-tubing installations is a good crew of specialists for installing, maintaining, and repairing tubing lines. Under average conditions the cost of installing flexible tubing is about one third that of iron tubing. Approximate average figures are about 2 cents a linear foot for flexible tubing and 6 cents a foot for iron pipe.

Costs per cubic foot of air delivered at or near the working place by fan-tubing methods vary widely, but a rough average figure for total cost—capital return, power cost, and maintenance—is about 0.2 cent per 1,000 cubic feet, which compares with rough averages of about 2.0 cents for compressed air released at the face and 0.05 cent for main-fan circulation.

DISTRIBUTION RECORDS

Maps showing the main features of the air-distribution system, as determined at regular intervals, are desirable at all metal mines, both for intelligent planning of the ventilation for normal operation and for guidance in fire emergencies. Where the lay-out and ventilation requirements are simple a simple plan or section map suffices—one showing directions and quantities of flow, position of control devices, and a few temperatures. For more complex lay-outs of openings and ventilation requirements intelligent mapping of the distribution system requires more detail and possibly a more diagrammatic type of map.

Attempts to record ventilation data without the use of symbols usually result in loss of desirable data or unintelligible data. Detailed data for complex ventilating systems ordinarily are recorded on plan maps made to a scale of 100 to 200 feet to the inch. Uniform groups of symbols are desirable, so that those officials who use the maps infrequently are not confused in interpreting them. The set used by the author (fig. 64) has proved practical and useful and provides for all but very special conditions.

For a large mine or one having a complex system of openings some sort of schematic drawing or section may be required to show the main features of the system free from confusing detail, so that broad analyses of conditions may be made simply and easily. Such drawings, however, are seldom made except for purposes of publication, since the average mine official relies on his mental picture of the ramifications of the mine openings. A number of diagrammatic sketches representing very complex systems of mine openings may be found in a recent Bureau publication.[46] For slope workings intersected by vertical shafts simple isometric forms are particularly applicable. For more complex systems of openings more involved isometric forms, such as that shown in figure 65, have a particular field of use. Such maps are to be preferred to separate plan maps or simpler types of diagrammatic sketches, since they permit a complete record of the distribution on one map from which continuous analyses of the system may be made.

Certain observations, such as temperatures, quantities, and fan-performance data, often are made at regular intervals and involve too much detail for recording on the ventilation maps. Loose-leaf notebooks with separate pages for each section where observations are made regularly provide a convenient form of recording observations, although in circumstances where these data are particularly important in analyzing and controlling the distribution it is desirable to transfer the records to a card form which is always accessible on the surface.

VENTILATING SYSTEMS

Any set of connected openings and the forces and appliances that cause flow of air through them may be considered a ventilating system, whether or not the flow is controlled in any way through systematic effort. The theory and practice of separate features of ventilating systems have been discussed in preceding sections; combinations of applications found in actual practice will now be discussed. Virtually

[46] McElroy, G. E., Ventilation of the Large Copper Mines of Arizona: Bull. 330, Bureau of Mines, 1930, 142 pp.

every metal mine differs from every other metal mine in the combinations of openings, pressure sources, and natural ground conditions presented. Few points of resemblance are found in comparing different mines, even where the ore deposits are quite similar; the major points of resemblance are found in comparing details or the separate parts.

The normal procedure at metal mines is to accept the natural flow through the openings made as required for mining operations; this

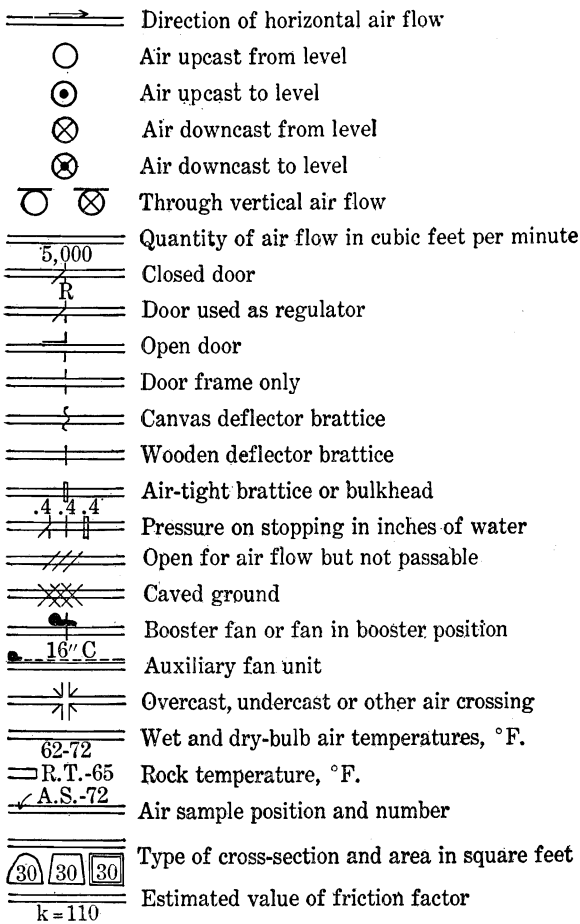

FIGURE 64.—Set of conventional symbols for plotting mine-ventilation data on small-scale level maps.

results in an uncontrolled natural-ventilation system. As the workings increase in extent, particularly with depth, air conditions normally grow progressively worse, and eventually some control of air flow is required; natural-draft currents may be controlled, resulting in a controlled natural-ventilation system, or the flows may be both controlled and augmented by mechanical means, resulting in a mechanical ventilating system.

PROTECTION OF LIFE AND PROPERTY

Uncontrolled systems of ventilation present natural hazards in case of fire; mechanical systems, although they may be installed with particular attention to the fire hazard, naturally increase the normal hazard through increased velocities of flow and added electrical installations. Consideration of ventilating systems is therefore inextricably tied to consideration of protection of men and property in case of fire; while many mines may be considered to have but superficial fire hazards yet there are few large metal mines in the world whose history does not include a disastrous or at least costly underground fire, regardless of the presence or absence of natural hazards, such as sulphides, which are liable to spontaneous combustion on crushing or caving. Moreover, there are few metal mines, either large or small, that have operated more than 10 years and have not experienced a fire in a surface structure which, under special conditions of occurrence, might have imperiled a mine opening.

Ventilation and fire protection should therefore be considered together. The difficulty is, of course, that whereas the one does not necessarily involve expense the other does. Most metal mines have fairly satisfactory air conditions, although where the natural conditions of temperature and vitiation are severe satisfactory circulation will be due largely to increase in use of mechanical installations dating back not more than 2 decades; these will be found to be coping more or less successfully with adverse conditions, according to individual natural conditions and operating personnel.

Recirculation.—The status of metal mines with respect to fire protection is not so satisfactory. Air currents that are satisfactory as regards air conditions will often recirculate to such an extent as to create a veritable firetrap in case of fire underground; the difficulty is that the condition of recirculation is not always manifest and is disclosed only by a survey of the distribution system. Such a condition can exist in mechanically ventilated mines but is more likely to be present in naturally ventilated mines, as the very existence of the mechanical system is evidence that considerable thought has been devoted to the distribution system. The common set of physical conditions responsible for recirculation in workings on inclined deposits is as follows: There are two zones of relatively large and numerous openings, one near the surface and another at depth in the zone of active operations; but between these zones and between the upper zone and the surface the openings are relatively small and few in number. With the natural drafts ordinarily present comparatively large volumes may be circulated in a deep mine, but large proportions of the upcast currents short-circuit through the upper open zone to the downcast airways; the proportion of intake air from the surface is determined by the relative resistances to flow as between the upper open zone and the connections from it to the surface. Where the latter are very small the amount of surface-air intake may be negligible compared to the quantities circulating in the active zone of operations; in case of an underground fire the smoke and gases from the return side of a fire anywhere in the mine soon reach the active zone. As a matter of fact, the warning agents of smoke and odor may be largely removed from the fire gases through condensation in the upper zone before they reach the active zone of operations, and

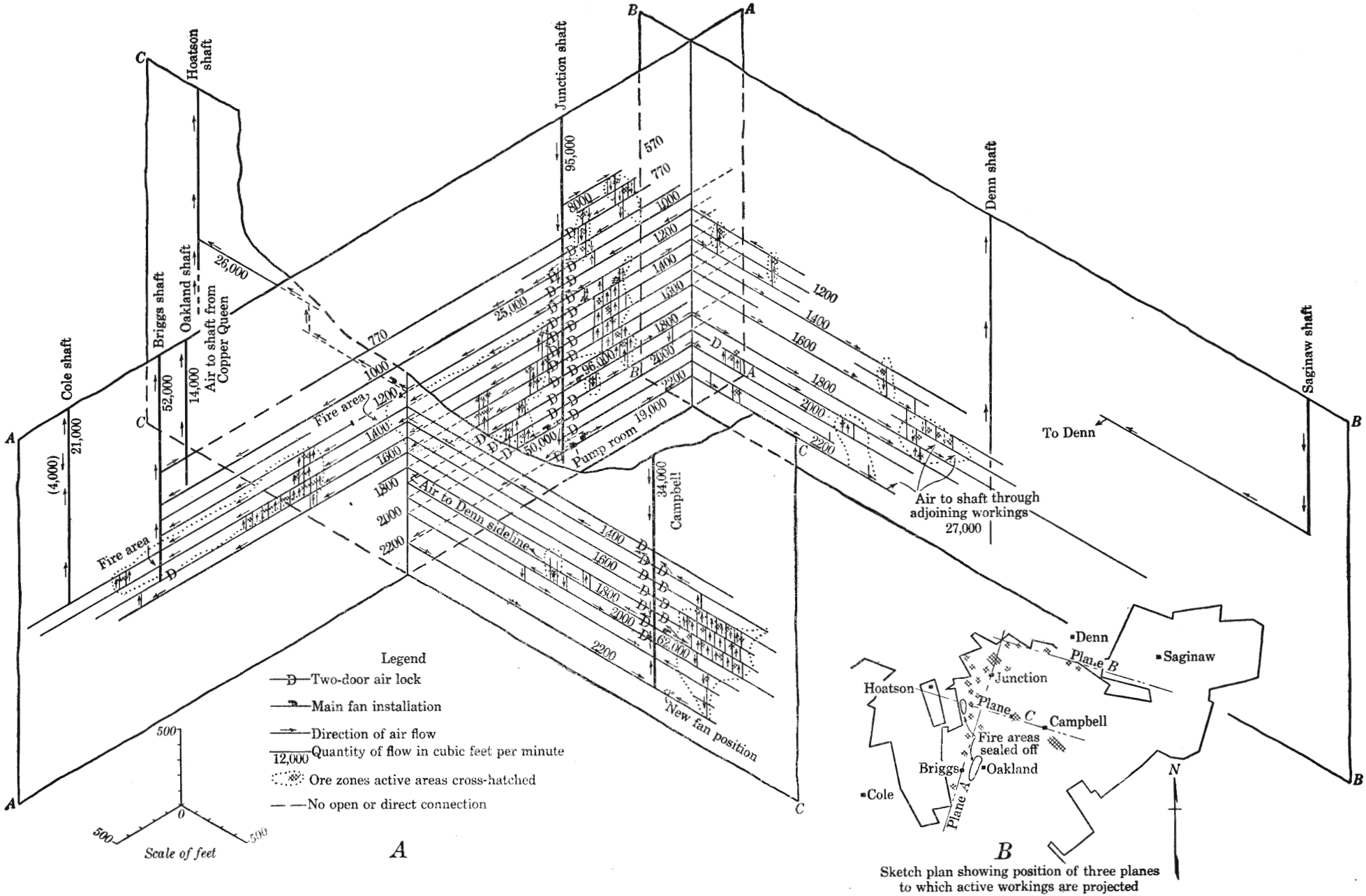

FIGURE 65.—Diagrammatic sketch showing ventilation system of Calumet and Arizona mine, Bisbee, Ariz., October 1929: *A*, Isometric projection of three intersecting vertical planes to which all active openings are projected; *B*, sketch plan showing position of three planes of projection.

the men may be overcome by carbon monoxide before they have any evidence of a fire in the mine. The situation is more dangerous than most metal-mine operators realize, and for this reason all metal mines should have controlled circulation, preferably a mechanical ventilating system. Because of its importance the subject will be stressed further by citing a few examples of recirculation recently found in some large, well-known, and otherwise progressive metal mines, in which the air conditions were so satisfactory that the ordinary account of the mining methods would dismiss ventilation with the statement "The mine is naturally well-ventilated."

Examples of recirculation.—In a large gold mine the total quantity of air leaving the mine was found to be but little larger than the estimated amount of compressed air liberated underground, and no surface intakes were found; yet the circulations underground were relatively large. Investigation revealed that the main-intake airway, a raise from old workings to the surface, had been tightly bulkheaded about 6 months before to prevent "high-grading", and its effect on the ventilation of the mine had not been considered or noticed. The circulation was practically a closed circuit between the lower active zone and a fairly open and abandoned zone near the surface, and only the compressed-air supply maintained the chemical purity of the air against the minimum vitiating agents of men, animals, lights, and blasting at the end of the single shift worked.

In another large gold mine the main-intake and main-return shafts were driven in the same vertical plane, one vertical and the other inclined, and crossed above the active zone of operations. They were entirely open at the intersection, with the result that much of the surface intake air never reached the workings and a large proportion of the return air was being recirculated by natural-draft pressures; some control of natural-draft circulations was attempted in this mine, but only to eliminate fog at points where it interfered with operations.

In a large copper mine the total quantities entering and leaving the mine were found to be large, and only a detailed survey revealed the fact that the total circulations in the extensive active zones of operation were about twice as large as the surface returns. Actually, about half the total returns in upcast shafts were crossing comparatively open zones close to the surface and short-circuiting direct to the downcast shafts.

In another large copper mine mechanically ventilated by underground fans a survey of the distribution had just been begun but was discontinued when a small fire broke out underground. After the fire was under control and all returns but a minor one from the fire zone were stopped off operations were resumed in a section adjacent to the main-downcast shaft. Within a few hours the men complained of headaches and were withdrawn. Several collapsed at the surface, and about 20 showed symptoms of moderately severe carbon monoxide poisoning. The mine maps were then consulted, and leakage paths from the fire zone via caved workings were traced to the main return and from the main return, as door and stopping leakages, to the main intake above the zone of operations. Although the leakage at one door or stopping may be negligible the total for a large number is often a surprisingly large amount, as it was found to be in this particular case.

UNCONTROLLED NATURAL-VENTILATING SYSTEMS

Most small metal mines make not even the simplest systematic effort to control natural flows. These are free, and as long as they provide fair to good air conditions most of the time to most working places they are accepted "as is"; in fact, many operators of naturally ventilated mines, whether small or large, have only the vaguest idea of what causes air to flow in their openings or the reason for dead ends clearing up if given enough time and often have a very incorrect or incomplete knowledge of the paths traveled by air currents. This lack of knowledge often has but little effect on air conditions, as there is a general attempt to make the intensity of operations conform to the average intensity of the ventilating currents, but it does have considerable bearing on what might happen in the event of a fire underground or at a surface opening.

Under favorable natural conditions, particularly where the mining methods involve large and numerous openings, as in open-stope methods of mining, metal mines have been carried to great depths by natural ventilation, both with but more generally without systematic attempts to control circulation; several groups of large mines are still so ventilated. Mines in mountainous country generally are well-ventilated by natural draft down to the horizon of the lowest surface connection and can be well-ventilated for a considerable depth below this level if two openings are available and some attention is paid to distribution; but reversal of direction with the seasons can be prevented only by mechanical installations.

CONTROLLED NATURAL-VENTILATING SYSTEMS

Even where the general system may be considered uncontrolled natural ventilation, separate parts or features may be controlled to some extent. The mining method in use, although not applied with regard to ventilation features, may have been developed in part to insure good ventilation through a process of trial and error. When conditions become exceptionally bad in any district of a shallow mine an opening generally is driven to the surface, in ore if possible, as it has been found that such procedure will in most instances improve circulation. Exceptionally long dead ends or shorter ones in hot rock may be ventilated by auxiliary fan-tubing units.

Often circulation can be both improved and controlled by erecting a few doors and bulkheads, as indicated for a typical small metal-mine lay-out in figure 66. The important consideration with respect to ventilation conditions in large mines is that the workings be divided into sections by natural or artificial barriers extending from the surface to the top of the active zone. The air currents are thus forced to cross from downcast to upcast sections through the active zone of operations, and little recirculation is permitted. Both ventilation and fire-protection conditions are then better than with uncontrolled cross-flow in abandoned openings above the active zone.

Such a system is well-illustrated by the copper mines of Michigan, which are well-ventilated by natural drafts at depths of 3,000 to 6,000 feet vertically and 4,000 to 9,000 feet on the dip below a practically level surface. In this district the deeper mines have reached the point where mechanical ventilation is desirable on account of high-temperature conditions, but some of them are so well-ventilated through

control exercised by natural conditions or by erection of vertical lines of stoppings along shafts for fire protection that reproduction solely by mechanical means would be almost impossible. An example of remarkable ventilation through unintentional control resulting from barrier pillars separating properties is the Ahmeek mine of this group, for which the distribution in the active zone is shown in figure 67. These are retreating open-stope workings with relatively large and numerous openings on a low-grade bedded native-copper deposit

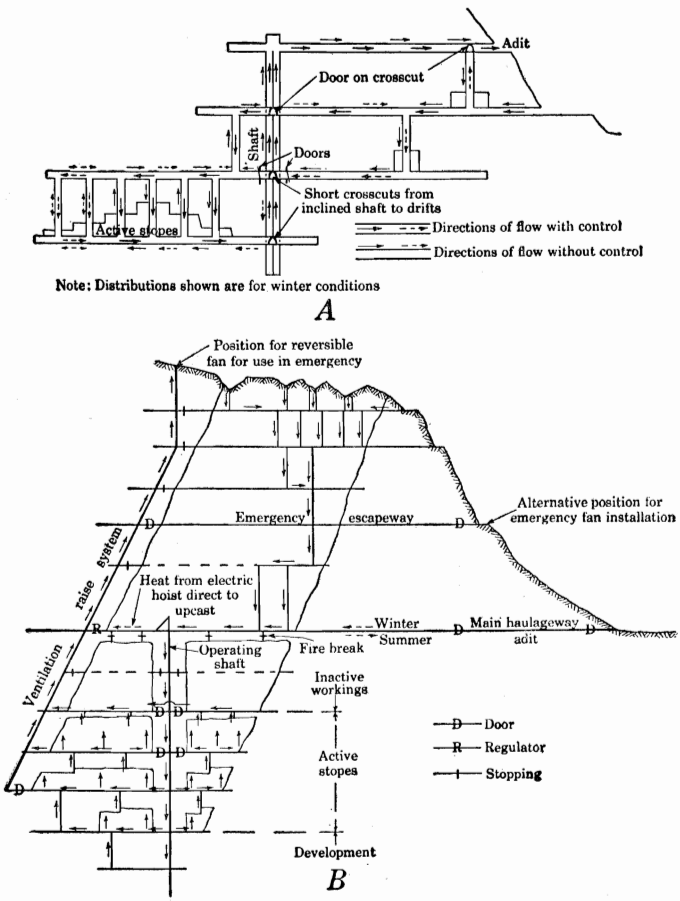

FIGURE 66.—*A*, Controlled natural-draft distribution in a small metal mine; *B*, control of natural-draft distribution to prevent reversal of directions of flow.

dipping about 35°. Details of the mine and ventilation conditions may be found in a separate Bureau publication.[47] The recirculation indicated at the bottom of the wedge of Mohawk ground was due to lack of connections through the lens of poor ground below this point. It was later reduced considerably and the total quantities of flow increased by driving connections through this section at all lower levels.

Examples of good ventilation resulting from intentional control designed for fire protection rather than ventilation are the Osceola

[47] McElroy, G. E., Natural Ventilation in the Michigan Copper Mines: Tech. Paper 516, Bureau of Mines, 1932, 40 pp.

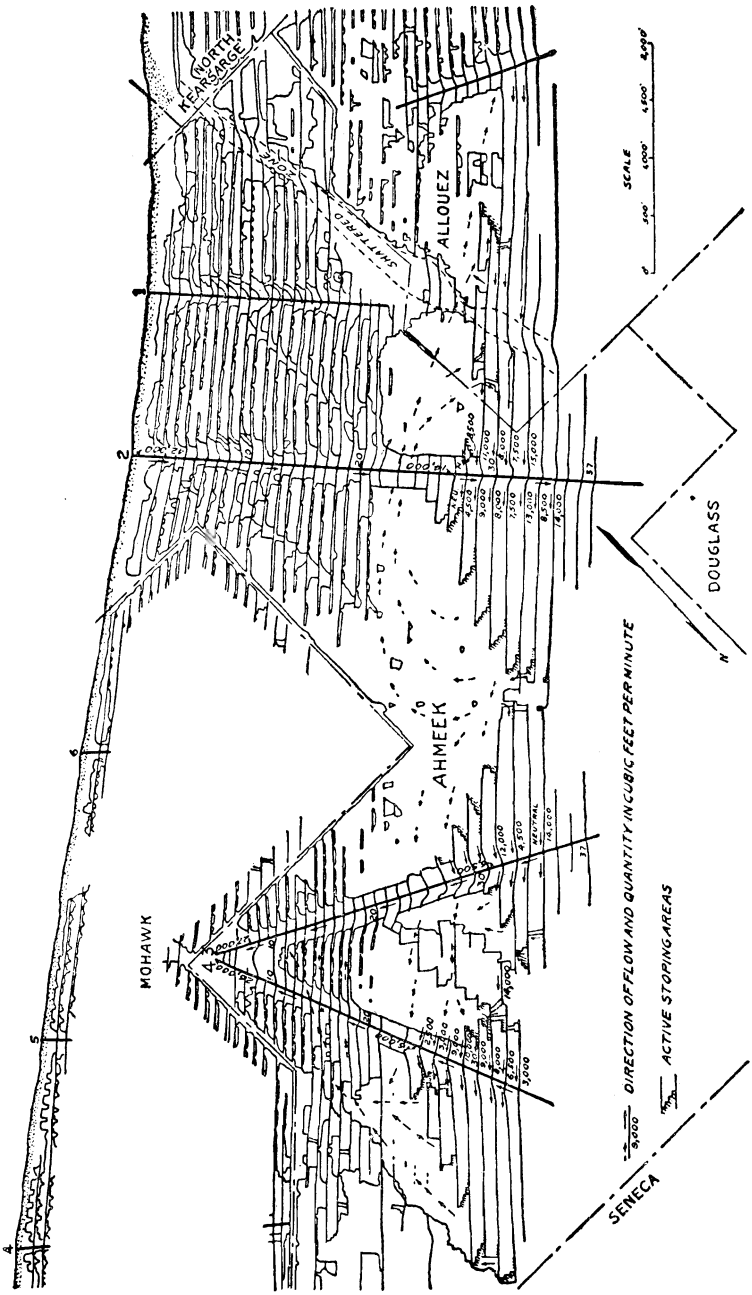

FIGURE 67.—Plan map of Ahmeek mine, Ahmeek, Mich., showing distribution of air currents by natural draft, July 1928.

and Isle Royale mines [47] of the same group working on ore of similar occurrence. Both mines are divided into downcast and upcast sections separated by barriers of lines of bulkheads along slope shafts from the surface to the active zone and by fire doors down through the active zone to the bottom connection. In the Isle Royale the lines of bulkheads and doors are on both sides of the slope shafts, whereas in the Osceola they are on but one side. With such a lay-out the air currents can be practically shut off, as far as shaft flow is concerned, if there is a fire underground. In general, 1 line of stoppings per slope shaft is sufficient for low-temperature conditions, but 2 are desirable on the downcast shafts for high-temperature conditions even though the total quantities of flow are decreased thereby.

The same general scheme of dividing a mine into separate sections can be applied in various lay-outs for naturally ventilated mines; control for fire protection is as complete as with nonreversible mechanical installations. However, the reversible feature provided with mechanical installations is not possible, and quantities of flow, although they may be increased somewhat by attention to temperature and airway conditions, cannot be controlled accurately.

Control of seasonal reversals.—Seasonal reversal of natural-ventilation flows is undesirable as regards ventilation, operation, and safety. Smaller average flows dependable in direction are preferable to the alternate feast-and-famine conditions of uncontrolled reversed flow; with the general abandonment of the use of steam pumps underground flow reversals are more common than formerly. Reversals are reduced considerably by eliminating large differences in surface elevation between inlets and outlets, by coursing intake downcast air at low velocities close to the surface so that surface-air-temperature variations affect only a relatively small part of the downcast air column, and by coursing the upcast air at relatively high velocities to maintain relatively high temperatures in the upcast column. Application of these methods to the general lay-out of openings of a medium-size mine in mountainous country is illustrated in figure 65, *B*. The necessity for the air lock shown on the main haulage adit would depend on circumstances; the resistance of a long adit might be so great compared with the resistance of the downcast raise system that the small inflow during the winter and outflow during the summer would not be a disadvantage. The outer door would be installed in any event for fire protection and in cold climates could be closed to minimize freezing troubles in winter.

MECHANICAL-VENTILATING SYSTEMS

When the results obtained through controlled natural ventilation are inadequate or unsatisfactory for intensive operations the common recourse is to increase the quantities of flow through mechanical installations. These are advisable for virtually all metal mines, regardless of whether they are normally used, to provide the reversible direction-of-flow feature for emergency use and in many instances for use at intervals when surface temperatures limit the natural drafts.

Consideration of mechanical-ventilation systems is nearly always a question of making the system conform with a set of existing air-

[47] McElroy, G. E., Natural Ventilation in the Michigan Copper Mines: Tech. Paper 516, Bureau of Mines, 1932, 40 pp.

ways and their possible extensions. Rarely are the outlines of the deposit known and the lay-out of openings planned in advance. The usual metal mine manages with natural draft until the air conditions get so unsatisfactory that more air and better distribution are required. In most metal mines the primary cause of the change is the increase in air temperatures encountered with depth; in undercut-caving operations it is the necessity for quickly removing the dust, smoke, and gases resulting from continuous blasting of boulders on grizzly levels; at some mines it is the desire for better control of air currents when there is an underground fire; and at a few it is difficulty with strata gases. Minor objectives are: Reduction of the amount of compressed air used for air-motion cooling, reduction of timber decay, and elimination of fog.

In installing a mechanical system in a mine the two main factors to be considered are quantity of flow and lay-out of openings.

QUANTITY REQUIREMENTS

In a natural-ventilation system quantities of flow are taken as they come, or if deficient some attempt is made to increase them. In a mechanical system increase in quantity over natural flows is obtained only at considerable expense, which increases rapidly with increase in quantity whether the latter is obtained by reducing the resistance or by increasing the pressure generated. It is therefore desirable to limit quantities to those that will give satisfactory conditions. In practice, quantity requirements cannot be calculated accurately and are not controlled by law, as in coal mines. Determination of quantities is largely empirical, differs for each mine, and usually is the decision of responsible officials, based on observations under natural-draft conditions or with temporary equipment. Probably the best general statements of requirements are those based on quantity per man underground and the general types of mine conditions. Data on quantities per man underground on the largest shift for a number of large metal mines may be found in table 5 (p. 192).

If air conditions are normal and general blasting is confined to the end of each shift 50 c. f. m. per man ordinarily would be satisfactory for operating concentrated working places two shifts with a long interval between the shifts, but if the working places are scattered widely requirements would approach 100 c. f. m. If operations are concentrated and blasting intermittent, as in undercut-caving operations, a velocity of about 100 c. f. m. is desirable on grizzly drifts for rapid removal of blasting smoke, dust, and gases; general requirements are 200 to 300 c. f. m. per man. Extensive timbering, moderate production of gas, such as usually accompanies work in sulphide ores, or moderately high temperatures would individually increase requirements 50 to 100 c. f. m. per man. Except under extreme high-temperature conditions, the requirements will seldom exceed 500 c. f. m. per man, but under such conditions as much as 1,000 c. f. m. per man might be required for satisfactory results. However, such wide variations in mine conditions and corresponding quantity requirements are possible that considerable play of individual judgment is involved, and the quantities necessary to give a satisfactory air supply are not only approached by trial steps under many conditions but are also subject to a continual though gradual change. They are

also limited to a large extent by the lay-out of openings and the expense of increasing quantities by new airways or modifications of existing airways, so there must also be a continual compromise between expense and the acceptable standard of satisfactory air conditions. In general, the mines that need the most air are those where the ground requires fill methods of mining and where other conditions obtain, such as difficulty of maintaining openings, that result in relatively few, small openings unless the details of the lay-out required by the mining methods are arranged with regard to ventilation to provide more and larger airways. Mining operations usually are facilitated to some extent by having more large openings, and the excess over absolute mining requirements need not be charged entirely to the ventilation system.

PRESSURE REQUIREMENTS

Where the quantities of flow are set at certain approximate figures the pressure requirements to give these flows are determined largely by the lay-out of the openings. If the quantities are constant the paid-for power requirements will vary directly with the excess of the pressure requirements over the natural-draft pressures; it is therefore desirable to maintain high natural-draft pressures. Where the lay-out of openings is constant the pressure requirements vary with the methods of use, and an endeavor should be made to obtain a balance between the resistance conditions of the major parts of the system, such as the main intakes and the main returns, even though multiple-fan systems thereby result. Pressure requirements are fundamentally a question of the resistance to flow of separate sections of the airway system; it is particularly desirable to use all the airways available and, where these would lead to excessively high pressures, either to add to their number or size or otherwise reduce the resistance to flow. Sections of restricted area relative to the rest of the system, or "bottlenecks", would account for an abnormal proportion of the pressure requirements of the system and should be avoided by all means. After the main openings have been fixed the lay-out of minor openings as required by different mining methods may have an important effect on the pressure requirements; these openings, however, usually are so numerous that the pressure requirements are practically determined by the lay-out of the main airways. The exact lay-out of openings in working zones determines more particularly whether or not the air supplied by the system reaches the actual working places.

Fans may be installed at one or more points in the system; one usually is enough for a small mine. In workings that are extensive laterally fans are often used in parallel on separate openings, but in those that are extensive in depth fans are often used in series on or adjacent to the same opening. In very large mines, where the workings are extensive both laterally and in depth, both parallel and series combinations may be found. The variations depend on the variations in lay-out of openings although ventilation by a single main fan is always desirable if it can be arranged economically.

Fan pressures are limited by maximum safe tip speeds to maximums of 5 to 10 inches of water for large centrifugal designs. The actual pressures used for main fans in metal-mine ventilation usually fall

within 3 to 5 inches of water. Higher pressures generally are avoided on account of leakage, difficulty of passage through high-pressure doors or air locks, and possible mechanical troubles at higher fan speeds. Under favorable conditions pressures up to 9 inches have been used. Equivalent resistance factors (p. 114) for main-fan operation at large metal mines rarely exceed 10; the average is about 5—comparable to operation to give 100,000 c. f. m. at 5 inches pressure. Small mines require small quantities, for which the economical sizes of airways are smaller and the equivalent resistance factors might be as high as 20 to 30 for single-fan systems.

DETERMINING FACTORS IN LAY-OUTS

In determining the lay-out or method of mechanical ventilation many factors must be considered, particularly those pertaining to safety, convenience of operation, and efficiency in the use of power, both for the present and the future. Usually the general features of the system can be determined on a nontechnical basis, as maximum use of existing openings with minimum interference with existing modes of operation is always the primary object. Few airways can be driven expressly for ventilation because of their high cost and relatively short life; rather, an attempt should always be made to utilize existing openings required by active mining operations or resulting from past operations and to limit the necessity of maintaining openings for ventilation only to the minimum.

MAJOR OUTLINES OF SYSTEMS

The number, size, condition, use, and position of surface openings, such as shafts and adits, should determine the major outlines of the system. The desired features to be incorporated have been discussed separately and include: (1) Utilization of openings for transportation of men into and out of the mine as intakes or fresh-air outlets; (2) utilization of all available surface openings or enough to give low-resistance systems; (3) coursing of intakes directly to the bottom levels of active zones, whence they ascend through active zones to connections to the main outlets; (4) limiting distances to be traveled to the minimum; (5) maintaining a balanced resistance relation between the main-intake airways and the main-outlet airways and between the separate sections of main airways; (6) reducing obstructions, such as doors and air locks, on active operating openings to the minimum; (7) avoidance of leakage, recirculation, and fog; (8) and circulation of air from active zones to caved ground rather than from caved ground to active workings.

INTERVENTILATION OF MINES

Interventilation of adjoining mines is rarely desirable, as cooperation of separate operating staffs on ventilation matters is not dependable. Where the mines belong to the same organization questions of economy usually dictate interventilation, in which case maximum provision for safety demands that the ventilation of the mines in any interventilated group be under the control of one official. Where the ownership or control of adjoining mines is separate interventilation should be considered only as a temporary expedient, that is, only until the openings necessary for a separate system can be driven.

Isolated sections sometimes can be ventilated conveniently only through a shaft on an adjoining property, in which case it is advisable to enter into a lease for full control of the shaft for the necessary period. Connections between adjoining mines are, of course, desirable as emergency escapeways, particularly for deep mines with hoist service in but one shaft; but such connections normally should be shut off with fire-door air locks which may be fastened, if considered desirable, with seals that can be broken easily in an emergency.

EXHAUST SYSTEMS PREFERABLE

Sometimes natural conditions, such as sealed fire areas or broken ground along the line of an adjacent mine ventilated by a pressure system, seem to favor a pressure system, whereas other factors may demand that the general system be an exhaust system. Such a conflict cannot always be remedied satisfactorily, but often a booster fan underground will supply the pressure conditions required in limited sections, while the major circulation is by the exhaust system.

Most metal mines can be ventilated satisfactorily—if safety is given the prominent position it deserves—by exhaust systems, with the fans on the surface at minor shafts or adits. Air is drawn in through the operating openings (which are assumed to be used also for transporting the men), is kept practically intact until it reaches the lower active levels, and is then allowed to upcast through the active workings to the minor openings through which it is drawn to the surface. Although a bulkhead suffices for a nonoperating shaft and a door or bonnet for a shaft that is used infrequently a surface-fan installation at an operating shaft requires some type of air lock to permit intensive use of the shaft. This condition is common in English coal mines [48] and is solved most effectively by the use of elaborate, automatically operated air-lock installations, which are satisfactory but undoubtedly very expensive. If no inactive opening is available for a main-fan installation the general procedure at American metal mines is to drive a system of raises to the surface or to drive a surface adit from the top of the active zone or from a good connection to this point; this practice by no means has been confined to relatively shallow mines, since raise systems of 1,000 to 2,000 feet are quite common. Similar raise systems are also used to continue openings provided by inactive shafts to lower horizons and often are driven in comparatively shallow mines to avoid driving long connections to existing openings, which, of course, is the logical procedure. Such raise systems usually are more expensive than air-lock systems on shaft tops but prevent operating inconveniences and permit a better selection as to position—features that compensate for the difference in cost. The cost of raise systems through ground that does not require support or maintenance is relatively small where the existing openings provide numerous points of attack, as the cost of driving raises is governed largely by the length of the single sections. A raise system paralleling a main upcast airway is a desirable method of obtaining the equivalent of an increase in area of the main upcast, since the upcast airway at a deep, hot mine may offer very disagreeable working conditions.

[48] Haddock, M. H., Mine Ventilation and Ventilators: London and Philadelphia, 1924, pp. 156–160.

LOCATION OF OPENINGS

To decrease the distance traveled, provide safety, and reduce the chances for leakage the intake and outlet openings, in general, should be so arranged that the active workings lie mainly on the line between them. If active workings extend in two or more directions from a downcast operating shaft an upcast shaft situated near the limit of the active workings in each direction would be desirable. If the shafts are spaced on a line at more or less regular intervals, as in working an extensive vein or lode, alternate shafts should be upcasts and downcasts, respectively, in order to distribute the resistances to flow and eliminate dead areas or those that receive little flow. This arrangement is particularly good where the openings in active zones are comparatively small; where these are large compared with the shafts dead areas rarely develop. Two shafts close together generally are more satisfactory when they carry air in the same direction because extensive stoppings are not required on the connections between them

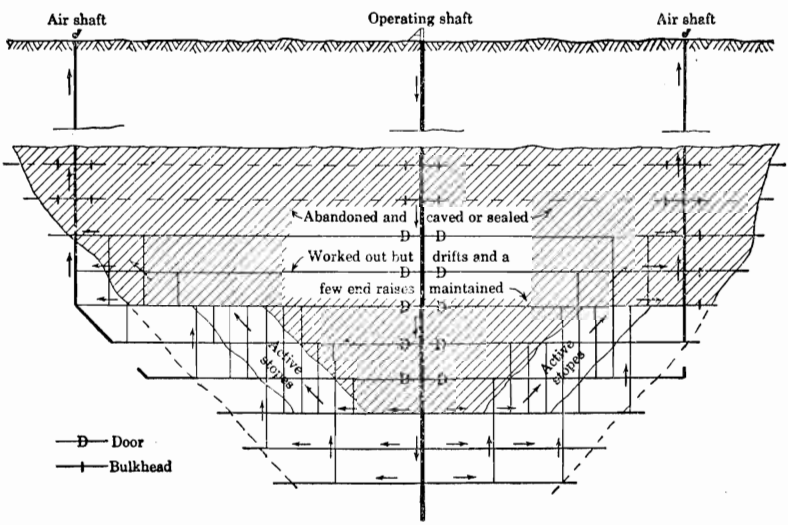

FIGURE 68.—Sketch indicating the generally desirable features of mechanical-ventilation systems applied to metal mines.

leakage is avoided, and the distance traveled by the air is also usually the minimum.

Openings well located and properly selected for use as upcasts or downcasts are a large factor in insuring a simple and effective ventilating system, but the major point is always to use every available opening that can be utilized to advantage in building up the system.

IDEAL LAY-OUT OF AIRWAYS

The lay-outs of metal-mine openings to which mechanical ventilation has been applied usually are so irregular that the main features of the system are obscured, in diagrammatic sketches designed to show the outlines of the system, by the necessity of including important details peculiar to the particular mine. Diagrams of actual systems therefore do not illustrate desirable major features as well as an ideal lay-out, such as is shown in figure 68, assumed for this pur-

pose. This system illustrates the generally desirable features that may be incorporated in most actual systems. Surface exhaust fans with reversible housings installed on outlying and relatively small inactive shafts in the hanging wall of a steeply dipping vein draw air in through the more centrally located and relatively large operating shaft to the bottom levels, through which it passes to active stopes and ascends through them to less active workings connected to the upcast shafts. The eight desirable conditions (see p. 184) previously listed as the objectives for major features are admirably met, and a safe and efficient system should result.

MINOR OUTLINES OF SYSTEMS

The minor outlines of ventilation systems are controlled largely by the individual characteristics that distinguish one mine from another, even though quite similar ore bodies are worked. Some of the more important are: Degree of concentration of operations, as determined by occurrence of ore, changes in economic conditions, and mining methods; position of active areas, as determined by occurrence of ore and sequence of operations; relative ease or difficulty of maintaining openings, as determined by ground conditions and methods of support; position of local sources of heat, as from large electrical installations or crushed sulphide stopes, of cold air, as from large compressed-air-operated appliances, of dust, as from crushers or rotary dumps, or of strata gases, as from fissures in limestone areas; position of sealed fire areas or areas liable to spontaneous ignition; condition of abandoned and inaccessible workings, such as unsealable openings between shafts near the surface and unfilled raises in filled stopes; and the precooling requirements of zones of warm rock. All these and many more special conditions may require certain modifications of the major outlines laid out for the system as a whole, but if the major objectives and their relative importance are kept in mind in planning the lay-out an effective system should result.

CONCENTRATION OF WORKING PLACES

Probably the greatest factor in insuring satisfactory air conditions at low cost is the degree to which active mining operations are concentrated in as small a territory as possible. At times the nature of the ore occurrence and other conditions make it impossible to concentrate operations in a relatively small zone. Concentrated working zones are especially desirable where the operations are in warm rock, as at depth, and good management usually will dictate a policy of cleaning up levels completely so that they can be abandoned in regular order as the depth increases, thus maintaining a uniform depth of active zone. Sections not commercial at the time of temporary abandonment should be sealed thoroughly, as maintenance of support can be preserved both by comlpete sealing and by thorough ventilation. The in-between condition of insufficient ventilation leads to the most rapid deterioration of timber support and the most extensive sloughing of rock walls.

Extensive operations above the active zone of first mining, such as reworking of gobbed areas, leasing operations on scattered pillars and lean zones, and retreating pillar extraction, should have separate intakes and, preferably, separate returns; the circuit should be entirely separated from the main circuit below it by continuous barrier pil-

lars, natural or artificial, as by bulkheading all raise openings in floor pillars or by completely filling all but main openings from one level to the level above. In fairly good ground a shallow depth of fill on timber bulkheads can be used to reduce the expense of reopening the barrier, but in heavy ground raises that are completely filled so as to exclude air completely usually can be opened more cheaply because they require fewer renewals of timber support. Insistence of mill operators on uniform grades of ore output from sections of non-uniform grade of occurrence complicates matters and is a large indirect factor in preventing the concentration of active workings.

LOCAL SOURCES OF HEAT, DUST, OR GAS

The proper handling of local sources of large amounts of heat, dust, or gas and of any other condition that interferes with the normal arrangement of an otherwise desirable system is often troublesome and occasionally very expensive. A separate split of the intake air usually must be diverted direct to an outlet not used for the transportation of men, or an entirely separate air circuit must be formed to include the local condition. For example, in the United Verde mine a circulation of 60,000 c. f. m. was planned (1930) for an underground generator and hoist installation, and in the Village Deep mine a split of 75,000 c. f. m. was allocated to a similar installation. In the Calumet & Hecla Conglomerate mine two large pumping stations on a return shaft each have a circulation of 30,000 c. f. m. At the Ray undercut-caving mine a separate concrete-lined shaft with a surface exhaust fan connects with 3 rotary dumps on the ore-hoisting shaft and handles over 40,000 c. f. m. Here the horizontal crosscuts from dumps to air shaft testify to the necessity of the installation, as they must be shoveled out at short intervals to prevent choking the system. At the somewhat similar Morenci mine, where large cars are dumped in transit over large pockets at the ore shaft, a raise driven from a drainage adit up to the shaft stations provides a separate natural-draft circuit for this shaft, which can be reinforced in case of inadequate or reversed flow by installing a small fan on the drainage adit.

Pump or pipe shafts—shafts devoted solely to pipe columns connected to underground pump stations—are good natural upcasts and should always be used as such if possible. Where required, they usually are close to another shaft, and it is comparatively simple in most cases to make the adjoining shaft a downcast and split off enough air from it to ventilate pump stations and other sources of heat, dust, and gas thoroughly, with the returns all in the pump shaft. Auxiliary fan-pipe installations are an effective means of carrying the return from a local source to a relatively distant return airway.

Where the doors and air locks required for an effective method of ventilating the local source of excessive heat, dust, or gas would interfere to an unreasonable extent with intensive operations the pressure requirements might be supplied economically by a relatively inefficient injector method, such as the "Saccardo" (see p. 134), that does not require obstructions in the airways; this method has been applied to railroad and vehicular tunnels.

According to those who have had extensive experience with such conditions, sealed fire zones are best handled by putting the sealed

area under enough pressure to overcome any reverse natural drafts and providing a small separate split to the main return or a small separate return from a point above the fire zone.

CIRCULATION THROUGH CAVED GROUND

Another difficulty, which appears in many forms, in making ventilation systems effective after they have been carefully planned and installed, is circulation through filled or caved ground. The most common form is leakage through the fill where the shaft collar has been built up above the natural ground level from the waste rock resulting from sinking operations. A concrete lining from the collar to solid rock or to tight material, such as clay, is required for a good job.

In caved ground the circulation depends mainly on the way the ground caves. Even if the proportion of fine material is large enough to prevent circulation through the mass itself, the spaces along the hanging wall and adjacent to pillars or solid ground may be large enough to permit relatively large flows. Enough air can circulate up through 15 to 20 feet of coarsely broken ore to ventilate a shrinkage stope adequately, and enough has been known to pass up through a 200-foot fill of moderately coarse rock in a raise of large cross-section to ventilate a small stope. In the retreating systems of mining employed in the Michigan copper mines air circulates very freely through the caved areas adjacent to active stoping operations, so that such caved zones are equivalent to low-resistance return airways.

Although, in general, it is not desirable to have intake air circulations pass from caved ground (with or without crushed timber) to working places, the degree of undesirability of the method depends on circumstances and seldom is important enough to justify the sacrifice of major objectives or the use of pressure rather than exhaust systems. Where the cover is shallow, holes and fissures serve as good intakes, since oxidation and timber decay are the minimum and the intakes are often in very favorable positions to give short lengths of travel. Where the cover is thick enough to prevent free intake of air the amount intaking is usually so small that even with easily oxidized materials and decaying timber in the caved ground its effect is negligible if the general circulations are of satisfactory volumes. Large circulations enter only through fissures in which the velocities are so high that the time of contact with oxidizing material and decaying timber is too short to affect air conditions seriously.

Under certain combinations of conditions a pressure method of ventilation may appear desirable to insure that the circulations will be from working places to and through caved ground, particularly in top-slicing under a heavy timber mat and 100 to 300 feet of overburden. In certain sections pressure could be maintained in a general system of exhaust ventilation by means of a booster-fan installation. If the whole mine is involved it is possible to use a general system of pressure ventilation and still attain the desirable major objectives listed—particularly that openings used for transporting the men carry intake air. This usually requires entrance through an air lock, a main fan underground on the intake side, or the use of the shaft for transporting men as an intake-air upcast. All these methods usually involve additional inconvenience to mining operations or considerable additional expense over a straight surface exhaust system.

Air circulation through loose material, such as the broken ore in raises, often adds to the problem of distribution, as a shallow depth of coarsely broken ore in a chute has no value as a stopping and makes an uncertain regulator. In many metal mines broken ore in chute raises must be relied upon for controlling the minor features of the distribution but proves very unreliable in practice. The men filling the raises can be relied on for a close record of the material put in the raise, even where the ore runs freely through a grizzly, as in undercut-caving methods; but almost superhuman discipline is required to prevent main-level motor crews from running chutes empty, regardless of orders to the contrary.

PERSONNEL

Ventilation at the ordinary metal mine is generally everybody's business and consequently no one's responsibility. Under favorable natural conditions no great harm results if the lines of responsibility and action are strictly defined when a fire occurs. However, in case of fire the loss of life or property damage may be large through ignorance of the air-distribution system. Under unfavorable conditions ventilation becomes an important factor in operations, and good results are obtained only where lines of authority are strictly drawn. In general, ventilation matters should be confined to three classes of officials—a trained crew to make the actual installations of distribution features, a ventilation foreman to supervise such installations and record the distribution, and a high official with authority to initiate major changes and control the major features. Even though the need for the installation of distribution features is intermittent, it is desirable to have them all installed by the same men, who can thus become proficient, particularly where dimensions, parts, and other features of installations are standardized.

In a small mine the mine foreman or an under official, such as a shift boss, may be required to act as the ventilation foreman, but in a large mine ventilation authority should be independent of the general operating force—the individual vested with this duty should report direct to the ranking operating official. The "ventilation" foreman, so termed here to distinguish him, might devote but a minor part of his time to ventilation and the remainder to other duties, or one man might serve a number of mines where conditions are favorable. Since he has little to do with the major features of the system, except in an advisory capacity, technical ability and a knowledge of ventilation theory, though desirable, are not so important as operating experience, intimate knowledge of the mine workings, and ability to gain the cooperation of the operating officials; but where the natural conditions are such as to make good ventilation a matter of primary importance in operations a high degree of technical ability and knowledge of ventilation theory are required to reduce the load on the higher responsible official.

The requisite technical ability and knowledge of ventilation theory for installing and extending ventilation systems should reside in one higher official selected to have sole responsibility for the ventilation system. In order that ventilation may be fairly represented, the higher the rank of this official the better, since decisions may involve large sums of money. In a moderate-size mine only one man in the organization is eligible—the superintendent or manager. In a large organization one man often is better fitted for the work, through

interest and knowledge of ventilation theory, than the others. Although the chief mining engineer, if there is one in the organization, is the usual selection, the chief electrical engineer or the chief mechanical engineer would often prove a wiser one. For a particularly large mine or group of mines operating under difficult conditions a mining engineer specializing in ventilation would be the natural selection for this position, with or without other duties of minor importance.

COSTS

Ventilation costs depend largely on mine conditions and quantity requirements and will necessarily vary over wide limits for different operations and conditions, regardless of the degree of economy practiced. (See Economic Size of Airways.) The large expenses come at irregular intervals, and as costs ordinarily are charged as they accrue the year-to-year totals for the same mine also are subject to large variations. Moreover, costs are largely determined by the particular bookkeeping methods employed. The ordinary ventilation system usually falls heir to an elaborate airway system already paid for and is charged only with extensions and maintenance of openings used for airways; but certain high-cost jobs, such as a new air shaft, properly chargeable to ventilation often are carried in separate accounts and are never entered or prorated on the ventilation account. Whether or not a proportionate charge is made for compressed air used for ventilation has an all-important effect on the cost data. Few mines make any charge to ventilation for compressed air; most of those that do, use such an unreasonable basis that the figures have no relation to facts.

Compiled cost data for two groups of mines, taken from recent Bureau publications [49] in which they are presented in more detail, are given in table 5.

This table includes average and unit-cost data for the 5-year period 1924–28 and pertinent operating and performance data for approximate conditions existing at the times the mines were visited. With one or two exceptions the cost data include a normal number of high-cost construction items, such as occur at irregular intervals, but contain no prorated charges, such as might be made for compressed air, for maintenance of openings used for a dual purpose including ventilation or for high-cost construction preceding the 5-year period. Few of the items have any real significance, and the significant items serve only to define the probable range that may be met in practice. For these large mines yearly mean total costs show a range of approximately $12,000 to $98,000, equivalent to 0.32 to 18.7 cents per ton of ore mined, 0.02 to 0.27 cent per pound of metal produced, and 0.8 to 11 cents per 100,000 cubic feet of ventilating air. If proportionate charges are made for compressed air and first cost and maintenance of all airways are spread over their estimated life the true costs for ventilation for the mines listed would range up to about double those given for the low-cost mines and up to 50 percent higher for the high-cost mines.

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[49] McElroy, G. E., Ventilation of the Large Copper Mines of Arizona: Bull. 330, Bureau of Mines, 1930, 142 pp.; Mine Ventilation in the Coeur d'Alene Mining District: Inf. Circ. 6382, Bureau of Mines, 1931, 38 pp.

TABLE 5.—*Ventilation cost data for two groups of large metal mines*

Mine	Operating data, 1929				Performance data, 1929					Cost data, 1924–28			
	Ore per day, tons	Metal, percent	Total underground employees	Number employees on largest shift	Total circulation, cubic feet per minute	Cubic feet per minute per man underground	Cubic feet per minute per horsepower, main fans	Horsepower per ton	Ventilating power, percent of mine power	Mean yearly total	Cents per 100,000 cubic feet of air	Mean yearly unit cost: Cents per ton of ore	Mean yearly unit cost: Cents per pound of metal
Morenci	5,000	2.0 Cu	350	235	168,000	715	1,040	0.045	2.0	$17,647	2.5	1.2	0.035
Ray	10,000	1.3 Cu	800	400	162,000	405	1,430	.011	2.7	11,752	.8	.39	.017
Inspiration	23,000	1.1 Cu	1,760	810	165,000	205	1,440	.005	3.2	16,535	2.0	.32	.019
Miami	13,500	.9 Cu	725	470	105,000	225	470	.020	16.6	27,567	4.5	.85	.052
Magma	800	7.0 Cu	400	200	156,000	780	390	.68	23.7	44,601	6.0	18.7	.156
Old Dominion	1,300	2.7 Cu	600	300	180,000	600	900	.32	14.0	58,625	7.9	17.0	.266
Copper Queen:													
Porphyry	2,200	2.1 Cu	295	150	100,000	250	540	.016	30.0	41,018	6.8	6.2	.179
Limestone	850	4.3 Cu	500	250				.24				9.5	.092
Calumet and Arizona.	1,750	5.0 Cu	680	450	140,000	310	470	.26	13.3	60,335	7.8	12.5	.128
United Verde	2,400	4.0 Cu	900	500	350,000	700	780	.13	35	97,899	4.6	11.8	.147
United Verde Extension.	2,200	9.0 Cu	450	330	90,000	270	670	.11	26	33,963	3.7	16.3	.078
Pilares	2,800	2.5 Cu	1,200	600	240,000	400	1,020	.11	35	18,174	1.6	2.3	.044
Bunker Hill & Sullivan.	1,400	10.01 Pb	580	540	40,000	75	500	.06	24	11,656	6	2.6	------
Morning	1,100	9.11 Pb, 6.0 Zn	450	305	40,000	130	570	.12	22	21,040	11	7.4	.022
Hecla	850	10.01 Pb	300	270	70,000	260	470	.19	7	12,865	6	4.0	.023

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